

TN  
1  
A5  
Vol. 187  
NK

# TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

Volume 187

*American Institute of Mining, Metallurgical,  
and Petroleum Engineers. //*

MINING BRANCH

1950

Metal Mining, Minerals Beneficiation, Coal,  
Industrial Minerals, Geology,  
Mineral Industry Education,  
Business Administration,  
Geophysics

---

THIS VOLUME IS MADE UP OF PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD  
AT CHICAGO, FEB. 25-28, 1946; ATLANTA, OCT. 8-11, 1947; NEW YORK, FEB. 15-19,  
1948; EL PASO, OCT. 24-28, 1948; LOS ANGELES, OCT. 15, 1948; WHITE SULPHUR  
SPRINGS, NOV. 3-4, 1948; SAN FRANCISCO, FEB. 13-17, 1949; COLUMBUS,  
SEPT. 25-29, 1949; FRENCH LICK, IND., OCT. 26-27, 1949;  
TAMPA, FLA., NOV. 8-12, 1949; NEW YORK, FEB. 12-16,  
1950; SALT LAKE CITY, SEPT. 1, 1950; ST. LOUIS,  
FEB. 19-22, 1951.

PUBLISHED BY THE INSTITUTE  
AT THE OFFICE OF THE SECRETARY  
29 WEST 39TH STREET  
NEW YORK 18, N. Y.

COPYRIGHT, 1951, BY THE  
AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS  
(INCORPORATED)

PRINTED IN THE UNITED STATES OF AMERICA



## FOREWORD

This, the 187th volume of the Transactions of the American Institute of Mining and Metallurgical Engineers, consists of the technical papers and discussion initially published for the Mining Branch in the Transactions Section of MINING ENGINEERING in 1950. The classification of this material is, by number of papers: Metal mining, 15; mineral beneficiation, 32; coal, 21; industrial minerals, 27; geology, 4; mineral industry education, 1; business administration, 1; geophysics, 2.

Attention is directed to the Technical Note which appears on page 1126. This volume contains only one such form of reporting data or ideas in brief form, but it is used frequently by the Metals Branch to make a preliminary report promptly and economically and is recommended to authors who wish to bring their work to the attention of others before they are ready to write a formal technical paper. The preferable maximum length is 500 words; notes in excess of 1000 words will not be considered for this type of publication. A graph or photograph may be included as part of the note when particularly illustrative.

As pointed out in the Contents page, MINING ENGINEERING was paged consecutively throughout 1950; therefore there are 11 interruptions in the pagination of this volume of the Mining Transactions. The total number of pages is 552 as against 460 for volume 184 (1949), Mining Branch.

To the Program and Auxiliary Publication Committees is due the credit for obtaining and selecting the material in this 187th volume of the Transactions.

The Mining Branch expresses its thanks and appreciation to the authors for the excellence of the papers, which cover a wide range of subject matter of current and permanent interest.

The bylaws of the Mining Branch were adopted at the meeting of the Mining Branch at Salt Lake City, August 29. At that meeting Roy E. O'Brien was elected Western Secretary with headquarters in Salt Lake City. Mr. O'Brien will serve on a part-time basis for the present.

All of the Divisions are moving forward, and that portends a vigorous growth for the Branch.

Many members of the Mining Branch are interested in the 18 papers published for the Extractive Metallurgy Division in 1950. They appear in the Metals Transactions volume 188 and are listed in this volume immediately following the table of contents.

E. R. PRICE, *Chairman*  
Mining Branch Council

## CONTENTS

MINING ENGINEERING was paged consecutively throughout 1950. Thus, in this volume of the Mining Transactions there are 11 interruptions in the pagination. The missing pages appeared in MINING ENGINEERING. This volume consists of all the Mining Transactions for 1950.

### Metal Mining

Sinking Star Shaft at Vanadium, New Mexico. By A. J. MAY .....	81
Practical Dust Control in Metal Mines. By W. C. WILLIAMSON and J. L. SHUGERT .....	86
Problems of a Mine Mechanization Program. By C. KREMER BAIN .....	201
Health and Safety Practices at Pioche. By S. S. ARENTZ .....	204
Faster Calculation of Plane Triangulation Systems by Calculating Machine and Semigraphical Methods. By RICHARD HAMBURGER .....	207
Use of Concrete Underground. By JOSEPH BERNHARDT .....	355
Review of Progress in the Caving of Asbestos Ore. By GERALD SHERMAN .....	467
Development in the Use of Steel for Underground Support. By F. J. HALLER .....	475
A Classification and Application of Drill Jibs for Rock Drill Mounting. By O. J. NESLAGE and R. W. JENKINS .....	571
Development Work with Trackless Equipment. By ELMER A. JONES .....	679
Wrapping Pillars with Old Hoist Rope. By B. T. WYKOFF .....	898
Roof Studies and Mine Structure Stress Analysis, Bureau of Mines Oil Shale Mine, Rifle, Colo. By E. M. SIPPRELLE and H. L. TEICHMAN .....	1031
An Unusual Test of the Accuracy of Well-Surveying Methods. By S. H. WILLISTON .....	1037
Some Applications of Millisecond Delay Electric Blasting Caps. By D. M. MCFARLAND .....	1123
Aspects of Structure and Mineralization Used as Guides in the Development of the Picher Field. By J. LYDEN .....	1251

### Minerals Beneficiation

Measurement of Equilibrium Forces between an Air Bubble and an Attached Solid in Water. By T. M. MORRIS .....	91
Discussion .....	1154
Northern Rhodesia Mufulira Copper Mines Limited Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills. By JACK WHITE .....	96
Discussion .....	1151
Developments in the Application of Activated Carbon to Cyanidation Including the Desorption of Gold and Silver from Carbons. By E. H. CRAETREE, JR., V. W. WINTERS, and T. G. CHAPMAN .....	217
Continuous Countercurrent Decantation Calculations. By T. B. COUNSELMAN .....	223
Discussion .....	1155
Effect of Conditioning on Flotation of Chalcocite. By S. B. TUWINER and S. KORMAN .....	226
Radiotracer Studies on the Interaction of Dithiophosphate with Galena. By G. L. SIMARD, J. CHUPAK, and D. J. SALLEY. (Correction, p. 789) .....	359
Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units. By JAMES J. BEAN .....	365
Discussion .....	1153
Behavior of Mineral Particles in Electrostatic Separation. By SHIOU-CHUAN SUN, J. D. MORGAN, JR., and R. F. WESNER .....	369
Discussion .....	1152
Use of an Induced Nuclear Reaction for the Concentration of Beryl. By A. M. GAUDIN, JOHN DASHER, JAMES H. PANNELL, and WILFRED L. FREYBERGER .....	495
Concerning the Adsorption of Dodecylamine on Quartz. B. A. M. GAUDIN and F. W. BLOECHER, JR. ....	499
Discussion .....	1154
Conditioning and Treatment of Sulphide Flotation Concentrates Preparatory for the Separation of Molybdenite at the Miami Copper Company. By C. H. CURTIS .....	506



# CONTENTS

PAGE

The Burt Filter. By W. G. WOOLF and A. Y. BETHUNE.....	585
Effect of BaCl <sub>2</sub> and Other Activators on Soap Flotation of Quartz. By R. SCHUHMAN, JR. and BRAHM PRAKASH.....	591
Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite. By R. SCHUHMAN, JR. and BRAHM PRAKASH.....	601
Discussion.....	1155
An Electric Tramp Iron Detector for Ore Conveyor Belts. By C. M. MARQUARDT.....	703
Discussion.....	706
Progress Report on Grinding at Tennessee Copper Company. By J. F. MYERS and F. M. LEWIS.....	707
Discussion.....	711, 1151
Relative Wear Rates of Various Diameter Grinding Balls in Production Mills. By D. E. NORQUIST and J. E. MOELLER.....	712
Discussion.....	714
Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries. By S. A. FALCONER.....	790
Discussion.....	1153
Automatic Controls on Sand Pumps. By WILLIAM B. STEPHENSON.....	801
A New Theory of Comminution. By FRED C. BOND and JEN-TUNG WANG.....	871
An Improved Method of Gravity Concentration in the Fine-size Range. By ARVID THUNAE and H. RUSH SPEDDEN.....	879
Discussion.....	1156
The Use of Spiral Classifiers as Ball Mill Feeders. By T. C. KING.....	951
The Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions. By C. G. FINK and G. L. PUTNAM.....	952
Preliminary Report of Massco Circuitron. By A. E. CRAIG, W. J. TAIT, and E. P. MCCURDY.....	1038
Discussion.....	1155
Conductance Electrostatic Separation with Convective Charging. By F. FRAAS and O. C. RALSTON.....	1041
Pilot-Plant Investigation of Concentration of Blackbird Cobalt Ore by Roast-Flotation Process. By S. R. ZIMMERLEY and S. F. RAVITZ.....	1044
The Effect of Mill Speeds on Grinding Costs. By HARLOWE HARDINGE and R. C. FERGUSON.....	1127
Discussion.....	1149
Separation of Precious Metals from Anode Slimes by Flotation. By R. T. HUKKI and V. RUNOLINNA.....	1131
Progress Report on Grinding at Tennessee Copper Company. By J. F. MYERS and F. M. LEWIS.....	1133
The Probability Theory of Wet Ball Milling and Its Application. By E. J. ROBERTS.....	1267
Effects of Rod Mill Feed Size Reduction. By J. J. STROHL and H. J. SCHWELLENBACH.....	1273
Laboratory Studies on Iron Ore Sintering and Testing. By F. M. HAMILTON and H. F. AMEEN.....	1275

## Coal

Cyclone Thickener Applications in the Coal Industry. By M. G. DRIESSEN and H. E. CRINER.....	102
Discussion.....	1158
Quantitative Efficiency of Separation of Coal Cleaning Equipment. By W. W. ANDERSON.....	256
Discussion.....	1162
Some Factors Influencing the Performance of Single Retort Underfeed Stokers. By H. A. BAUMANN, T. S. SPICER, and C. C. WRIGHT.....	265
Research in Coal Geology. By GILBERT H. CADY.....	275
Organizing and Financing Cooperative Research. By ELMER R. KAISER.....	374
Operating Data for a Bird Centrifuge. By A. C. RICHARDSON and ORVILLE R. LYONS.....	381
Discussion.....	1166
Coal Preparation for Synthetic Liquid Fuels. By W. L. CRENTZ, D. DOHERTY, and E. E. DONATH.....	507
Discussion.....	1158
Laboratory Control in Coal Washing and Drying Plants. By JAMES J. MERLE and RICHARD A. MULLINS.....	609
The Mechanism of Coarse Coal and Mineral Froth Flotations. By SHIOU-CHUAN SUN and R. E. ZIMMERMAN.....	616
The Colmol—A Continuous Mining Machine. By C. H. SNYDER.....	715
Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries. By S. A. FALCONER.....	790
Discussion.....	1153
Low-temperature Coke as a Reactive Carbon. By C. E. LESHER.....	805
An Oxidation Method for Investigating the Petrographic Composition of Some Coals. By REYNOLD Q. SHOTTS.....	889
The Cleaning of Fine Sizes of Bituminous Coals by Concentrating Tables. By R. E. ZIMMERMAN.....	956
Specific Safety Problems Applicable in West Virginia Mines. By ARCH J. ALEXANDER.....	967
Air Pollution by Industrial Fumes, Gases, and Dusts. By LOUIS C. MCCABE.....	971
Drying of Fine Coal in the Entrained and Fluidized State. By V. F. PARRY and E. O. WAGNER.....	974
Kerosine Flotation of Bituminous Coal Fines. By L. E. SHIFFMAN.....	1047
Discussion.....	1159
Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal. By M. R. GEER, H. F. YANCEY, C. L. ALLYN, and R. H. ECKHOUSE.....	1057
Discussion.....	1160
Rheolaveur System of Fine Coal Cleaning. By JOHN GRIFFEN.....	1137
Mechanized Cutting and Face Stripping in the Ruhr. By R. R. ESTILL.....	1260

## Industrial Minerals

Chromite and Other Mineral Occurrences in the Tasepe District of Eskisehir, Turkey. By FERID KROMER.....	108
Comparative Furnace Designs for the Expansion of Perlite. By JOHN B. MURDOCK and HERBERT A. STEIN.....	111
Notes on the Geology of the Potash Deposits of Germany, France, and Spain. By J. P. SMITH.....	117
California Talcs. By LAUREN A. WRIGHT.....	122
Ground Water in California. By J. F. POLAND.....	279
Discussion.....	1168
Studies of the Effect of Freight Rates on Marketing Northwest Industrial Minerals. By LESLIE C. RICHARDS.....	285
Fuel Economy in the Lepol Kilns. By R. A. KINZIE, JR.....	289
Industrial Mineral Economics and the Raw Materials Survey. By RAYMOND B. LADOO and C. A. STOKES.....	292
Discussion.....	1169
Production of Graded Glass Sand by Grinding and Classification. By M. M. FINE.....	385
Production of Ammonium Sulphate and Manganese Oxides. By NORMAN KETZLACH.....	391



Some Factors in the Selection and Testing of Concrete Aggregates for Large Structures. By ELLIOT P. REXFORD .....	395
Discussion .....	1168
Recent Developments in the Manufacture of Lightweight Aggregates. By JOHN E. CONLEY and JOHN A. RUPPERT .....	479
Discussion .....	1170
Economic Factors in the Western Phosphate Industry. By ROSCOE E. BELL .....	486
Corrosion Resistant Materials and Coatings in Trail Chemical Operations. By E. A. G. COLLS .....	491
Production and Marketing of Garnet Abrasive Sands from Emerald Creek, Benewah County, Idaho. By JOHN S. CRANDALL .....	575
Nonmetalliferous Mineral Resources in Arkansas. By W. B. MATHER .....	577
Determination of the Coefficient of Linear Thermal Expansion of Rock Specimens by Means of Resistance Wire (SR-4) Strain Gauges. By LOUIS MOYD .....	683
Deposits of Heavy Minerals on the Brazilian Coast. By JOSEPH L. GILLSON .....	685
Kaolin Production and Treatment in the South. By PAUL M. TYLER .....	694
Some Properties of Pseudowavellite from Florida. By W. L. HILL, W. H. ARMIGER, and S. D. GOOCH. (Correction p. 702) .....	699
Effect of Waste Disposal of the Pebble Phosphate Rock Industry in Florida on Condition of Receiving Streams. By RANDOLPH C. SPECHT .....	779
Sillimanite in the Southeast. By KEFTON H. TEAGUE .....	785
Building Stone of the Crab Orchard District, Tennessee. By BENJAMIN GILDERSLEEVE .....	883
Fluoride in Ground Water of Alabama. By PHILLIP E. LA MOREAUX .....	886
An Oxidation Method for Investigating the Petrographic Composition of Some Coals. By R. Q. SHOTTS .....	889
Petalite—A New Commercial Mineral. By J. D. CLARK .....	1068
Special Methods for the Beneficiation of Glass Sand. By PAUL M. TYLER .....	1139

### Geology

Radioactivity at the Caribou Silver Mine, Boulder County, Colorado. By G. CARMAN RIDLAND .....	98
Canadian Deposits of Uranium and Thorium. By W. F. JAMES, A. H. LANG, RICHARD MURPHY, and S. N. KESTEN .....	239
Tin Deposits of the Monserrat Mine, Bolivia. By R. GIBSON and F. S. TURNEAURE .....	1071
Aspects of Structure and Mineralization Used as Guides in the Development of the Picher Mining Field. By J. LYDEN .....	1251

### Mineral Industry Education

The Young Mining Engineer in the Coal Industry. By M. D. COOPER .....	718
---	-----

### Business Administration

Organizing and Financing Cooperative Research. By ELMER R. KAISER .....	374
---	-----

### Geophysics

Radioactivity at the Caribou Silver Mine, Boulder County, Colorado. By G. CARMAN RIDLAND .....	98
Magnetic Fields Associated with Igneous Pipes in the Central Ozarks. By CHARLES R. HOLMES .....	1143

### Technical Note

The Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate. By MAURICE REY, PAUL CHATAIGNON, and VICTOR FORMANEK .....	1126
Discussion of Papers Published in Transactions Volume 184, 1949 .....	1147
Discussion of Papers Published in Transactions Volume 187, 1950 .....	1147

### Extractive Metallurgy

As many members of the Mining Branch are interested in the papers of the Extractive Metallurgy Division, the titles of the papers published for that Division in 1950 are listed below. These papers, and discussion, appear in Transactions (1950) **188**, Metals Branch.

Fundamental and Practical Factors in Ammonia Leaching of Nickel and Cobalt Ores. By M. H. CARON. With discussion.	
Separation of Nickel and Cobalt. By M. H. CARON. With discussion.	
Reflections on the Electrolytic Cells Used in the Production of Aluminum. By BRUNO B. A. LUZZATTO. With discussion.	
The Graphical Representation of Metallurgical Equilibria. By C. J. OSBORN. With discussion.	
Conditioning Dwight-Lloyd Gases to Increase Bag Life. By R. E. SHINKOSKEY.	
An Improved Automatic Smoke Sampler. By J. J. DONOSO.	
Extractive Metallurgy of Aluminum. By R. S. SHERWIN.	
Water Sealed Wind Boxes for Dwight and Lloyd Sintering Machine. By E. McL. TITTMANN and E. A. HASE.	
Acid Conditioning of Metallurgical Smoke for Cottrell Precipitation. By A. L. LABBE.	
The Calbeck Process for Refining Zinc Oxide. By JOHN H. CALBECK, WILLIAM T. MAIDENS, and OSCAR J. HASSEL.	
Treatment of Electrolytic Copper Refinery Slimes. By J. H. SCHLOEN and E. M. ELKIN.	
Modern Baghouse Practice for the Recovery of Metallurgical Fumes. By A. L. LABBE and J. J. DONOSO.	
Thermodynamic Relationships in Chlorine Metallurgy. By HERBERT H. KELLOGG. With discussion.	
A Survey of the Thermodynamics of Copper Smelting. By R. SCHUHMAN, JR.	
Development of Mechanical Puncher at the McGill Smelter. By LEONARD LARSON.	
The Electrical Resistivity of Titanium Slags. By JAMES L. WYATT. With discussion.	
The Development of the Lead Blast Furnace at Port Pirie, South Australia. By L. A. WHITE.	
Production of Malleable Zirconium on a Pilot-Plant Scale. By W. J. KROLL, W. W. STEPHENS, and H. P. HOLMES.	



# Sinking Star Shaft

at Vanadium, New Mexico

by A. J. May

Shaft sinking, usually considered a tough and disagreeable job, has with the development and use of mechanical mucking become a more sought after job by experienced miners. This improvement is no doubt the elimination of exhausting, backbreaking mucking cycles which contributed to the large labor turnover in handmucked shafts. This was apparent in the Star shaft.

THE Star shaft is near the north boundary of the group of mining claims belonging to the Ground Hog Unit of the American Smelting and Refining Co., near Vanadium, N. Mex. The shaft bins and surface plant are served by a siding from the Atchison, Topeka and Santa Fe railroad, and this factor of railroad transportation influenced the choice of site.

This paper describes the shaft sinking only, and does not cover the surface plant or other parts of the mine or its equipment.

A. J. MAY is Engineer in Charge of Shaft Sinking, Ground Hog Unit, American Smelting and Refining Co., Vanadium, New Mexico.

El Paso Meeting, October 1948.

TP 2737 A. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received Nov. 29, 1948; revision received June 22, 1949.

Experience with this shaft seems of interest because of the mechanical mucking, good rate of progress, and the costs obtained at the present level of labor and material costs.

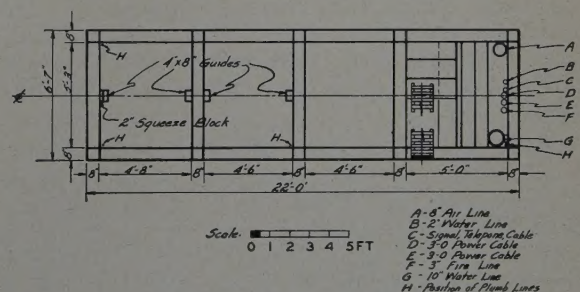
The shaft has four compartments, as shown in fig. 1.

**Equipment:** *Riddell Mucker:* The mucking was done by a clamshell bucket and a Riddell mucker, which, as in other shafts sunk with this machine, consisted of a carriage mounted on axles with double flanged wheels that traveled back and forth on a swinging track, parallel to the long axis of the shaft (fig. 2 to 6). Three air motors were mounted on the carriage; one hoisted the clam, one operated the bucket opening and closing device, and the third moved the carriage along the track. The carriage was designed with guide shoes and could be hoisted to the surface in either skip compartment without trouble (fig. 4). During blasting operations the clam was tied on the rails and the carriage was hoisted up a few sets to protect it from flying rock. No attempt was made to hoist the carriage and clam together lest the clam catch in the timbers.

The track on which the carriage operated was built of extra heavy 6 in. pipe with 1x2 in. strap welded on top for rails. A ½x4 in. angle was welded on the bottom of the 6 in. pipe with a 1 in. round rod welded in the angle for a filler and additional strength. The end or cross pieces of the track which kept the track to gauge, were also made of reinforced, heavy 6 in. pipe and were bolted to the long members on which the carriage traveled (fig. 2 and 3). This section of track was used, instead of heavy railroad rails, for two reasons: (1) this section would bend before it snapped and was much safer for the men in the bottom; and (2) it was a better blasting shield, and protected the timbers better, in fact, only five wall plates were broken by blasting in 1926 ft of shaft, although 150 to 225 lb of 40 pct powder was loaded per round, depending on the ground conditions.

The swinging track was supported by four  $\frac{5}{8}$  in. hoisting cables anchored to the end plates. There were also four safety cables fastened two to three sets above the carriage, which would catch the carriage if one of the supporting cables should fail for any reason (fig. 7).

**Clamshell Buckets:** Two clams were used, each of  $\frac{3}{8}$  yard capacity, both manufactured by Blaw-Knox, Nos. 662 and 663. The No. 663 clam was 4 in. wider and 5 in. shorter than the No. 662, but seemed to dig and load more satisfactorily. It was easier to hoist and lower when it came up for repair. It did not turn over as easily, had less spill



**Fig. 1—Plan of Shaft.**



when loading, and did not jerk the man holding the tag line rope as badly. Both of these clams were equipped with bottom, corner, and side teeth, although the last named were not used (fig. 6 and 7).

The carriage, rails, and clam as shown in fig. 2, 3, 4, and 6 weighed as follows: carriage, 2720 lb; rails with hangers, 3360 lb; clam No. 663, 2430 lb; for a total weight of 8510 lb.

**Sinking Hoist:** The hoist used was made by Wellman-Seaver-Morgan and had the following specifications: drums, 60 in. diam; rope (nonrotating),  $\frac{7}{8}$  in.; rope pull, capacity of hoist, 8500 lb; rope speed, 600 to 636 ft per min; and motor, 2200 v, 225 hp.

The maximum load handled by the hoist in sinking the shaft 1926 ft was as follows: crosshead,

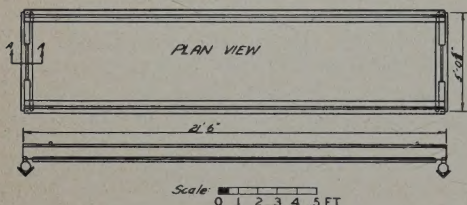
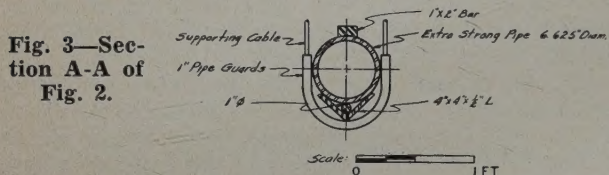


Fig. 2—Side view of track frame for Riddell shaft mucker.



1300 lb; bucket, 1000 lb; load (wet), 3000 lb; rope, 1.32 lb per ft for 1926 ft, 2542 lb; torpedo, clamp, shackles, 300 lb; for a total weight of 8142 lb.

No doubt a hoist with a faster rope speed would have allowed faster sinking, especially the last 600 to 700 ft, as considerable delay occurred in waiting for the bucket to return to the bottom. The mechanical mucker in a shaft is comparable to any other mechanical mucker, that is, to obtain maximum efficiency empty cars or buckets must be kept at the machine with a minimum of delay.

**Crosshead:** The crosshead used was a Joshua Hendy Iron Works type safety crosshead, now manufactured in a custom shop. This type crosshead was designed so that the safety dogs would hold both the crosshead and bucket in case of rope failure. The crosshead also allowed the cable to travel freely when placed on stops in the headframe for dumping, or near the bottom when loading.

**Sinking Buckets:** The sinking buckets were 32 cu ft capacity and 40 in. in diameter (fig. 5). In average muck a bucket was loaded in four to six clam loads, which took from 40 to 60 sec. This allowed the crew from 2 to 3 min waiting for the bucket to come back, thus giving them time to pick down and to level off a place to set the next bucket.

**Ground Conditions and Blasting:** The ground through which the shaft passed consisted of shales, limestones, sills, and dikes, and in places was badly broken. In some sections it was very soft and in others hard. This condition required a change in the number of holes drilled in the round as well as in the location of the holes. In other words, when

the ground was soft, the outside rows of holes were kept inside the timber, and when the ground was hard, the holes were placed farther out to keep down the number of plugs necessary for timber room. In hard ground relief cuts were also used. The standard round was a 42 hole V cut using 10 ft steel in the cuts and 8 ft steel in the back part of the round. If relief cuts were necessary, 6 ft steel was used in them.

The electric blasting caps were from zero delay to No. 8 delay. No instantaneous caps were used. The primers were placed as low as possible in the holes to avoid the clams digging a primer from a missed hole.

In extremely hard ground, insert bits were used which greatly increased the drilling speed over regular detachable bits. The average footage drilled per insert bit, with three or four grindings, was 403 ft.

**Mucking Operations:** It is very important that the track and carriage for the mucker be designed so that the mucker can be operated in the timbers by leaving out the dividers, as was done in this case (fig. 7). This allowed the shaft timbers to be placed near the bottom to take care of bad ground. Care was used when the broken muck piled up above the timbers to insure that the wall plates and end plates were not torn out by the clam.

While the clam worked better in finely broken muck, it also handled large heavy boulders that could not be moved by hand. Breaking large pieces of rock by dropping the clam on them was not good practice because it bent the clam lips so that the teeth would prevent the clam from closing. Then a trip to the surface and shop was necessary to make repairs.

Comparing the old type of sinking by hand mucking with mechanical mucking, the Riddell mucker required more compressed air (at least 600 cfm), more head room, heavier dumping device in the headframe, more rope pull on the hoist, and a faster rope speed. In fact, it would have been economical to place the permanent headframe, hoist, compressor, and other equipment in use during the sinking operations in a permanent setup instead of the temporary setup formerly so commonly used, and used on this shaft.

The clam operator cleaned one end of the shaft of all broken muck which the clam would pick up. The clam usually left from 3 to 5 in. of broken muck, which the shaft crew then picked and mucked by hand into the clam while waiting for the next bucket to return to the bottom. This pro-

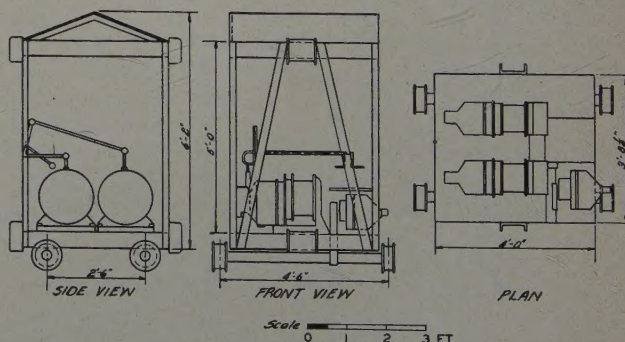


Fig. 4—Mucker carriage.



cedure was followed until the entire length of the shaft had been picked clean, which prepared the bottom for the next drill cycle when the mucking of a round was finished.

In this way only two or three buckets were mucked by hand. The rougher the bottom, the longer it took to clean up. This made the drilling and placing of holes important so that the bottom would have a minimum of bootlegs and break as nearly level as possible.

**Timbering and Plumbing:** The shaft sets were made of 8x8 in. timber in good ground and 10x10 in. timber in weak ground. The grade of material was No. 1, or better, Douglas fir, framed at the mill in Oregon according to drawings submitted by the company. Wall plates, end plates, posts, dividers, blocks and wedges were wolmanized-treated after framing to 0.35 lb net retention of wolman salts per cubic foot of timber.

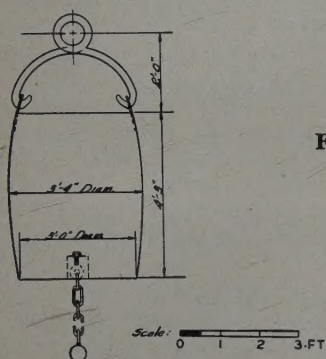


Fig. 5—Sinking bucket, 32 cu ft capacity.

The framing on the first 35 sets received was only fair. However, the framing improved until near the end of the sinking an excellent job was being done.

This class of material, plus the wolmanizing, should give a long life to the shaft timbers, especially in a downcast shaft.

When the framing and treating of timbers at the saw mill before shipment was first discussed, some fear was expressed that the wall plates would warp or shrink. This fear was groundless for it was found by careful handling and piling in the yard that the timber neither warped nor shrank enough to cause difficulty. The timbers were used about as fast as they were delivered which did not allow any great exposure to the sun.

The shaft was plumbed by the engineers every 14 days. This was change day for the shaft crew as they worked 13 days and laid off one. The plumb line brackets were moved down approximately 100 ft each plumbing and were used to establish the line on the next two weeks' sinking operations.

The tolerance was 3/16 of an inch and sets off more than this were relined and reblocked. The 3/16 in. figure was taken because this was as close as the timbers were framed.

Four lines were used in plumbing the shaft: two in the west corners of the west skip compartment; one in the southeast corner of the east skip compartment and one in the southeast corner of the manway compartment (fig. 1).

The line in the east skip compartment was used because of the length of wall plates and to be sure that a bow was not blocked into the 8x8 in. wall plates.

**Ventilation:** The shaft was ventilated by a 15 hp electrically driven fan, located at the collar of the shaft, blowing through a 20 in. vent tube. This delivered 5000 to 7000 cfm, depending on the depth of the shaft. The smoke was cleared out in from 1/2 to 1 hr, depending on the depth.

The compressed air motors operating the mucker at times caused fog in the bottom of the shaft which interfered with the vision of the mucker operator and bell ringer. To overcome this condition the vent tube was lowered close to the bottom of the shaft.

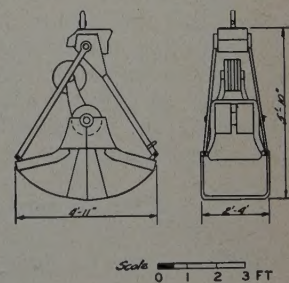
At the 1600 level the shaft connected with the north drift of the Ground Hog mine. From the time this connection was made mine air upcast through the shaft. The fan was then moved from the collar to the 1600 level and mine air was blown to the shaft bottom. Another connection with the mine was made at the 1800 level, and the fan was again moved down and mine air was blown from the 1800 level to the shaft bottom until the shaft was completed.

**Water:** On March 29, a small flow of water, about 9 gpm, was encountered 190 ft below the collar of the shaft. The flow soon decreased to about 3 gpm. A water ring was placed at 210 ft below the collar where an electric pump operated by a float was installed and the water was pumped to the surface.

From 190 ft down, small quantities of water were encountered which were pumped by an air-driven sump pump into the bucket and hoisted to the surface. The water from the drilling operations was handled in the same manner. Up to 20 gpm were handled in this way without delay or trouble. During the mucking cycle the water was pumped into the bucket along with the broken muck, and it was seldom necessary to send up a bucket loaded only with water. During the drilling and timbering cycles, the buckets, which held 225 to 250 gal, were pumped full and hoisted.

Below the 1800 level, 100 gal of water per minute was encountered. In three days this flow dropped to 25 gpm. This water was pumped by tandem-

Fig. 6—Clamshell bucket.



driven air pumps to the 1800 level where it was discharged into the mine pumping system.

Below the 1950 station when the head became too high for the sump pumps, an electrically-driven float-controlled pump was installed on the 1950 station. The water from the bottom was pumped by air-driven sump pumps to the 1950 from which the electric pump lifted it to the 1800 level and into the mine pump system.

**Sinking Cycle:** The sinking cycle varied greatly with the ground conditions. In the better ground where the timber could be kept 20 to 25 ft above the bottom, a complete cycle took about 20 hr and advanced the shaft about 7 ft, which allowed an



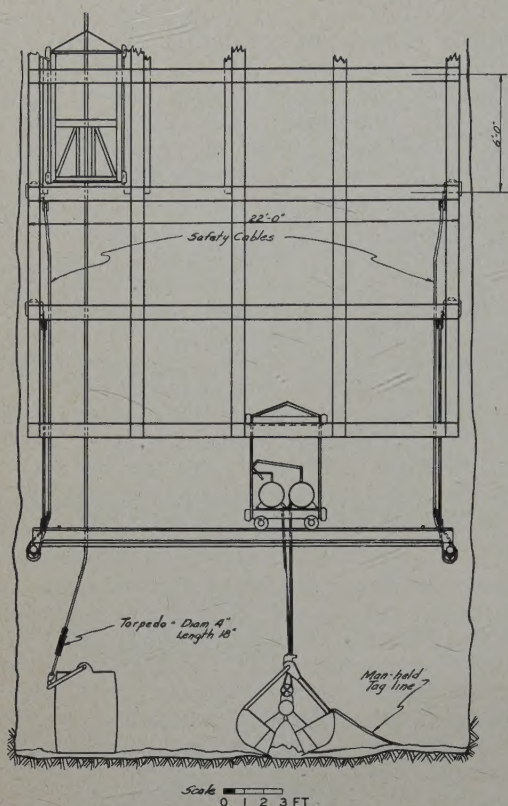


Fig. 7—Mucking operation.

extra set of timber to be picked up every five or six rounds. The cycle was approximately as follows:

	Hour
Drilling .....	5
Loading .....	1
Smoke .....	1
Mucking .....	7
Timbering .....	6

In the section where the ground was badly faulted and the timber was carried as close as 6 ft to the bottom, considerably more time was necessary for timbering to stop any sloughing from the walls and to make it safe for the men in the bottom. Also, the mucking operations were slowed up because of the danger of tearing out timbers when

Table I. Lost Time Accidents

Accident No.	Place	Days Lost	Description
1	Shaft bottom	6	Rock in bucket rolled on finger.
2	Surface	29	Leg cut by power saw.
3	Surface	38	While loading crosshead on truck, crosshead slipped and fell on foot.
4	Surface	40	While unloading 10x10 timbers, winch line slipped and timber fell fracturing foot.
5	Surface	22	Lifted bucket of drill bits into shaft bucket and strained left side.
6	Shaft bottom	13	Rock fell from jacket set, striking hat and knocking head against jackhammer, cutting face and loosening teeth.
7	Shaft bottom	1	Clam snapped shut on rock which splintered and particle lodged in eye.

the broken muck was over the wall plates and end plates.

**Distribution of Shaft Crew:** The shaft crew consisted of the following: one foreman (day shift only), one hoistman, one top man, five shaft men, and one lead man in the bottom in charge of the shift.

The foreman worked all day shift. He was on salary and was subject to call at any time day or night.

The hoist man ran the hoist, started and stopped the compressor and looked after the compressor while in operation.

The top man on the mucking shift operated the dump doors, ran the truck which transported muck from the shaft, and dumped the muck out on the patio. On the drill shift he made up primers, collected the bus wires, connecting wires, and tamping sticks, and had the explosives ready to send down when the drilling was completed. During the timbering operations after the wall plates and end plates were hung in place, he sent down the dividers, posts, blocks, wedges, and lagging in that order. When blowing smoke, he oiled the machines, changed the bits and laid out the proper sets of steel ready to go down when the next drill cycle started.

During loading operations, three men loaded the powder and primers into the holes and three tamped.

When timbering, one man blocked in each of the four corners. A plumb line was hung in the center of the south wall plate, which allowed the fifth and sixth man to line and block the center compartments. The men always drilled and timbered in the same corners. In this way it was possible to supervise blocking and drilling.

The crew was split as follows during mucking operations: one bell ringer, one Riddell operator, one operating the tag line in the bottom, and one leveling off a spot to place the next bucket and scaling down the wall. Two men usually stayed up in the timber working on odd jobs, such as manways, ladders, platforms, and vent lines. If there was no other work for them they went to the bottom and helped to bar down, to clean the bottom, and to get ready to drill.

When possible, timbering was done right after blasting, as this only required the bottom to be cleaned up once for the drill cycle.

The shaft men when possible, ate lunch during the smoke cycle, moving their lunch period forward or back in order not to lose any time. They also moved the wall plates, end plates, posts, and so on, to the collar ready to lower as soon as the smoke cleared.

**Accidents:** Table I gives the lost time accidents showing the number of days lost and location of accidents:

No permanent disability resulted from any of these accidents, except the loss of one tooth in accident number 6. The most severe accidents, as indicated by the time lost, occurred on the surface, and not in the shaft.

Many precautions were taken to avoid accidents.

When an empty bucket was lowered below the collar the hoistman blinked the lights twice signaling the bell ringer that the bucket was on its way



Table II. Sinking Rate

Month	Sinking, Ft	Timber, Ft	Stations	Loading Chutes	Bearer Sets	Concrete Rings	Water Rings	Spill Pocket	Remarks
1948									
Feb.	75	60							
March	112	108							
April	168	164			1		1		Hand mucking Riddell mucker installed 3/3 Larger hoist and larger compressor installed
May	205	212							
June	211	210				1			
July	218	218			1				
August	215	222	1200 ft		1				
Sept.	115	106	1400 ft						
Oct.	170	161		1	1	1			
Nov.	118	128	1600 ft			1			
Dec.	157	160	1800 ft	1	1	1			
1949									
Jan.	115	116	1950 ft	1	1				
Feb.	47	59						1	
Total	1926								Sinking com- pleted Feb. 10

down. If the bell ringer answered, the hoistman slowed up at the crosshead stop, spotted the crosshead, and then lowered directly to the bottom without any further signal. If the bell ringer did not answer the hoistman's signal, the bucket was stopped at the crosshead stop until a signal to lower to the bottom was given.

The bell ringer was stationed on the set directly above the mucker carriage and rang all signals. He had complete view of the mucker operator and the men in the bottom. The bucket had to pass him on the way down.

The bell ringing job is the most unpopular job of the whole operation because of the discomforts of the exposed position and the necessity of being alert at all times.

A good alert bell ringer can keep the accident rate low on the mucking cycle more than all the rest of the crew, as he has a complete view of the ground and all equipment and men involved.

Another reason for the low accident rate is the fact that there has not been a change in the shaft crew personnel since June 1948. This crew as of May 15, 1949 is still intact and will move over to No. 5 shaft, which is to start sinking immediately.

**Sinking Rate:** Table II shows the sinking rate per month along with the stations, loading chutes, bearer sets and concrete rings.

The time required to complete the stations, and so on, when converted to sinking brought the rate of sinking to approximately 207 ft per month, after the Riddell mucker and large hoist were installed.

**Costs:** Table III shows the cost per foot of shaft, excluding costs of shaft stations, permanent air lines, pump column, and power cables.

These costs are presented, not as an example of low costs, but as actual performance in shaft sinking during the year 1948, when labor and material costs have been at a very high level.

Table III. Cost of Shaft-sinking Operations

	Labor <sup>a</sup>	Supplies	Other	Total
<b>Preparatory Work:</b>				
Temporary change house .....	\$ 0.65	\$ 0.62	\$ 0.01	\$ 1.28
(Housing, attendant, heating supplies) .....				
Sinking headframe and bucket dump .....	2.45	0.34	0.07	2.86
Sinking hoists and housing .....	2.56	10.82	0.24	13.62
Sinking pump equipment .....	0.12	0.36	0.01	0.49
Special sinking cages, buckets and equipment .....	1.31	6.32	0.01	7.64
Temporary electric air compressor .....	0.49	0.15	0.01	0.65
Concrete collar .....	0.74	1.30	0.10	2.14
Preparatory work .....	\$ 8.32	\$ 19.91	\$ 0.45	\$ 28.68
<b>Shaft Sinking:</b>				
General .....	\$ 1.82	\$ 3.53	\$ 1.59	\$ 6.94
Drilling and blasting .....	22.41	5.24		27.65
Explosives .....		7.96		7.96
Mucking .....	32.89	1.62	0.07	34.58
Truck and dozer (waste disposal) .....	0.17	0.42	2.12	2.71
Timbering and concreting (includes guides) .....	26.06	56.03		82.09
Framing shaft sets and lagging at mine <sup>b</sup> .....	6.08			6.08
Hanger bolts .....	0.95	3.86	0.01	4.82
Air and water lines in shaft <sup>c</sup> .....	0.90	1.69		2.59
Hoisting .....	12.60	0.91		13.51
Lighting in shaft .....	0.27	0.46		0.73
Ventilation in shaft .....	0.28	1.38		1.66
Power .....	0.07	0.24	3.80	4.11
Signals in shaft .....	0.47	1.28		1.75
Pumping water .....	0.59	0.90		1.49
Supervision .....	9.45	0.02	0.21	9.68
Engineering .....	0.80	0.08	0.03	0.91
General office accounting .....	0.73			0.73
Royalty on Riddell mucking machine .....			3.08	3.08
Shaft sinking .....	\$116.54	\$ 85.62	\$10.91	\$213.07
Total preparatory and sinking .....	\$124.86	\$105.53	\$11.36	\$241.75

<sup>a</sup> Labor at mine. <sup>b</sup> Most shaft sets were purchased, framed and treated, but a few sets were framed at mine. <sup>c</sup> Not permanent air and water lines.



# Practical Dust Control

in Metal Mines

by W. C. Williamson and J. L. Shugert

The paper covers the different phases of dust dissemination in the Butte mines and methods of its control. Dust inspection and reporting same are taken up in detail. Methods of dust and dust counting techniques with their associated problems are discussed. Effects of different methods of dust depression are demonstrated under controlled conditions.

DUST control is receiving considerable attention today by most companies as is evidenced by the many articles, papers, and books written on the subject. Most of the larger mining companies have given this subject close study and have established standardized methods of control.

W. C. WILLIAMSON and J. L. SHUGERT, Members AIME, are Assistant Ventilation Engineers, Anaconda Copper Mining Co., Butte, Mont.

San Francisco Meeting, February 1949.

TP 2705 A. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received Jan. 17, 1949.

**Permissible Limits:** The question of permissible dust limits has been considered by medical committees and various national conferences. Dr. Lanza, of the Metropolitan Life Insurance Company, Dr. Leroy U. Gardner, of the Saranac Laboratory for the study of Tuberculosis, and Dr. Pancoast, of the University of Pennsylvania, have been members of these committees. A statement included in reports of some of these committees is as follows:

Concentrations to which the dust must be induced in order to be safe have not been absolutely determined. There is evidence that a concentration of more

than five million particles per cubic foot of a highly siliceous dust is dangerous. Therefore, it may be stated that it is desirable to avoid counts of more than five million particles of dust containing free silica to a high percent in working places.

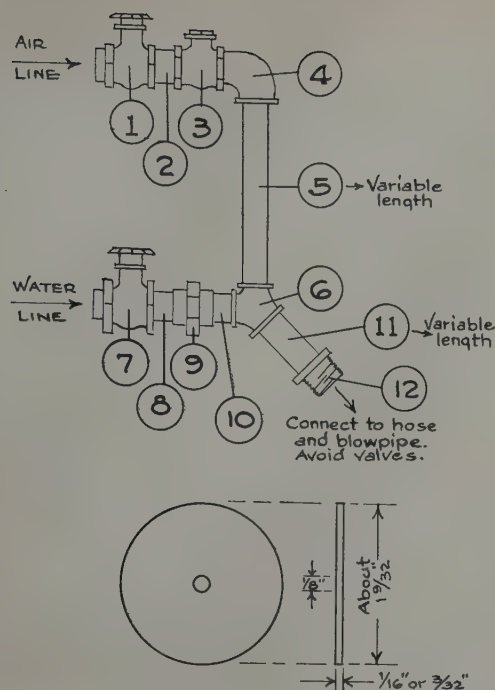
Table I gives the safe limits adopted by several state industrial codes.

Table I. Safe Limits of Dust Concentration

State	Limit Million Particles Per Cu Ft
California	
Silica (less than 10 pct)	50
Silica (10 pct to 50 pct)	10
Silica (more than 50 pct)	5
New York	
Free silica less than 10 pct by weight of rock formation	100
Free silica more than 10 pct by weight of rock formation	10
Oregon	
Silica (over 40 pct)	5
Silica (20 to 40 pct)	10
Silica (10 to 20 pct)	20
Wisconsin	
15 million particles, under 10 microns in longest dimension, per cu ft of air, when the quartz content of the dust is 35 pct, variations in free silica content will make proportional inverse changes in this standard.	

Close attention to dust concentrations has been given to mining operations, since these operations are frequently carried on at considerable distances





### ORIFICE DISC.

Brass with  $\frac{1}{8}$ " hole to be inserted in male half of 1" ground joint union.

Fig. 1—Compressed air and water blow-pipe assembly for use in freeing skip pockets. No. 1-7, 1 in. Mueller valve; No. 2, 8, 10, 11, 1 by 2 in. common pipe nipples; No. 3, 1 in. Crane (disc) check valve; No. 4, 1 in. malleable street ell; No. 5, 1 x 6 in. common pipe nipple; No. 6, 1 in. malleable Y; No. 9, 1 in. ground joint union (gem); No. 12, 1 in. air hose spud-thread variable. Fine thread. Coarse thread Emma-O'Girl-Leonard.

from fresh air sources, and often in areas lacking normal air circulation. Favorable progress has been made in the prevention of silicosis, and with the proper use of modern equipment it is possible today to keep concentrations of dust well within safe limits.

**Main Hoisting Shaft:** Since the main hoisting shaft frequently serves as the source of fresh air for underground workings, it is necessary to maintain a clean, clear atmosphere around the collar. This can be effected by sprinkling and oiling or hard surfacing the mine yard and surrounding areas.

Conditions in the shaft may be responsible for contamination of the main air supply. If the shaft is dry, water sprays at several points should be turned on during hoisting periods to keep the shaft timber damp, thus preventing dust that has settled on the shaft timber from being stirred up by the movement of skips and cages or by spilled rock. Small mist sprays may be used over air takeoffs from the shaft.

Overloading of skips should be minimized to stop spillage, thus avoiding dust resulting directly from falling rock. Measuring pockets with air operated gates helps prevent overloading. Dust caused by loading skips from pockets may be kept down by using sprays on the chute gates to cover the rock as it flows to the skips. Skip loading platforms should be kept clean and should be wet down periodically. If blowpipes are used in freeing chute hangups, water should be introduced into the compressed air. This can be accomplished by connecting the water line to the air line just ahead of the hose connection, and providing a check valve in the air

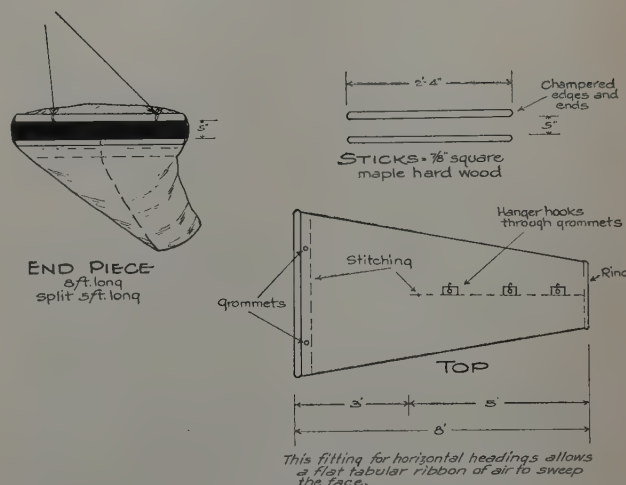


Fig. 2—Fan tube discharge end piece.

Size of Ventube, In.	Length of Sticks, In.	Orifice Opening, Sq Ft	Area of Ventube, Sq In.	Ratio of Area
12	22x5	110	113	1:02
16	30x5	150	201	1:34
21	38x5	190	346	1:82

line and a  $\frac{1}{8}$  in. orifice in the water line to regulate the amount of water introduced (fig. 1).

Shaft stations on the various levels should be open and clean, and should be washed occasionally to prevent any buildup of material which might contaminate the intake air. The top of the skip pocket into which cars are dumped may require water sprays. If waste passes are located on the



stations, the same precautions should be observed as at skip loading pockets. For cleaning car bottoms, a long handled metal blade is preferable to a compressed air blowpipe, but where the blowpipe is used, water should be introduced into the compressed air in the manner previously described.

**Working Places:** Locomotives and trains passing through a haulageway decrease the net cross-sectional area, causing an increase in air velocity. This leads to the lifting of dust from the sill in dry areas.

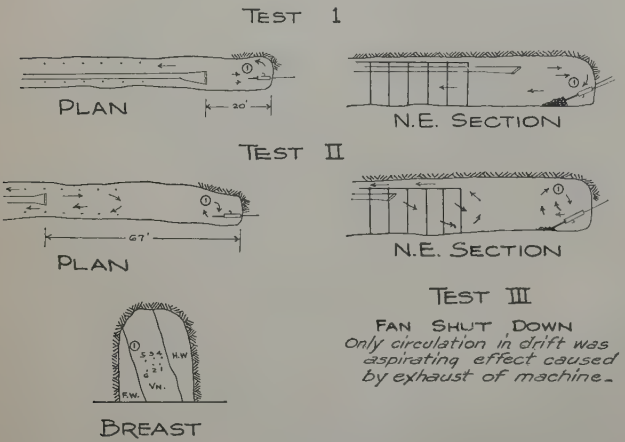


Fig. 3—Air circulation during experimental tests.

To eliminate such action the haulageways should be wet down. Mist sprays, used in higher velocity air flows, and occasional wetting down with a sprinkling hose are both effective. Water connections should be available at convenient intervals. Calcium chloride has been used to maintain a damp condition but is not always satisfactory as it eventually is covered by dust and has to be applied repeatedly to remain efficient.

Opening and closing of ventilation doors also causes temporary high air velocities. The immediate area around a door should be kept damp. Some mines place salt on the sill to hold moisture at such places. Repeated applications are necessary as previously stated.

Temporary dust contamination in air from working places should be traced to its source and the cause ascertained. Most frequent cause for this condition is movement of dry rock. Corrective measures should be applied at once. If dust occurs at raise or stope chutes, where loading causes high dust concentrations, sprays should be used.

**Level Headings:** Headings (crosscuts, drifts and laterals) should be supplied with sufficient ventilation to remove gases and dust from the active work area. If blowers are used to force air to the working face, they should be located in the best available air in the vicinity, which would contain no return air. Air is generally carried to the working place by flexible tubing; it is recommended that the end of the tubing be not more than 25 ft from the face, and as near the back as possible. A special end piece, such as shown in fig. 2, discharges a narrow ribbon, 5x30 in. for 16 in. fan tubing of high velocity air near the back of the heading. The return air will be at the bottom, so that the miner is not

subjected to the stream of air striking his body. This scheme is decidedly advantageous when ventilating air is cold.

Persistent effort is required to secure compliance with these regulations because harmful airborne dust is invisible to the naked eye and, also, because there is little knowledge as to the increase in dust concentrations at the working face that may result from neglect in observing these precautions. The effect of distance from the working face to the end of the ventilation tubing was, therefore, determined experimentally under controlled conditions. Discharge of air from the end of the tubing was made through an oblong opening that caused the air to flow in a flat, tabular stream that struck the face near the back and swept drilling dust downwards and out along the floor line (fig. 3). Table II presents the results of an experimental test in a drift.

Table II. Results of Experimental Dust Tests in a Drift

Distance from Breast to End of Flexible Tubing, Ft	Volume of Air for Ventilation, Cu Ft per Min Through Tubing	Depth of Hole Drilled, Ft	Dust Concentration, Millions per Cu Ft of Air	Remarks
20	2,600		0.29	No activity Drilling. Much chuck splash.
20	2,600	3.8	7.93	
20	2,600	3.9	8.79	Drilling. Much chuck splash.
67	2,300	4.2	24.90	Drilling. Air circulating in closed swirls. Misty.
67	2,300	4.3	29.75	Drilling. Air circulating in closed swirls. Misty.
67	No ventilation	4.5	23.08	Drilling. Air circulating in closed swirls. Misty.
67	No ventilation	4.5	32.06	Drilling. Air circulating in closed swirls. Foggy.

The following notes describe conditions under which the above test was made:

Hard ground. Good supply of drill water. Automatically rotated drill. All holes were collared before the tests and were upward cut holes drilled at angle of about 30° above horizontal.

For effect of distance from working face of end of ventilation tubing during drilling, see fig. 3.

All dust samples were taken at the same place, at the breathing zone of man drilling. The average for the 20 ft distance is 8.36 millions, for the 67 ft distance 27.33 millions, and with no ventilation 27.87 millions. It is, therefore, evident that when the end of the tubing is allowed to remain at a distance of 67 ft from the breast, dust conditions are no better than they would be if no ventilation equipment were supplied, and are above safe limit.

In this connection, it is also interesting to note the work of G. E. McElroy, of the U. S. Bureau of Mines, who has shown that no matter how much the volume of air discharged from a ventilation tube is increased, the distance at which it is effective in creating air motion in a drift remains practically constant.

After establishing ventilation, the working area including face, back, sides, and muck pile, should be wet down to prevent dust rising from the barring down and mucking operations. If dry sections are exposed in the course of mucking, these should be wetted again. A dry working place should be wet down before timbering.

Although nearly all drilling today is done with wet drills, it is not generally realized that use of an



insufficient amount of water can result in high dust concentrations. It is usually advisable to use as much water as can be put through the machine and drill steel. Manufacturers of rock drills increased the size of water tubes used in these machines to help combat dusty conditions. Collaring holes dry should be discouraged, as this practice introduces a dust hazard of the highest order.

Blasting should be done under cover of a compressed air and water blast, sometimes called a "fogger." This is turned on after spitting the round, and shut off immediately after the blast. It should be located within 35 ft of the face, close enough to cover the blast location with mist. The effect of this device in freshening the air is clearly shown by fig. 4 which presents test data obtained under similar conditions with and without the use of the air and water blast. The average of four dust samples taken during the period indicated in the curves showed a 99.6 pct decrease in dust concentration when the air and water blast was used.

**Raise Headings:** Raises are also dead ends and should be treated the same as sill headings. Each raise should have its individual blower, of the same capacity as that used for sill headings, and located with the same care.

Wetting down the working area and the manway of a raise should be the first operation after a blast or at the beginning of the shift, if this can be done with safety. Barring and cleaning down which follow are then not apt to raise dust.

At the time of drilling, the vent tubing should be brought up so as to introduce fresh air above the drill staging. This insures a clean sweep of the back and allows exhaust with little recirculation.

The use of a compressed air and water blast in a raise, vastly improves conditions for the workmen. The device converted for this purpose, as shown in fig. 5, will freshen the air in a raise so that a miner can return to the place with safety 15 min after a blast. It can be used in either of two ways, spiked in place to jet upward toward the back, or spiked to jet downward into the chute if the back is bad.

**Stopes:** Stopes with facilities for through ventilation should receive as careful attention as places requiring auxiliary ventilation. Care should be taken that manways to levels above and below the active mining area are open sufficiently to allow free circulation. The movement of air through manways can be facilitated by the use of grates rather than solid landings. Wetting down these air courses is especially important when they are located alongside working chutes and this may require use of fine sprays. Vibration, set up by the action of rock passing through a chute, can cause dust that has accumulated on timber to be picked up by the air stream unless the manway is kept damp.

Material being slushed, mucked, or cleaned down in a stope should be kept damp enough so that no dust is visible in the air. At most mines today, wet stope filling is used because it has been found that wet fill consolidates better than dry. Where dry waste is used, it is essential to provide a water hose at the point where waste is dumped into the fill raise. Cars can also be cleaned with this hose.

Stopes not having through circulation of air should be provided with auxiliary ventilation as in any dead end working place, and the same precau-

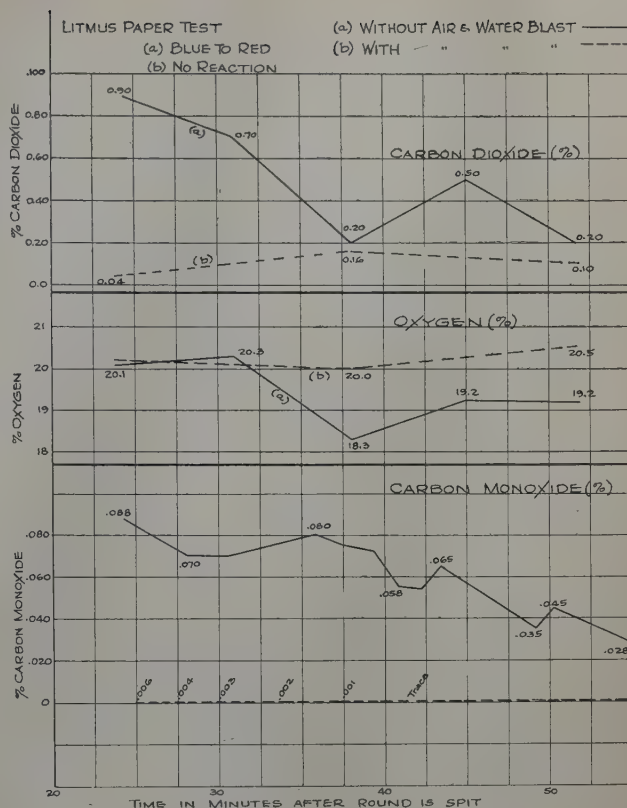


Fig. 4—Environment after blast in heading.

tions should be taken in the various operations, including use of a compressed air and water blast during blasting.

Repairmen should keep their working places wet down just as thoroughly as any other, in order to prevent contamination of air by dust.

**Standardization:** Where a large group of mines is operated under one management, as in the Butte district, it is both practical and desirable to standardize procedures for dust control. Miners move from one mine to another, and, having become familiar with the practice at one mine, do not have to learn new practices elsewhere in the district. Most of the equipment has been standardized, including fans, fan tubing, air and water blasts, and sprays. Methods of installing and using these have also been standardized as far as practicable.

Differences in temperature may require differences in the application of ventilation equipment. For instance, a man working in a warm place will be more likely to place the fan tubing in proper position for an effective "sweep" of air over the working face than a man in a cold place. The man in the cold place is prone to leave the ventube at an ineffective distance from the face. This tendency can be overcome by using less air and introducing it close enough to the work for a definite circulation of fresh air at the face. Some mines in high altitudes and cold climates find it necessary to heat the air used for ventilation purposes.

**Dust Survey Methods:** In the Butte district, ventilation engineers usually make routine inspection trips in company with the shift bosses. Such trips



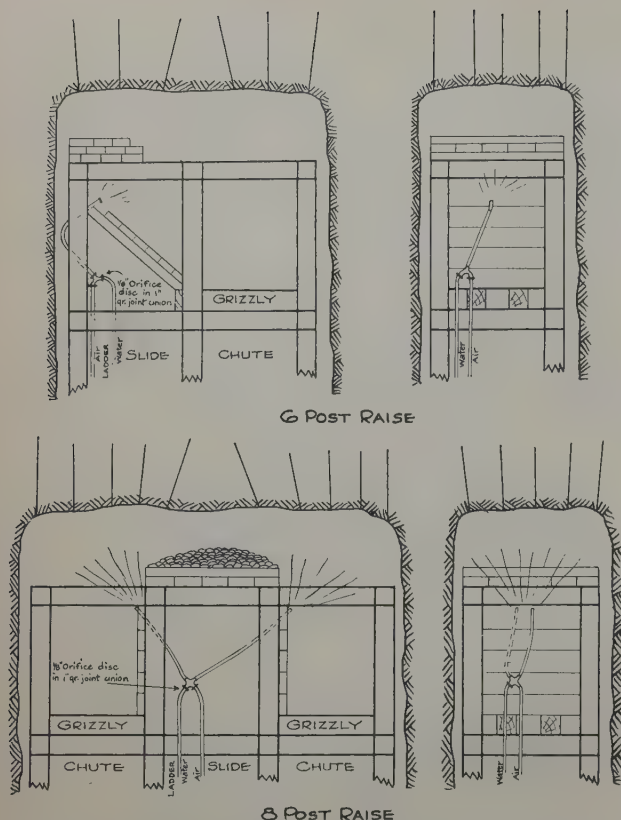


Fig. 5—Raise compressed air and water blasts.

serve to acquaint the inspectors with the working problems and conditions. The areas requiring further attention are readily recognizable and a spot check or a return visit can be made as desired. It is well to leave a daily report of findings with the mine foreman for his information. The time between inspections should not be allowed to exceed two months.

The midget type impingers used for sampling have U. S. Bureau of Mines approval. The constant volume type pump is gaining in favor over the constant pressure type. They combine ease of carrying with speed of setting up. It also makes it possible to obtain a sample representing any desired period of time which may cover several high or low peaks of dust concentration. A measure of the average exposure is obtained with this instrument by an un-weighted sample.

By the use of some of the other instruments such as the Konimeter, the sample is not taken throughout a period of time but may record only a high or low peak of dust concentration in one sample. This necessitates many samples for the same result obtained with one midget impinger sample.

The sampling equipment should be kept in good order to avoid errors from this source. The pump should be calibrated frequently. Also, as may be seen from the paper\* by Richardson, Warren and

\* Presented before National Safety Council Meeting, Chicago, Oct. 1945. "Some Sources of Error in Dust Sampling Work Done by the Impinger Methods."

Williamson, it is well to number and calibrate the individual impinger tubes.

This paper, also, calls attention to the importance of cleaning the glassware thoroughly, especially the tubes which are difficult to clean.

Another source of error in sampling is demonstrated by test on sampling in high velocity air. From this, it can be seen that when velocities exceed 750 ft per thousand, an elbow is necessary to bring the tube in line with direction of flow.

U. S. Bureau of Mines approved practices of particle counting are followed. Counting cells should be accurate in depth of one mm, and settling time of all cells should be practically identical.\*

\* Presented before National Safety Council Meeting, Chicago, Oct. 1945.

The counts with necessary remarks are reported to the mine operators and management with recommendations for corrective measures.

For thorough coverage, a general dust survey (as contrasted with spot coverage) should be made at less frequent intervals. In this survey, every working place should be sampled, covering every phase of the operation. By compiling the results of this sampling, a figure representing average dust concentration or worker exposure may be obtained. If this dust concentration is appreciably below the safe limit as determined by the U. S. Public Health Service for the type of dust encountered, the dust control measures existing in that operation may be considered adequate.

**General:** There is considerable variance in the opinions of experts as to the importance of the different phases of dust which causes silicosis. Some feel that, if dust contains no silica, it is not harmful no matter what the size or concentration.

Others feel that, if the dust is not small enough to get in the alveoli of the lungs, it cannot cause silicosis no matter what the composition or concentration may be. At present, there seems to be some agreement as to size that is harmful which is anything smaller than 3 microns.

Still others feel that no matter what the size or composition may be, the dust is not harmful if the concentration is low enough. Since there is no known practical method of controlling particle size or composition of dust in metal mines, it is evident that all efforts must be directed towards keeping the dust concentration below the established safe limit.

In the above discussion it has been shown that there are three particularly effective weapons for keeping down dust concentrations; ample ventilation, wetting down, and compressed air and water blasts.

In experimental tests where the amount of ventilation has been increased, it has been established that the effect is nearly proportional, that is, if a volume of dusty air is diluted by an equal volume of clean air, the dust concentration of the combined air is only slightly higher than one half that of the original dust laden air.

Thus the role of ventilation in reducing dust concentration is of major importance.

The human factor must also be given the attention that it deserves, as cooperation between workmen and management is necessary in order to apply an effective program for dust control.

In conclusion, it may be said that if a proper dust control program is put in force and administered by responsible men with full cooperation of the workmen concerned, no harmful dust condition should exist.



# Measurement of Equilibrium Forces between an Air Bubble and an Attached Solid in Water

by T. M. Morris

The forces acting between a small rod, one end of which was made water repellent, adhering to a much larger air bubble in water were measured. An equation is deduced which correlates these forces and the influence of each force is discussed. The importance of the size of bubbles in a flotation cell is emphasized.

A SEARCH of the literature reveals that no measurements have been made of the forces acting between a small solid particle whose surface is hydrophobic, and an air bubble to which the solid adheres, both immersed in water. Analyses have been made of the forces acting to support a greased solid on the surface of water, and the forces acting to cause a solid, whose surface is hydrophobic, to adhere to an air bubble in water. The latter analysis

T. M. MORRIS, Junior Member AIME, is in the Department of Metallurgical Engineering and Mineral Dressing, School of Mines and Metallurgy, Rolla, Mo. New York Meeting, February 1950.

TP 2734 B. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received May 16, 1949.

This paper is the result of work done for part of a doctor's thesis at the Missouri School of Mines and Metallurgy.

is often incomplete, however, because the internal pressure of the bubble has been neglected. It will be demonstrated that the internal gas pressure is not a negligible factor when dealing with bubbles of the size encountered in flotation.

A study of the forces acting between an air bubble attached to a large flat surface is informative. It must be borne in mind, however, that this is not the condition present in a flotation cell, where the particle is small compared to the size of the bubble. The bubble is allowed to spread to its maximum contact angle on a large flat surface in the first case, whereas in the second case, the spread of the bubble is limited to the surface of the small particle which is presented to the bubble.

Kabanov and Frumkin<sup>1</sup> studied the forces acting to cause adhesion of bubbles of hydrogen to a large surface of mercury, which served as an electrode in a dilute sulphuric acid solution. The force acting to hold the bubble to the mercury surface was found to be the vertical component of the surface tension between hydrogen and the sulphuric acid solution. The forces tending to cause the bubble to separate from the mercury surface were found to be: (1)

the force exerted due to the internal pressure of the bubble acting upon the area of contact between the bubble and the mercury surface, and (2) the buoyant force of the bubble minus the hydrostatic force acting at the base of the bubble.

These investigators photographed bubbles that were just on the verge of separating from the mercury surface. From these photographs, they measured the contact angle between the mercury surface and the tangent to the hydrogen-solution interface at the point of contact between bubble and mercury surface. They calculated the volume of the bubble and the internal pressure of the bubble. The equivalence between the upward acting and downward acting forces was remarkable.

Wark<sup>2</sup> pursued an investigation similar to that of Kabanov and Frumkin, and at the same time. His deductions verified those of Kabanov and Frumkin. He also considered the conditions present in flotation and was aware of the effect of the internal pressure of the bubble. Further, he proposed several conditions under which a small solid particle would adhere to an air bubble in water.

In 1922, Edser,<sup>3</sup> an English physicist, made the following statement. "It must be remembered that no particle could float stably, but for the possibility of variation of the contact angle, for if this were a constant, a slight tilt would inevitably cause the particle to sink." Wark criticized this statement, maintaining that the contact angle does not vary. The experimental data to be presented indicates that Edser was correct.

**Experimental Procedure:** Briefly, the experimental procedure was as follows. A bubble of air was generated in distilled water. A rod of known diameter, one end of which was water repellent, was attached to this bubble. The weight of the rod was measured. The internal pressure of the bubble was measured with a manometer. The hydrostatic head from the surface of the water to the bottom of the rod was measured. The angle between the horizontal projection of the end of the rod and the tangent to the bubble at the circle of contact be-



tween rod and bubble was measured. This will be denoted as  $\theta$  hereafter. The forces acting upward and downward, between rod and bubble were then evaluated.

As shown in fig. 1, the bubble of air was generated by displacement of air by water, from a stoppered bottle. A square glass jar contained the water in which the air bubble was generated. The glass tee terminated in a manometer, as shown.

The glass jar was placed on a metallograph bench. The magnified image of the bubble and attached rod, and the surface of the water in the jar was observed on a ground glass plate. The manometer was placed in a lantern slide projector and its magnified image was thrown onto a screen.

Various sizes of bubbles could be generated by changing the size of the glass tube (a), which fitted into the short length of the tee by means of a piece of rubber tubing. The size of bubble was more closely controlled by means of an adjustable screw clamp at (b).

A tracing of the image on the ground glass plate of the metallograph was made, using onion skin paper. Simultaneously the difference in liquid levels of the limbs of the manometer image was measured. From the tracing, the angle  $\theta$  and hydrostatic head were measured, as well as the diameter of the rod and of the bubble tube.

The magnification factor for the metallograph was obtained by measuring the outside diameter of the tip of tube (a), with a micrometer, and also measuring the width of the image of the tip of this tube on the ground glass plate. The ratio of the two measurements gave the magnification factor obtaining in each test. The magnification factor of the lantern slide projector was obtained in a similar manner by measuring the outside diameter and the outside diameter of the image of the manometer tube. The tracing appeared as shown in fig. 2.

In order to check the accuracy of the pressure measurements, La Place's equation was used.

$$\Delta P = \gamma \left( \frac{1}{R_1} + \frac{1}{R_2} \right) \quad [1]$$

where  $\Delta P$  = pressure differential across a gas liquid interface in dynes per  $\text{cm}^2$  and  $\gamma$  = surface tension in dynes per cm, and  $R_1$  and  $R_2$  are the principal radii of curvature of the interface at the point where the pressure differential is measured, both expressed in centimeters.

A bubble was generated in distilled water. No rod was attached to it. The radius of curvature of the bubble at its nadir was measured. The hydrostatic head from the surface of the water to the nadir of the bubble was measured. The difference in levels in the limbs of the manometer was measured. Acetone, rather than water, was used in the manometer because the difference in levels was greater than if water had been used; and also this liquid responded more quickly to pressure changes than did water, because of its lower surface tension. Since a differential manometer was used to measure internal pressure, atmospheric pressure was not taken into account in measuring hydrostatic head.

The radius of the bubble at its nadir is difficult to measure accurately. A pair of calipers was used and by trial and error the radius of the line passing through the nadir of the magnified image of the bubble was measured. The calipers traced the

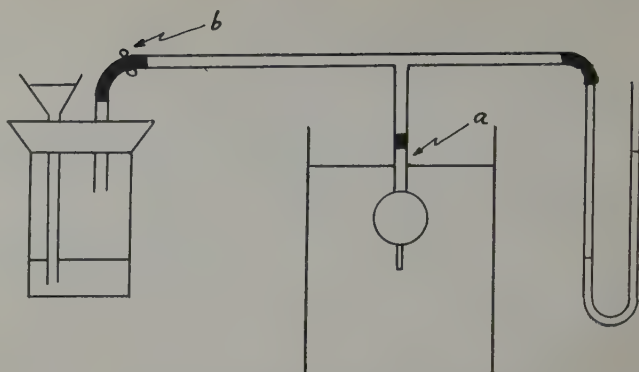


Fig. 1.—Experimental apparatus.

curve for at least a half inch on each side of the nadir. This fit was considered satisfactory.

Applying Eq 1, substituting 72 dynes per cm for the surface tension of distilled water at  $26^\circ\text{C}$  and  $R_1 = R_2$  since the radii of curvature are equal at the nadir, we can calculate values of  $\Delta P$  and check against observed values of  $\Delta P$ , Table I.

Table I. Sensitivity of Measurements

Test	Diam of Tube(d),Mm	Pressure,Mm Water		$R_1$ , Mm	$\Delta P$ , Dynes per $\text{Cm}^2$	
		Manometer	Hydro. Head		Calc.	Obs.
1	9.10	8.37	5.72	5.74	251	259
2	9.10	12.25	8.65	4.12	350	352
3	4.28	21.30	10.30	1.34	1,075	1,077

The density of the acetone used in the manometer was 0.79. This factor was used to convert millimeter of acetone to millimeter of water. The magnification factor for the metallograph camera was 9.78 and that for the lantern slide projector was 10.24.

These tests indicate that the apparatus was sensitive enough to measure the pressure differential between the outside and inside of the bubble.

**Attachment of Small Rods to Bubbles:** In the first series of tests, glass rods of varying length and di-

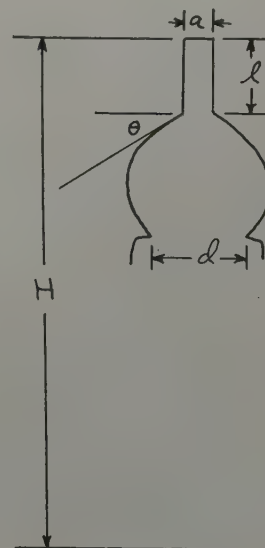


Fig. 2—Tracing of system bubble particle.



Table II. Glass Rods Used

Test	Millimeters			Weight, Grams	Mm H <sub>2</sub> O		$\theta$	Forces, Dynes				Sum	
	$a$	$l$	$d$		$H$	$M$		Wt.	Down	Up	S. T.	Down	Up
1	0.83	9.40	3.30	0.0150	19.10	15.50	45	14.75		1.90	13.25	14.75	15.15
2	0.41	10.10	7.25	0.0033	16.90	9.70	12	3.23		0.93	1.92	3.23	2.83
3	1.05	6.82	7.25	0.0120	13.95	9.92	22	11.75		3.45	8.80	11.75	12.25
4	1.05	6.82	3.30	0.0120	16.30	15.40	30	11.75		0.76	11.85	11.75	12.61
5	0.88	9.67	3.30	0.0184	19.10	14.45	49	18.00		2.80	15.00	18.00	17.80
6	0.70	11.90	3.30	0.0118	21.80	15.20	30	11.60		2.41	7.80	11.60	10.21
7	0.70	4.10	3.30	0.0042	13.10	15.70	20	4.10	0.96		5.36	5.06	5.36
8	0.70	4.10	7.25	0.0042	11.45	9.87	12	4.10		0.61	3.34	4.10	3.95
9	1.37	7.12	7.25	0.0261	14.50	8.57	32	25.60		8.60	16.50	25.60	25.10
10	0.83	7.18	3.30	0.0092	16.85	15.30	23	9.00		0.82	7.32	9.00	8.14
11	0.83	7.18	7.25	0.0092	14.20	9.62	20	9.00		2.43	6.40	9.00	8.83

ameter were used. The ends of the rods were dipped into molten paraffin and cooled. This end was pressed against a clean glass slide in order to give a plane surface at the end of the rod.

In the second series of tests, copper rods of constant diameter, but varying length, were used. The ends of the rods were filed and then rubbed with crocus cloth in order to give a reasonably plane surface. These rods were cleaned in ethyl alcohol, followed by distilled water, and then dipped into a dilute nitric acid solution and again washed in distilled water. For the first lot of tests in this second series, the rods were placed in a ½ pct solution of sodium aerofloat in water, and in the second lot of tests a ½ pct aqueous potassium ethyl xanthate solution was used. In both cases the rods were removed from the conditioning solutions and attached to a preformed bubble of air in the glass jar. A small stainless steel wire holder was used to attach the rods to the bubble.

**Data Obtained:** Table II lists the pertinent data concerning the first series of tests, in which glass rods were used. Table III lists the data pertaining to the tests in which copper rods were used. In Table III, the first seven tests are those in which sodium aerofloat was used. The last three tests are those in which potassium ethyl xanthate was used.

In both Tables II and III,  $a$ ,  $l$ ,  $d$  refer to the measurements denoted in fig. 2.  $H$  is the hydrostatic head from the surface of the water to the bottom of the rod.  $M$  refers to the manometer reading. Under the heading Forces, Dynes, the column marked Down refers to the difference between  $H$  and  $M$ , expressed in dynes, acting downward. Conversely the column labeled Up refers to the difference between  $H$  and  $M$ , expressed in dynes, acting upward. The last two columns are the sum of all

upward and downward acting forces. The column labeled S.T. refers to the upward force due to the vertical component of the surface tension between air and water. The value 72 dynes per cm was used, since the temperature of the distilled water was 26°C.

**Discussion of Data:** The difference between upward acting and downward acting forces in the several tests is small and can reasonably be ascribed to experimental error. More refined measurements would undoubtedly show closer correspondence.

The bubble covered the whole surface at the end of the rod. In some cases it was difficult to draw the tangent to the bubble at the point of contact with the end of the rod. This was especially the case when the angle between the tangent and a horizontal line was large, since the bubble necked down and was on the verge of giving a reentrant angle.

The series of tests show that the four forces considered account for the equilibrium maintained when a conditioned solid adheres to an air bubble. The angle  $\theta$  varies as the weight of the rod varies, in order to maintain equilibrium.

**Contact Angle and Flexibility:** The value of contact angle measurements made in the usual manner is very important because such measurements tell us the maximum contact angle which can be obtained with a given collector, such as potassium ethyl xanthate.

The static contact angle is the angle measured in the tests listed in Tables II and III. The static contact angle cannot exceed the maximum contact angle for a given collector, otherwise our contact angles measured in the usual manner do not mean anything.

Static conditions do not prevail in a flotation cell. Centrifugal forces are generated which tend to in-

Table III. Copper Rods Used

Test	Millimeters			Weight, Grams	Mm H <sub>2</sub> O		$\theta$	Forces, Dynes				Sum	
	$a$	$l$	$d$		$H$	$M$		Wt.	Down	Up	S. T.	Down	Up
1	0.70	1.35	2.07	0.0045	11.05	16.45	25	4.40	2.04		6.65	6.44	6.65
2	0.70	1.35	3.30	0.0045	11.65	14.65	21	4.40	1.18		5.65	5.58	5.65
3	0.70	3.20	7.25	0.0107	13.52	12.80	43	10.52		0.27	10.80	10.52	11.07
4	0.70	3.20	3.30	0.0107	13.70	16.60	50	10.52	1.10		12.20	11.62	12.20
5	0.70	1.95	7.25	0.0065	12.95	13.10	25	6.37	0.06		6.60	6.43	6.60
6	0.70	1.95	3.30	0.0065	12.65	16.50	30	6.37	1.46		7.90	7.83	7.90
7	0.70	1.95	2.50	0.0065	11.30	15.80	31	6.37	1.65		8.10	8.02	8.10
8	0.70	2.84	7.25	0.0099	13.00	12.85	36	9.65		0.06	9.45	9.65	9.51
9	0.70	2.04	7.25	0.0071	14.10	14.65	26	6.90	0.21		7.06	7.11	7.06
10	0.70	1.41	7.25	0.0049	14.10	14.90	19	4.77	0.39		5.15	5.16	5.15



crease the "weight" of the particle. Therefore the contact angle must be greater than the static contact angle for that particular particle bubble system, in order to allow equilibrium to be maintained between bubble and particle. If the centrifugal force is so great that a contact angle greater than the maximum possible for the collector used, is demanded, disruption of the bubble particle system must occur.

Now, the greater is the difference between maximum contact angle and static contact angle, the greater is the ability of the bubble particle system to withstand disruption. This difference is herein referred to as the flexibility of the system. For example, if the static contact angle for a given system is  $50^\circ$ , and if the maximum contact angle is  $60^\circ$ , the degree of flexibility is low, namely  $10^\circ$ , and such a system could not withstand disruptive forces to the extent that a system whose static contact angle is  $20^\circ$ .

Gaudin and Vincent<sup>4</sup> reported that the maximum contact angle decreases as the concentration of collector decreases. Thus, the degree of flexibility of the system is decreased. The coarse particles are affected more than the fine particles, because the contact angle required for static equilibrium is less for fine particles than for coarse particles for a given sized bubble. This is demonstrated by the tests cited in Tables II and III.

The flotation of fine particles of gangue (in addition to the fine gangue mechanically carried over in the froth) can be understood in view of the forces acting. If partial activation of the gangue is obtained, the maximum contact angle would be small when such a partially activated particle reacts with the collecting agent. Coarse particles of gangue will not adhere to an air bubble because the maximum contact angle and hence degree of flexibility is not large enough.

Sutherland<sup>5</sup> has discussed the disruptive effect caused by the greater initial acceleration of the bubble compared to the particle and also the disruptive effect of the viscosity of the fluid.

Spedden and Hannan<sup>6</sup> have photographed the path of bubbles passing through a dilute pulp of galena in water. The disruptive effect of bubble oscillation can be clearly observed.

It is the writer's opinion that Edser meant that the "working" contact angle changes to achieve stability, but not that the maximum contact angle changes.

**Size of Bubbles:** In order to show the magnitude of the effect of size of bubble and hence internal pressure on the degree of flexibility of the bubble particle system, an equation is deduced. Fig. 3 shows the bubble and particle in liquid.

Let  $h$  = distance from surface of liquid to midpoint of bubble, in cm.

$r$  = radius of bubble, in cm.

$l$  = length of rod, in cm.

$a$  = diameter of rod, in cm.

$w$  = weight of rod in air, in dynes.

$P$  = internal gas pressure of bubble, in dynes per  $\text{cm}^2$ .

$\theta$  = static contact angle.

$\gamma$  = surface tension of liquid gas interface, in dynes per cm.

$\rho$  = density of liquid, in dynes per  $\text{cm}^3$ .

Then, equating upward and downward acting forces, when static equilibrium obtains:

$$P \frac{a^2 \pi}{4} + w = a \pi \gamma \sin \theta + (h + r + l) \rho \frac{a^2 \pi}{4} \quad [2]$$

Eliminating the factor  $\frac{a^2 \pi}{4}$ , and simplifying:

$$P - h \rho + \frac{4w}{a^2 \pi} = \frac{4\gamma \sin \theta}{a} + (r + l) \rho$$

But  $P - h \rho = \Delta P_0$  = difference between internal and external pressure of the bubble at its midpoint,

and  $\Delta P_0 = \gamma \left( \frac{1}{R_1} + \frac{1}{R_2} \right)$ , where  $R_1$  and  $R_2$  are the principal radii of curvature of the bubble at its midpoint, or

$$\gamma \left( \frac{1}{R_1} + \frac{1}{R_2} \right) + \frac{4w}{a^2 \pi} = \frac{4\gamma \sin \theta}{a} + (r + l) \rho$$

and

$$\frac{4}{a} \gamma \sin \theta = \gamma \left( \frac{1}{R_1} + \frac{1}{R_2} \right) + \frac{4w}{a^2 \pi} - (r + l) \rho$$

$$\sin \theta = \frac{a}{4} \left( \frac{1}{R_1} + \frac{1}{R_2} \right) + \frac{w}{\gamma \pi a} - \frac{(r + l) \rho a}{4 \gamma} \quad [3]$$

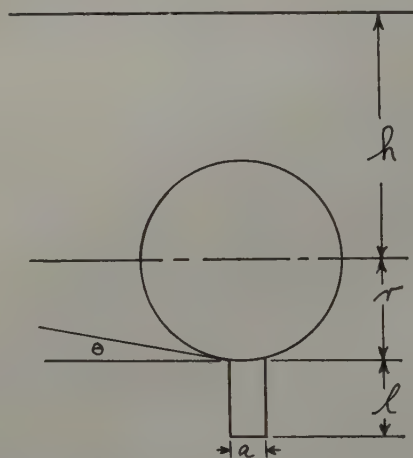


Fig. 3—Bubble and particle in liquid. Static equilibrium.

In order to check Eq 3 we may use the data pertaining to test 1 in Table III. Fig. 2 is the tracing of the image of the bubble and particle used in this test. The magnification factor was 9.20.  $R_1 = 1.52$  mm, and  $R_2 = 1.72$  mm.

Then,

$$\sin \theta = \frac{0.70}{4} \left( \frac{1}{1.52} + \frac{1}{1.72} \right) + \frac{4.40}{7.2 \pi (0.70)} - \frac{(1.52 + 1.35)}{(4) (7.2)} \left( \frac{980}{1000} \right) (0.70)$$

$$\sin \theta = 0.427$$

$$\theta = 25^\circ 15'$$

compared to the observed angle of  $25^\circ$ .

For the purpose of comparison we may compute the value of  $\theta$  if the internal pressure of the bubble had been neglected.

$$w \left( \frac{7.9}{8.9} \right) = \pi a \gamma \sin \theta$$



$$\sin \theta = \frac{w \left( \frac{7.9}{8.9} \right)}{a \gamma \pi} \quad [4]$$

$$\sin \theta = \frac{7.9}{8.9} \frac{4.40}{(0.70) (7.2) \pi}$$

$$\sin \theta = 0.25$$

$$\theta = 14^\circ 30'$$

The ratio  $\frac{7.9}{8.9}$  modifies the weight of the copper rod in air, to that in water, taking 8.9 as the density of copper.

Let us assume that we have a particle of galena in the shape of a rod, the diameter of this rod being equal to 0.15 mm, and the length being equal to 0.15 mm. Its weight in air is then equal to 0.019 dynes. Let  $\gamma = 72$  dynes per cm. The variation of  $\theta$  with change in the size of bubble would then be, according to Eq 3.

Table IV. Variation of  $\theta$  with  $r$

Static Conditions				Acceleration of 50 g	
$r$ Mm	Sin $\theta$	$\theta$	$b$ , Dynes	Sin $\theta$	$\theta$
1.00	0.075	$4^\circ 18'$	4.10	0.349	$20^\circ 26'$
0.50	0.152	$8^\circ 45'$	0.51	0.430	$25^\circ 29'$
0.25	0.304	$17^\circ 42'$	0.064	0.580	$35^\circ 27'$

In Table IV,  $b$  is the buoyancy of the bubble, expressed in dynes. The right hand side of the table shows what  $\theta$  would have to be if the difference in acceleration of bubble and particle were 50 times the acceleration of gravity.

For the purpose of comparison with the data in Table IV, let us calculate  $\theta$  if the effect of the size of bubble and hence internal pressure of the bubble is neglected. According to Eq. 4:

$$\sin \theta = \frac{6.5}{7.5} \frac{0.019}{(0.15) (7.2)}$$

$$\sin \theta = 0.0049$$

$$\theta = 17 \text{ min.}$$

Further, if we assume a difference in acceleration between the bubble and particle of 50 times that of gravity,  $\theta = 14^\circ 15'$ .

It can be observed that the influence of the size of bubble and hence internal pressure of the bubble is very important. Thus for a given sized particle, the value of the static contact angle  $\theta$  increases as the size of bubble decreases as shown by Table IV, and Tables II and III. The difference between the maximum contact angle possible for a given collector and the static contact angle is greater the larger the bubble; this means that the degree of flexibility is greater. The previous discussion has pointed out that the greater the degree of flexibility, the greater is the chance for a bubble particle system to withstand rupture caused by centrifugal force, oscillations of the bubble, and other factors, in a flotation machine.

Now this does not mean that only large bubbles are desirable in a flotation cell. Experience has shown that a large number of bubbles are desirable for best results. Thus, the greater the number of

bubbles the greater is the number of particles attaching to them per unit of time, and hence the greater is the rate of flotation. For a given cell volume, and a given air intake per unit of time, the smaller the bubble, the greater the number of them.

The size distribution of bubbles should be governed by the size distribution of particles present in a cell. For the largest size of particle, small bubbles which demand a large static contact angle would not be desired for the reasons already discussed. But for the smaller sized particles these small bubbles would not demand a large static contact angle and hence would be very desirable because of their number.

However, the bubbles which are so small that they demand a large static contact angle when attached to the smallest size particles are undesirable because the degree of flexibility is small. If a flotation cell is used in which disruptive forces are extremely small, these small bubbles may be desirable.

A comparison can be made between the size distribution of bubbles in a flotation cell and the size distribution of balls in a ball mill. We know that large balls are required to crush the largest pieces of solid. But this does not mean that we want all large balls. Experience has taught that a graded ball charge is desirable, the size distribution of balls depending upon the size distribution of the particles of solid to be ground. Those balls which are so small that they do no useful work are not desired. They take up space that could be occupied by larger balls which do useful work. They are an inevitable result of the wearing down of a ball charge and have to be accepted. Similarly in a flotation cell, the very small bubbles have to be accepted in creating the size bubble desired.

Speeding up the impeller in a flotation cell results in smaller bubbles, similar to a finer ball charge in a mill. A point may be reached where the percentage of bubbles that are too small to be useful is greater than the percentage of small useful bubbles. That is a problem for the design engineer.

### Summary

1. The forces acting between an air bubble and an attached solid in static equilibrium have been measured.
2. The importance of the flexibility of a particle bubble system is discussed.
3. The importance of the size of bubble has been emphasized.
4. An equation has been deduced that correlates the forces acting between a gas bubble and an attached solid particle.

### References

- <sup>1</sup> Kabanov and Frumkin: *Ztsch. fur Phys. Chemie Abt. A* (1933) **165**, 433-452.
- <sup>2</sup> I. W. Wark: *Jnl. Phys. Chem.* (1933) **37**, 623-644.
- <sup>3</sup> E. Edser: British Assn. Repts. of Colloid Chemistry. (1922) **4**, 263.
- <sup>4</sup> A. M. Gaudin and K. Vincent: *AIME, Mining Tech.* (Nov. 1940) TP 1242.
- <sup>5</sup> K. Sutherland: *Jnl. Phys. Chem.* (1948) **52**, 394.
- <sup>6</sup> H. R. Spedden and W. S. Hannan, Jr.: *Trans. AIME* (1949) **183**, 208; *Mining Tech.* (March 1948) TP 2354.



## on Conical Trunnion Overflow and Cylindrical Grate Ball Mills

by Jack White

This paper gives details of the results of careful testing carried out on two types of ball mills, conical trunnion overflow and cylindrical grate discharge, on identical ore. The object of the test work was to determine which ball mill was the most economical to install for future extensions to the Mufulira concentrator.

MUFULIRA Copper Mines, Limited, conducted tests on the concentrator in 1942 to determine which type of ball mill, that is, Conical Trunnion overflow or Cylindrical Grate discharge, produced the greater quantity of finished product at the lowest cost.

All the data which follows applies to Mufulira ore and to the particular degree of grinding produced with the classifiers installed at Mufulira concentrator.

*JACK WHITE, Member AIME, is Concentrator Superintendent, Mufulira Copper Mines, Ltd., Mufulira, Northern Rhodesia.*

*San Francisco Meeting, February 1949.*

*TP 2729 B. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received Feb. 28, 1949.*

**Test Procedure:** The test was carried out using ball mills No. 1 to 9 (conical trunnion overflow) as the standard with which to compare No. 10 (cylindrical grate discharge mill). The conical trunnion overflow ball mills are 10 ft diam by 72 in., and the cylindrical grate discharge mill is a No. 99. Each mill runs in closed circuit with a Dorr type DSFX classifier 8 ft wide by 30 ft long. The feed is considered as being identical to all ten units. The figures for the test were derived as follows.

**Dry Tons Milled:** For No. 10 unit (cylindrical grate discharge mill) this figure was taken from the ball mill weightometer, which was corrected for error, and then for moisture content. The weightometer was checked twice a week during the test, and the error did not exceed 3 pct at any time. The moisture was obtained from a 24 hr moisture sample taken by the ball mill operator hourly, and placed in a closed bucket. The sample was cut where the conveyor drops the feed into the ball mill chute. The Mill Sampling Department made a daily moisture determination on this sample. To obtain the dry tons on units No. 1 to 9 the No. 10 figure was deducted from the month's dry tons milled figure,

which, in turn, was taken from the crusher weightometer tons corrected for weightometer error and moisture. All tons reported in this test are short tons of 2000 lb.

**Ball Mill Hours:** This was taken daily from the Shift Boss reports giving actual running hours for each unit.

**Screen Analyses: Coarse Heads (Ball Mill Feed):** The monthly composite coarse heads screen analysis, done by the Mill Sampling Department, was used for the minus 200 mesh in the feed to the mills.

**F. X. Classifier Overflow:** The ball mill operators took an hourly 24 hr sample of the overflows of units No. 1 to 9. This was a hand sample, but cut with a special cutter, and is believed to be very reliable. On No. 10 unit, an hourly 24 hr sample, and an hourly shift sample were cut in the same way. These five samples were split, filtered, and dried by the Research Department and were then wet screened on a 200 mesh screen. The average of the three shifts samples on No. 10 unit had to check the 24 hr sample within 1.5 pct to be acceptable. A weighted composite was cut each day from both 24 hr samples, and at the end of each month it was screened as follows:

Wet screened through 325 mesh, the plus 325 then screened on a rotap for 15 min through 65, 100, 150, 200, and 325. This was done in duplicate. All splitting, drying, and screening of samples was done by one of the Mill Research staff, to insure accuracy.

**F. X. Classifier Overflow Densities:** These were taken from the Shift Boss reports. Densities were taken by the ball mill operator every hour.

**Circulating Loads:** These were taken a number of times during the test both by direct method (weighing a measured portion of classifier rake sand, and deducting moisture) and by calculation using the formula:

$$Y = \frac{X(b-a)}{a-c}$$



Where  $X$  = tons per hour original feed.

$Y$  = tons per hour classifier sand.

$a$  = percentage of minus 100 mesh in ball mill discharge.

$b$  = percentage of minus 100 mesh in classifier overflow.

$c$  = percentage of minus 100 mesh in classifier sand.

**Ball Loads:** No. 10 unit was charged with 75,000 lb of balls at the beginning of the test and an effort was made to keep mill amperage constant. This met with some success, but on several occasions, by inspection, ball level was found to have dropped. For the last month's test, the consumption figure for the preceding months was used as a guide in adding balls, and this resulted in a very steady ball load. From the average level, as measured when the mill was down, it was calculated that the average load was 72,000 lb.

**Power Figures:** The power meter on No. 10 unit was read every morning at 7 a.m. and the power consumed for the preceding day calculated. The month's figures, the sum of these daily power figures, corrected for meter error, were used for the No. 10 mill. For the mills No. 1 to 9, the monthly power figures as provided by the Electrical Department were used. This figure was obtained by monthly meter readings, instead of daily, and was subjected to meter error correction.

**Screen Analysis of Ball Mill Feed:** A typical screen analysis of coarse heads (feed to all ball mills) is given in Table I.

Table I. Screen Analysis of Coarse Heads

Screen Size		Per Cent
Plus	0.525 in.	0.4
	0.371 in.	9.8
	3 Mesh	15.2
	4	8.3
	6	8.6
	8	5.8
	10	8.9
	14	5.5
	20	4.5
	28	4.3
	35	5.0
	48	2.9
	65	2.9
	100	4.7
	150	2.4
	200	2.3
	325	1.7
Minus	325	6.8
		100.0

**Duration of Test:** The investigation was carried out over a period of seven months, from June to December, 1942, inclusive. The September test is not included because of the time down for lining the No. 10 mill, and changing the drive motor. Tests for June, July, and August are thought to be less

Table II. Summary of Test

	June	July	August	Oct.	Nov.	Dec.
	Mills No. 1 to 9	Mill No. 10	Mills No. 1 to 9	Mill No. 10		
Horsepower	385	463	379	450		
Per cent—200 mesh in classifier overflow	50.55	52.44	52.46	52.03		
Dry tons per ball mill hour	31.85	35.48	31.68	37.55		
New tons—200 mesh per ball mill hour	13.04	15.20	14.00	16.32		
Kw-hr per dry ton milled	9.00	9.74	8.92	8.92		
Kw-hr per ton—200 mesh produced	22.04	22.78	20.32	20.55		
Grinding balls used in lb per ton	2.88	3.68	2.78	3.53		
Liners (gross) used in lb per ton	0.43	0.63	0.43	0.63		

Table III. Screen Analysis of F. X. Classifier Overflows

	June	July	August	Oct.	Nov.	Dec.
Mesh	Mills No. 1 to 9	Mill No. 10	Mills No. 1 to 9	Mill No. 10		
Plus 65	6.0	4.5	6.4	5.0		
100	14.1	12.9	13.3	13.9		
150	15.3	15.3	14.3	15.1		
200	13.8	14.3	13.8	14.3		
325	9.6	10.1	9.5	9.8		
Minus 325	41.2	42.9	42.7	41.9		
	100.0	100.0	100.0	100.0		
			Conical Trunnion Overflow	Cylindrical Grate		
Average per cent solids in F. X. Classifier Overflow				24.7	21.9	
Average circulating load, per cent				420.0	460.0	
Ball load in mills in short tons				36.0	36.0	

accurate as regards power figures than those for October, November, and December, because during the former months it was discovered that the No. 10 ball mill motor was somewhat defective. The summary of the test is, therefore, split into two halves in Table II and III.

**Observation:** (1) To obtain a comparable grind it was necessary to overflow the F.X. Classifier at 2 to 3 pct less solids with the cylindrical grate mill than with the conical trunnion overflow mills.

(2) There appeared to be little difference with regard to ease of operation. Overloads on the conical trunnion overflow mills were reflected immediately in the sand load on the classifier, whereas the cylindrical grate mill overloaded internally. This condition was indicated by a drop in amperage and a definite decrease in mill noise.

(3) From the maintenance point of view, the cylindrical grate mill was much more difficult to inspect, and required more time to reline.

**Conclusions:** (1) The cylindrical grate mill with a ball load of 72,000 lb had a one sixth greater grinding capacity than a conical trunnion overflow mill with the same tonnage of balls, on Mufulira ore.

(2) Consumption of grinding balls was 27 pct greater for equal tonnage ground in the cylindrical grate mill.

(3) Consumption of liner steel at 0.63 lb per ton (gross) was 32 pct higher than for the conical trunnion overflow at 0.43 lb per ton (gross).

(4) In terms of power cost per ton of finished product produced, the cylindrical grate mill required from 1 to 3 pct more power than the conical trunnion overflow, on Mufulira ore.

(5) The cylindrical grate mill produced about 2 pct less oversize (plus 100 mesh) than the conical trunnion overflow, because the classifier was run at a lower density, but at equal densities the cylindrical grate grind was inferior to the conical trunnion overflow.

(6) Work done by either type of mill was directly proportional to power input.

(7) Considering power costs approximately equal, the cylindrical grate ball mill produced more finished product at the expense of considerably more steel consumption.



# Radioactivity

at the Caribou Silver Mine, Boulder County, Colorado

by G. Carman Ridland

A program of exploration for radioactive deposits, conducted in 1945 in the well-known mineralized areas of the Front Range, Colorado, was rewarded with the discovery of pitchblende in a dump at Caribou Hill. The presence of pitchblende in the silver veins at depth apparently is not expressed as gamma-ray anomalies at the surface.

## Introduction

**Front Range, Colorado:** The majority of the rocks comprising the Front Range of Colorado are pre-Cambrian schists, gneisses, and intrusives which have been elevated to form part of the Southern Rocky Mountain physiographic province. The region is noted mainly for its production of gold and silver, but ores of tungsten, fluorspar, uranium, copper, and iron have been worked. Geological literature<sup>1-4</sup> describes pitchblende and other radioactive minerals occurring at Jamestown, Boulder County; Central City, Gilpin County; and Lawson, Clear Creek County (fig. 1). These three adjoining

G. CARMAN RIDLAND, Member AIME, is a Consulting Geological Engineer, New York, N. Y. San Francisco Meeting, February 1949.

TP 2738 IL. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received March 30, 1949.

counties cover most of the highly mineralized section of the Front Range which extends from Jamestown to Breckenridge. As far as the writer is aware, uranium has been found only in the portion northeast of Lawson.

**Silver Deposits at Caribou Hill:** The historic silver deposits of Caribou Hill, located approximately

half way between Jamestown and Central City, and 35 miles in a straight line northwest of Denver, appeared particularly intriguing as an area to prospect for uranium for the following reasons: here exists an isolated group of high-grade silver veins in the center of a mining district containing occurrences of uranium, and, in two of the well-known uranium deposits of the world, those at Great Bear Lake, Canada, and Joachimsthal, Czechoslovakia, uranium is associated with silver in similar fissure veins. An exploration program, conducted in the spring of 1945, directed special attention to Caribou Hill and was rewarded with the discovery of heavy, canary-yellow stained specimens of pitchblende.

The veins are confined to shear zones cutting a north-south, elongated quartz-monzonite stock of Tertiary (?) age lying in the axis of a major anticline of the Idaho Springs biotite schist, the oldest pre-Cambrian formation in the district. The stock is one and three-quarter miles long by more than a mile wide, and Caribou Hill is approximately at its center.

The ore zones vary in width from a few inches to 15 ft, and one structure has been traced for a length of over 4000 ft. The ore minerals are galena, sphalerite, ruby silver, native silver, chalcopyrite, pyrite, and pitchblende. They occur most common-



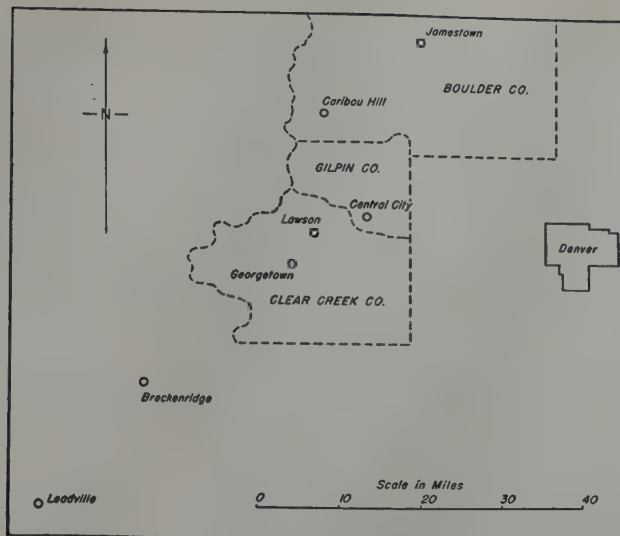


Fig. 1—Index map showing position of some of the major mining districts of the Front Range relative to Denver, Colo.

ly, finely and inconspicuously disseminated in vein matter or in the adjacent altered monzonite. Gangue minerals are mostly quartz, various colored carbonates, barite, some fluorite, and clay minerals that are the alteration products of feldspar and biotite of the wall rock.

An investigation of the mining properties of Caribou Hill resulted in the consolidation of the old Caribou mine property and other, adjacent properties believed to be important. In 1947, the Consolidated Caribou Silver Mines, Inc., began the rehabilitation of the old mine workings.

#### Geiger-Müller Survey

The surface of Caribou Hill and particularly the outcrops of the major ore-bearing structures were traversed with a Geiger-Müller counter sensitive to gamma radiation. As the Caribou mine was unwatered, the accessible levels and stopes were thoroughly traversed with the instrument. At intervals of from 50 to 100 ft, or wherever radioactivity was suspected, Geiger-Müller counts were recorded. The counts were taken for periods of from one to ten minutes and expressed as the average number of impulses per minute.

**Explanation:** Experience has shown that the count for the instrument used may fluctuate from  $\frac{1}{2}$  to  $1\frac{1}{2}$  times the average rate and still be in no way influenced by any near concentration of radioactive material. Hence, any count between 10 and 29 per minute was considered in the normal ( $N$ ) range; from 30 to 47, in the twice normal or  $2N$  range; from 48 to 66, in the  $3N$  range, and so on. If an average count of below 10 was obtained, it was considered below normal and designated  $N/2$ . Experience has further shown that, if a radioactive element is present in sufficient quantity to be seen in its mineral form, or to be detected by ordinary chemical analysis, the count, at the place of exposure, is likely to be  $5N$  or higher.

**Normal (or Negative) Activity:** The surface (except part of the Caribou dump) and most of the

levels registered no above-normal radioactivity.

**Below-normal Activity:** On the 300 level a reading of 8 was obtained on one of the veins but, at this time, no particular significance is attached to this phenomenon.

**Above-normal Activity:** Four levels produced counts registering above-normal radioactivity, and the profiles of the gamma-ray intensities of these levels are illustrated in fig. 2. A single count at an isolated place on the 360 level ran 34, or 5 above the limit of the normal range. A group of counts taken on the 530 level ran 27, 18, 39, 24, 22, 38, 28, 36, 25, and 25 over a length of 220 ft. This is undoubtedly an anomaly, but too mild to indicate a concentration

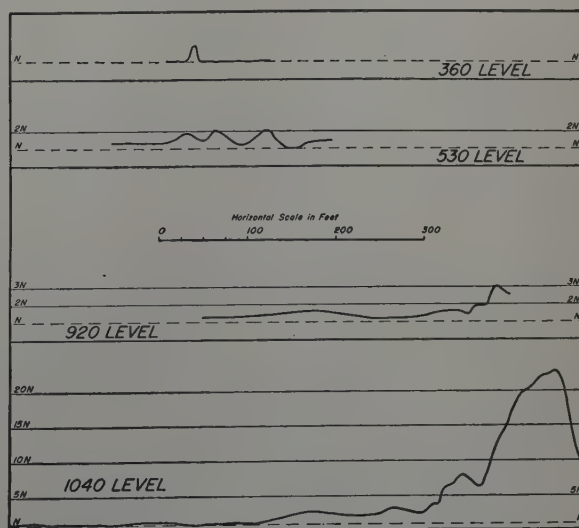
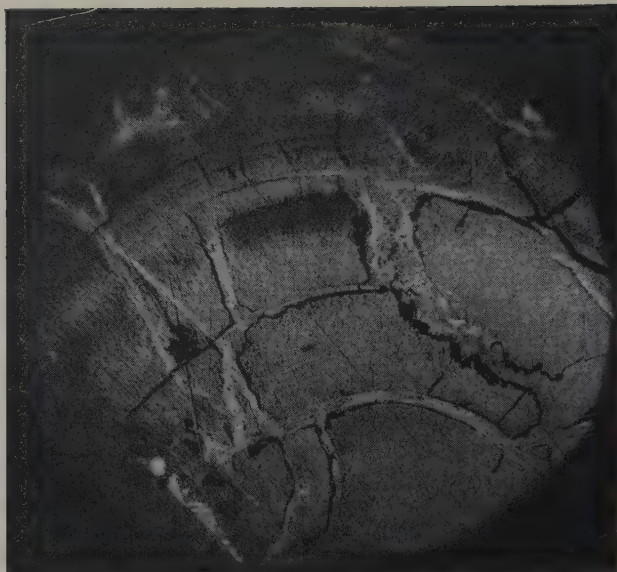


Fig. 2—Profiles of the gamma-ray intensities on the 360, 530, 920 and 1040 levels of the Caribou Mine.

The horizontal scale is in feet and is shown on the diagram. The vertical scales are the gamma-ray intensities expressed in the number of counts over the normal, background count of the country rock. The broken line marked  $N$  represents normal intensity.





**Fig. 3 — Low magnification micrograph of soft, black, colloform radioactive substance from the 1040 level of the Caribou mine. X17.**

The light gray veins are proustite and sphalerite, each containing blebs of chalcopyrite.

of radioactive material that could be seen or detected by ordinary chemical analysis. The average count on the 920 level is noticeably higher than on any of the upper levels, and at the northeast end of the drift it begins to climb abruptly to 3N. The 1040 level is normal in gamma-ray activity except for the northeast 350 ft. The intensity of this anomaly is sufficient to indicate that a radioactive mineral is exposed along approximately 100 ft of this drift.

#### **Description of Occurrence**

**Ore Body of the 1040 Level:** At the northeast end of the 1040 level, channel samples, taken at 10 ft intervals, reveal an ore body 350 ft long with an average width of 4.08 ft and an average grade of 0.061 oz per ton gold, 18.96 oz per ton silver and 1.76 pct lead. It follows a northeast trending structure for 220 ft, then turns off to the east along a subsidiary east-west fracture for the remaining 130 ft. The ore varies from one to nine feet in width and is a siliceous- and carbonaceous-vein filling containing, in places, galena, sphalerite, and a little pyrite. In the east-west section, pitchblende and ruby silver are also visible. The pitchblende occurs in a continuous, black seam 1 to 16 in. wide which begins 45 ft southwest of the junction of the northeast and subsidiary structures and follows the foot-wall of the ore to the junction, then the center of the east-west striking vein for an additional 110 ft. This black material is radioactive and intensely so at places. It is friable and has an earthy luster and readily stains the fingers when handled. Veinlets of an easily shattered, black, vitreous, colloform pitchblende form part of the dark streak, and galena, sphalerite, ruby silver, and colloform pyrite also may be identified by hand lense.

#### **Radioactive Ore under Low Power Magnification:**

A microscopic view of a polished section of the black material, viewed in reflected light under low power magnification, shows pitchblende and coarsely crystallized galena, sphalerite, and proustite. Chalcopyrite is present as small exsolution blebs in sphalerite and proustite and also in places



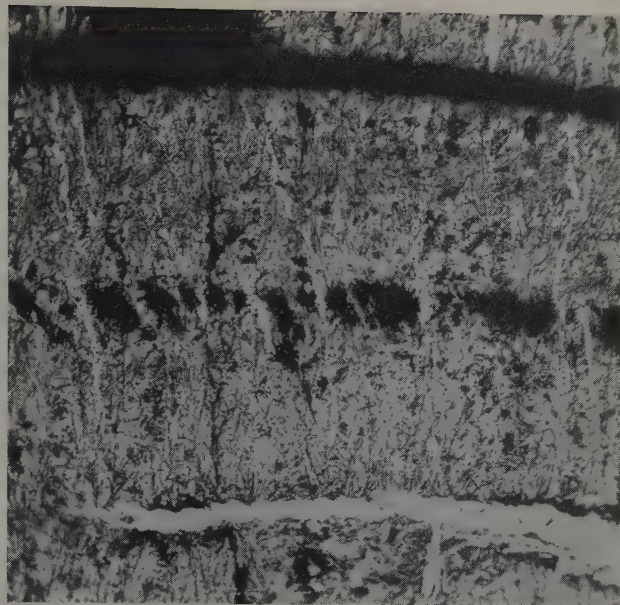
**Fig. 4—Radiograph of polished section, part of which is shown in fig. 3 and fig. 5. X4.**

The white areas are strongly radioactive, relatively pure pitchblende blebs. The dark gray area is the moderately radioactive, colloform substance. The black areas of the specimen are areas where radioactivity is negative.

independent of these minerals. Most of the section, however, is composed of a conspicuous, impure, gray colloform substance (fig. 3) resembling pitchblende in microscopic appearance, but too soft to be pitchblende. It has a sort of ground mass relationship to the more coarsely crystallized minerals. A radiograph of this section (fig. 4) shows it to be less radioactive than pure pitchblende, but does show that it contains, in places, blebs of a strongly



Fig. 5—High magnification micrograph of the colloform substance shown in fig. 3. Oil emersion. X170.



radioactive mineral which has the physical, chemical, and radioactive properties of pitchblende.

**Radioactive Ore under High Power Magnification:** High power magnification of the colloform mixture (fig. 5) reveals an intimate mixture of a silver-white mineral and a dark gray mineral. The latter is believed to be pitchblende because of its color, habit, and reaction to reagents. The white-colored mineral is deeply etched by potassium cyanide and, when so etched and examined under high power magnification, it exhibits a deep red internal reflection. Hence, this second component is apparently a ruby silver and probably proustite which is present elsewhere in the polished section.

### Conclusions

The study of the radioactivity on the surface and within the Caribou mine brings to light three observations of possible interest to those using the Geiger-Müller counter in the search for pitchblende-bearing veins.

1. The surface survey failed to detect any evidence of above normal gamma-ray intensity over the country rock or along the outcrops of the veins, and, if the Caribou mine workings had not reached a depth of 1040 ft and pitchblende discovered on the Caribou dump, the pitchblende occurrence may never have been detected.

2. On the 1040 level, as the Geiger-Müller counter was moved along the drift toward the radioactive ore body, it first registered the anomaly at a distance of approximately 160 ft from the point where the nearest channel sample returned a value for uranium (fig. 2, 1040 level profile). The writer does not believe that the intensity at this location is due to radiation penetrating the full 160 ft; it is more likely due to the result of local radiation emanating from trace amounts of a radioactive element, or elements, in the vein within 5 ft or less of the instrument.

3. Both on the surface and underground, the quartz-monzonite host rock was found to possess a normal gamma-ray count. This observation is in direct contradiction to results obtained from similar

studies<sup>5</sup> made by the writer in the Great Bear Lake Area, N.W.T., Canada, where the host rocks of the silver-pitchblende veins were found to be approximately one and one-half times as radioactive as the adjacent, nonhost formations. Hence, the prospector in search of radioactive deposits may reason that, from the results obtained at Caribou Hill, he need not confine his attention to formations possessing higher-than-average gamma radiation.

### Acknowledgment

The writer is extremely grateful to Mr. Boris Pregel, President of Consolidated Caribou Silver Mines, Inc., for making possible the original program of exploration of the Front Range uranium areas in 1945, the subsequent geological and geophysical studies of the Caribou mine, and, finally, the preparation and reading of this paper. The writer is also grateful to Mr. Elmer Hetzer, Mine Superintendent, for his cooperation and assistance with the field work.

The polished sections, micrographs, radiograph, and microscopic studies were made in the geological laboratories of Princeton University. The writer wishes to thank Professor Edward Sampson and the Department of Geology for the use of its laboratory facilities in the preparation of this report.

### References

- <sup>1</sup> Edson S. Bastin and James M. Hill: Gilpin County and Adjacent Parts of Clear Creek and Boulder Counties, Colorado. U. S. Geol. Sur. Prof. Paper 94 (1917) 123-125.
- <sup>2</sup> E. N. Goddard and J. J. Glass: Deposits of Radioactive Cerite near Jamestown, Colorado. *Amer. Mineralogist* (1940) 25, 381-404.
- <sup>3</sup> Percy R. Alsdorf: Occurrence, Geology, and Economic Value of the Pitchblende Deposits of Gilpin County, Colorado. *Econ. Geol.* (1916) 11, 3.
- <sup>4</sup> E. N. Goddard: Fluorspar Deposits of the Jamestown District, Boulder County, Colorado. *Proc. Colo. Scientific Society* (1946) 15, 1, 19.
- <sup>5</sup> G. Carman Ridland: Use of the Geiger-Müller Counter in the Search for Pitchblende-bearing Veins at Great Bear Lake, Canada. *Trans. AIME* (1945) 164, 117.



# Cyclone Thickener

Applications in the Coal Industry

by M. G. Driessen and H. E. Criner

Possible applications of cyclone thickeners for: (1) clarification of the washery water and, (2) recovery of fine coal from the plant bleed. The paper shows: (1) that it is possible to remove all particles including the very finest, from the circulating water, thus obtaining a closed water system and, (2) that reduction of the ash content of the solids of the plant bleed is feasible by means of cyclone thickeners. Performance data for different size cyclones is given in the form of graphs.

**T**HE cyclone thickener has two important applications in wet washing plants: (1) water clarification, and (2) fine coal recovery.

The thickener consists of a conical chamber into which the fluid is injected through a tangential nozzle at the periphery of the large end. The solids of the slurry are concentrated and removed from the apex of the cone while the "thinner" portion is

M. G. DRIESSEN, Member AIME, is a Consulting Engineer, Pittsburgh, Pa., and H. E. CRINER is Development Engineer, Heyl and Patterson.

New York Meeting, February 1950.

TP 2735 F. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received June 29, 1949.

discharged from a central tube in a flat head at the large end, fig. 1. The advantages of the cyclone thickener are that large volumes of slurry may be treated with small space requirements and that the underflow concentration may be controlled easily and readily to meet varying feed conditions.

**Control of Washery Water Solids Concentration:** The first requirement, in the process of controlling solids concentration, is that the solids be removed from the body of circulating water at the rate at which they are introduced. If this condition is satisfied the concentration will remain at a fixed value and will not increase to uncontrollable levels. The second requirement is that the solids concentration

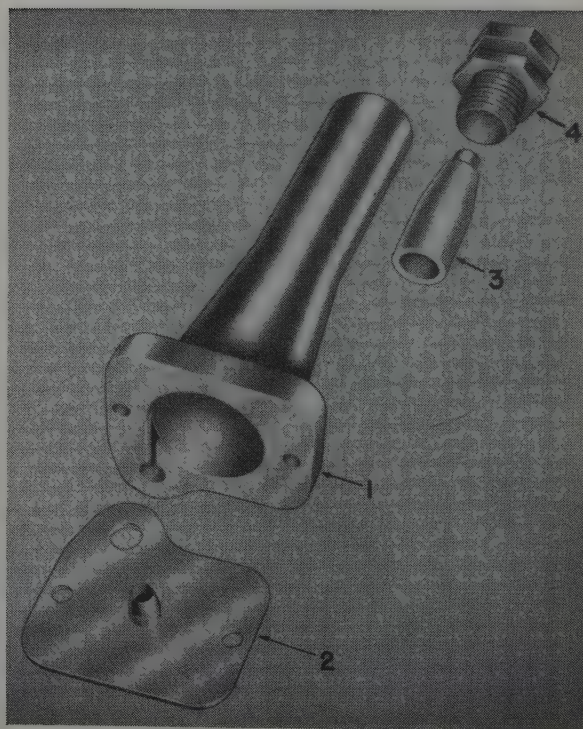


Fig. 1—A 3 in. rubber cyclone thickener.

1—Rubber cyclone body. 2—Cover with overflow opening. 3—Soft rubber orifice. 4—Nozzle retainer.



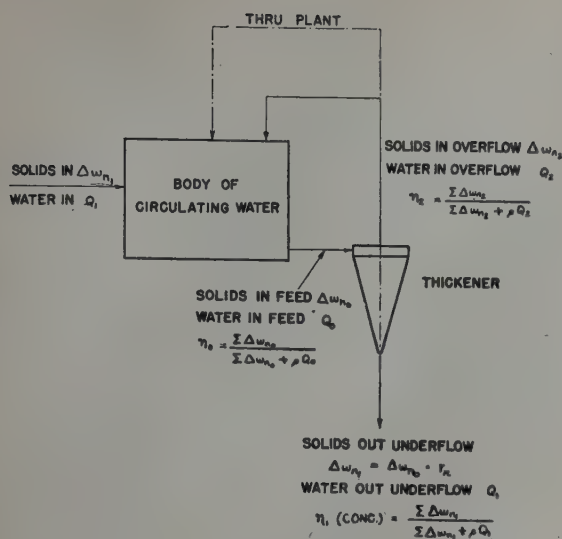


Fig. 2—System to separate solids from circulating water.

Fig. 3—Relative recovery efficiencies of various sizes of cyclone thickeners.

RECOVERY  
(%)

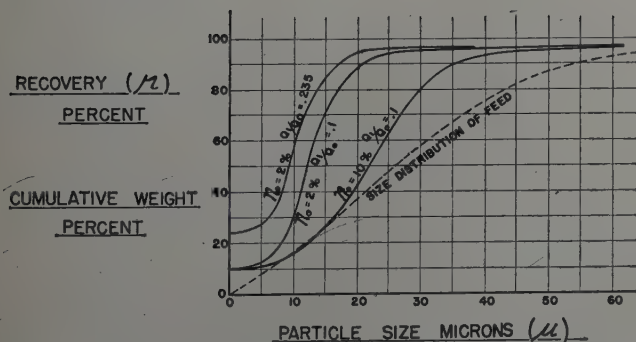
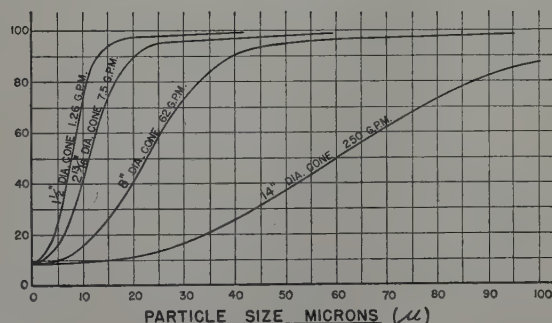


Fig. 4—Influence of feed concentration and flow ratio on recovery of 2 13/16 in. diam cyclone thickener.

be maintained at a usually low level for good washery performance.

To properly apply the cyclone thickener it is necessary to understand the general problem of solids control, to know the amounts and size consist of the solids to be removed from the circulating water,

and to know the recovery characteristics of the thickener.

Suspended solids are introduced to the circulating water through the plant feed and through degradation of the feed within the plant. They are lost to the circulating water through removal with the coal and refuse. (The determination of the rate of loss may be complicated by final clear water flushing or drying with subsequent fines recovery by wet scrubbers.) The difference between the incoming and lost solids must be removed by the thickener if a closed circulating water system is being used. We will designate the amount of this difference by "the required removal rate."

The problem of solids control may be formulated in the following manner. Given the required removal rate in each solid size range, the thickener feed flow and recovery efficiency in each size range; find the stabilized thickener feed concentration and size distribution and the corresponding quantities for the overflow. The plant circulating water will usually consist of thickener overflow so that the result of the study specifies the concentration and

characteristics of the circulating water. Fig. 2 will help in the understanding of the problem.

Let  $\Delta\omega_{n1}$  be the required solids removal rate for size range  $n$ . Let  $r_n$  be the thickener recovery value for this size range and  $\Delta\omega_{n0}$  be the amount in the feed. Then  $\Delta\omega_{n0} = \frac{\Delta\omega_{n1}}{r_n}$  or when the system is in balance it is necessary to circulate  $\Delta\omega_{n0}$  pounds of solids through the thickener to remove the amount of solids ( $\Delta\omega_{n1}$ ) coming into the circulating water. The indices 0, 1, and 2 refer, respectively, to feed, underflow, and overflow of the thickener.

As an example; it is required to remove 10,000 lb per hr of 375 to 230 mesh solids from the circulating water and the thickener recovery for this size range is 0.5 then it is necessary to feed 20,000 lb per hr of 375 to 230 mesh solids to the thickener.

The separate sums of the  $\Delta\omega_{n1}$ ,  $\Delta\omega_{n0}$  and  $\Delta\omega_{n2}$  weights will give the entire weight rates contained in the underflow, feed and overflows respectively. The difference  $\Delta\omega_{n0} - \Delta\omega_{n1}$  will give the separate size range weights in the overflow. Let  $\eta_0$ ,  $\eta_1$  and  $\eta_2$  be the overall concentration in the feed, underflow



and overflow respectively, and  $Q_0$ ,  $Q_1$  and  $Q_2$  be the flow at the corresponding points; then

$$\eta_0 = \frac{\Sigma \Delta \omega_{n0}}{\Sigma \Delta \omega_{n0} + \rho Q_0}, \eta_1 = \frac{\Sigma \Delta \omega_{n1}}{\Sigma \Delta \omega_{n1} + \rho Q_1} \text{ and } \eta_2 = \frac{\Sigma \Delta \omega_{n2}}{\Sigma \Delta \omega_{n2} + \rho Q_2}$$

$\rho$  being the density of water, 8.33 lb per gal.

Sometimes it is difficult to understand how a balance between the amount of solids coming into the system and the amount being removed through the underflow is achieved. Perhaps the following example will clarify this point. The amount of solids fed to the thickener will be almost proportional to the feed-solids concentration at a given total feed flow. Suppose we are removing 10,000 lb per hr through the underflow at 5 pct feed concentration, but there is only 5000 lb per hr coming into the circulating water. Since there is more material going out than coming in the concentration must be decreasing. As the concentration decreases less solid material enters the thickener feed and consequently the amount recovered through the underflow decreases. This gradual adjustment goes on until only 5000 lb per hr is going out through the underflow with a feed concentration of 2½ pct.

The second phase of the application problem is the determination of the required solids-removal rate. This may be most conveniently accomplished by analyzing the plant bleed. Suppose the feed rate to the plant, the volume of circulating water, and the bleed rate have all been maintained at constant values for several hours until no change is discern-

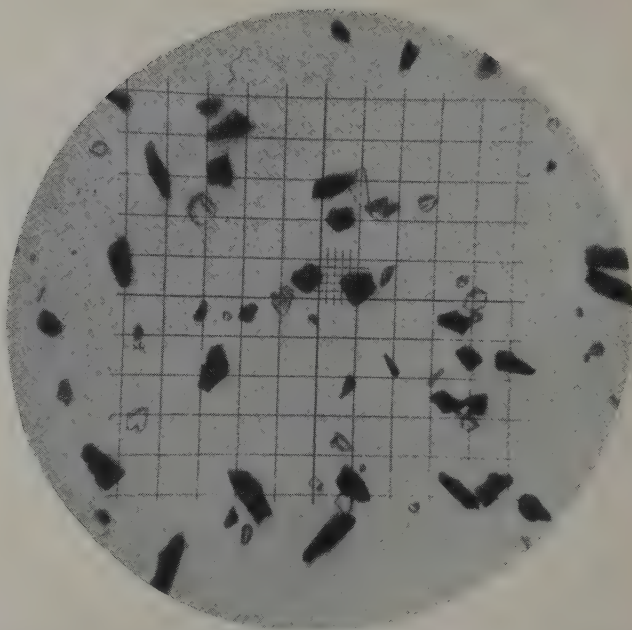


Fig. 5—Micrograph of 10 to 15 micron coal particles.

Small grid square 5.1 micron. Size analysis based on free settling rate of 1.35 sp gr spherical particle.

ible in the bleed concentration. At that time a sample of the bleed water is secured, the bleed volume measured, and the average surface moisture content of the leaving coal and refuse secured. The bleed water sample is analyzed for concentration and solids size distribution. The volume leaving with the coal and refuse is added to the bleed to

Table I—Calculation for Solids Concentration in Thickener Overflow Water

Measured Bleed Volume, 1,282 gpm	Total Weight of Plant Coal and Refuse, 1,400,000 lb per hr
Total Bleed Weight, 663,700 lb per hr	Average Leaving Moisture, Coal and Refuse, 7.25 pct
Water Bleed Weight, 597,000 lb per hr	Weight of Lost Water, 101,300 lb per hr
Solids Bleed Weight, 66,400 lb per hr	Weight of Lost Solids with Coal and Refuse, 11,256 lb per hr
Total Volume of Sample, 1.0750 gal	Total Weight of Solids Lost with Coal, Refuse and Bleed, 76,600 lb per hr
Water Volume in Sample, 1.0000 gal	
Total Weight of Solids in Sample, 0.9252 lb	
Specific gravity of Sample, 1.034	
Total Solids in Sample, 10 pct	

Size Groups (Microns)	(1)  Solids Weight in Bleed Sample (Lb per Hr)	(2)  Weight Solids out with Coal Refuse and Bleed (Lb per Hr)	(3)  Weight Solids out with Coal and Refuse When Rinsed with Thickener Overflow at 3 Pct (Lb per Hr)	(4)  (2)–(3) $\Delta \omega_{n1}$ (Lb per Hr)	(5)  $r$	(6)  $\Delta \omega_{n0}$ (Lb per Hr)	(7)  $\Delta \omega_{n2}$ (Lb per Hr)
0–10	0.0387	3,243	1,323	1,920	0.12	16,000	14,080
10–20	0.0456	3,825	745	3,080	0.28	11,000	7,920
20–30	0.0761	6,376	376	6,000	0.60	10,000	4,000
30–44	0.0926	7,721	127	7,650	0.85	9,000	1,350
44–62	0.102	8,558	52	8,450	0.94	9,000	550
62–125	0.1392	11,683	33	11,650	0.97	12,000	350
>125	0.433	36,250	0	36,250	1.00	36,250	0
	0.9252	77,656	2,656	75,000		103,250	31,250

For underflow concentration of 40 pct at flow ratio of 0.1  $Q_1=224.5$  gpm or 112,500 lb water per hr.  
 $Q_0=2245$  gpm  $\eta_0=8.4$  pct.  
 $Q_2=2020.5$  gpm  $\eta_2=3.0$  pct.



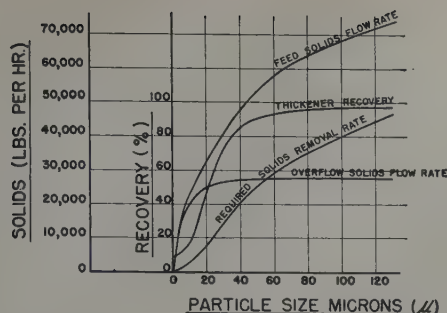


Fig. 6—Amounts and size distribution of solids in feed, overflow and underflow of cyclone thickener application to circulating water solids control.

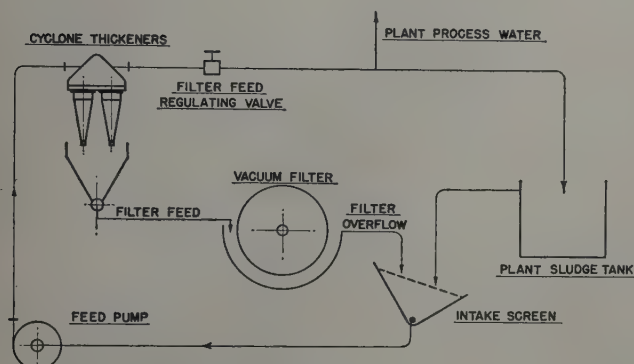


Fig. 7—Schematic diagram for washery water clarification system.

give an overall rate of slurry removal. Call this rate  $Q$  and the sample volume of bleed  $V$ . From the bleed sample volume it has been found that in volume  $V$  there is a weight in size range  $n$  of  $\Delta E_n$  pounds of solids. Then  $\Delta \omega_{n1} = \frac{Q}{V} \Delta E_n$ . Assume for the moment that the thickener overflow concentration and size distribution has been calculated as a first approximation from the above  $\omega_n$ 's. If this fluid is used as the last flushing water for the leaving coal and refuse then the residual moisture on this material will carry some solids in the entrained water. Let the weight of these solids be  $\Delta \omega_n^2 = m \omega_c \Delta \omega_{n2} / \rho Q_2$  where  $m$  is the plant discharge average moisture percentage,  $\omega_c$  the weight of coal and refuse, and the other factors as previously defined. Then as a second and closer approximation, the required removal rate for the thickener is only  $(\Delta \omega_{n1} - \Delta \omega_n^2)$ .

The third requirement in the analysis of the problem is the thickener characteristics. The factors influencing the thickener recovery values are: (1) size and proportions of the unit, (2) the feed concentration, (3) the flow ratio, and (4) the size consist of the solids.

The term flow ratio is defined as the fraction of the feed-water volume that leaves by the underflow orifice. If  $Q_0$  is the feed water amount and  $Q_1$  the rate of water flow from the underflow, then  $Q_1/Q_0$  is the water-flow ratio.

The effect of unit size on the recovery efficiency is shown by fig. 3. The size of the thickener most adaptable to the problem is chosen by considering the size consist of the solids which must be removed. In general if there are large percentages of very fine solids present a small diameter unit must be chosen for the job.

Fig. 4 shows the effect of flow ratio and feed concentration, at one size consist on the recovery values. Curves for the full range variation of these parameters are too voluminous to include in this paper, but these will serve to indicate the trends. The lower solids recovery curves show the effect of changing feed concentration from 2 to 10 pct while maintaining the flow ratio,  $Q_1/Q_0$ , at 0.1. The effect of flow ratio on the recovery is shown by comparing the 2 pct feed curves at  $Q_1/Q_0 = 0.1$  with that for  $Q_1/Q_0 = 0.235$ . We have been speaking of particle size and our curves are shown with particle dimensions in microns ( $\mu$ ). A micron is meant to be a length of 0.0001 cm. We are talking about an equivalent particle as determined by settling rate rather than an actual physical dimension. The actual particle is one whose settling rate is the same as that for the specified diameter, of 1.35 sp gr, settling in water, spherical in shape, and not influenced by solids volumetric concentration.

To secure the characteristic curves discussed above it was necessary to develop an extremely accurate means of isolating the size groups in the samples of feed, overflow, and underflow from the various thickener test runs. Our laboratory uses the decantation method with certain techniques to give the required accuracy. The "purity" of the size group separated by such an analysis may be seen by examining the micrograph, fig. 5.

The method used is one of elutriation by decantation. The sample is at first dispersed in a medium of methyl alcohol and carbon tetrachloride (sp gr, 1.00), and allowed to settle a known distance. As particles of very small terminal velocities should be removed, decantation takes place in a centrifuge. Those particles which are not settled in a certain time, and thus have terminal velocities smaller than a certain calculated velocity (Stokes law), are removed. The rest is diluted with the above mentioned liquid and the process of removing small particles of a certain size range is repeated. For coal particles of a size range 0 to 2.5 microns the process of diluting and decantation has to be repeated up to 50 times.

Although a little elaborate, it is the only method which yields reproducible results, and where the different size ranges are isolated in order to determine further data, such as ash and sulphur content.

An example of the calculation principle discussed above is shown in Table I. Appendix A gives a discussion of the calculation sequence. The results represent several trials gradually yielding the correct size distribution in the feed and overflow. It has been calculated on the assumption that the leaving coal and refuse was rinsed with the overflow water. The curves of fig. 6 summarize the results and give an idea of the size consist of the circulating water after the system has been balanced.

Prospective users of the cyclone thickener are frequently puzzled as to why the fine solids do not continuously "build up" in the circulating water



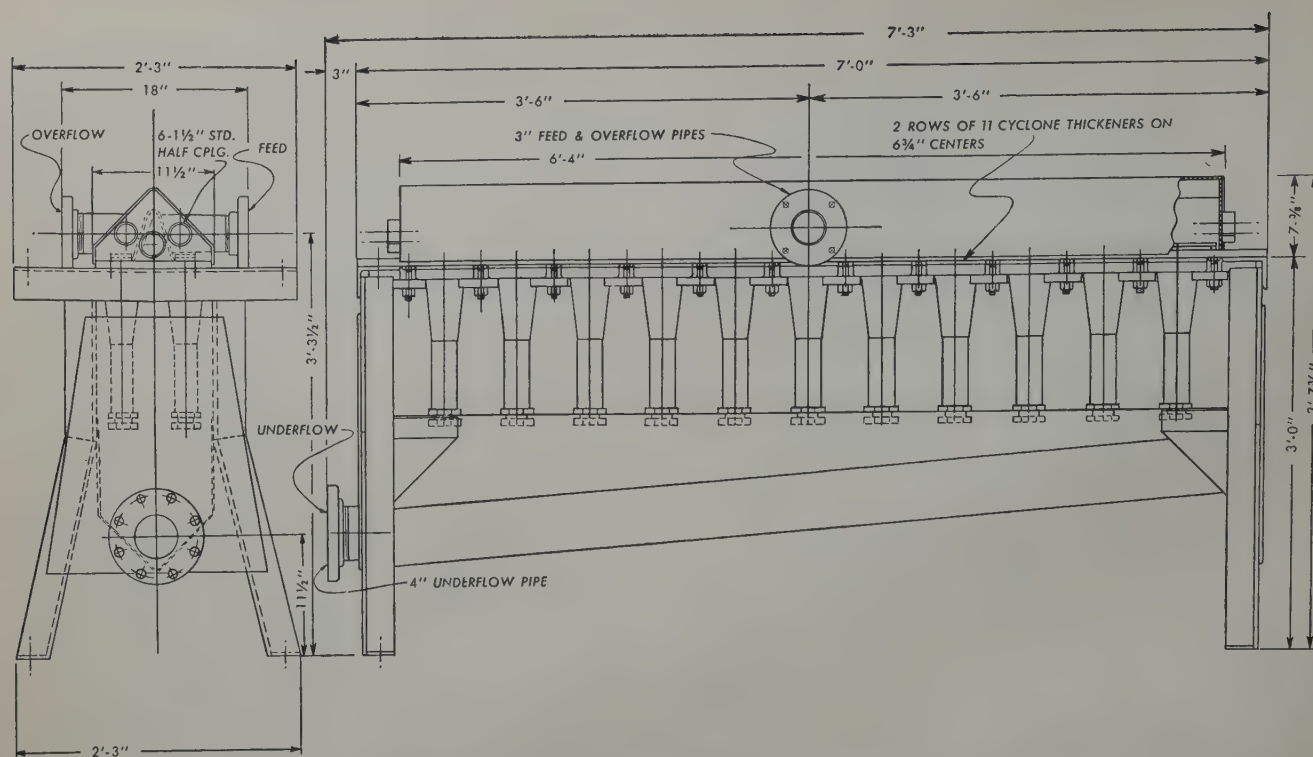


Fig. 8—Arrangement for a 200 gpm cyclone thickener manifold.

because they are being returned through the overflow to the circulating water. The answer to this is that not all of even the very smallest particles are being returned. A fraction is being continuously removed from the underflow along with the concentrated body of larger particles. As long as the recovery value ( $r$ ) does not approach zero this will be true. As long as there is any fluid contained with the underflow solids  $r$  cannot be zero, since the very finest particles tend to make themselves a "part of the fluid." The experimental evidence verifying this viewpoint, is shown on the recovery curves of fig. 4. Particle size groups down to  $2\frac{1}{2}$  microns have been isolated in producing the curves, and all data shows that  $r$  approaches the value of the flow ratio at the smallest particle diameters.

A practical example of such a water clarification system is shown in the flow diagram of fig. 7. This apparatus is being installed in a modern bituminous coal washery handling about 350 tph. The thickeners and filter must remove  $17\frac{1}{2}$  tph of suspended solids from the circulating water. About 32 pct of the material is less than 325 mesh. They must deliver the slurry to the filter at about 30 pct concentration and hold the circulating water to less than 8 pct solids. The thickener feed will be 1600 gpm at 40 psi inlet pressure. The power requirements will be about 3.3 hp per 100 gpm at 70 pct pump efficiency neglecting head and piping losses.

The installation consists of 8 manifold assemblies of standard design flowing 200 gpm each. Fig. 8 shows the manifold arrangement.

**Fine Coal Recovery by Means of the Cyclone Thickener:** The practice of discarding fine coal to a disposal basin or stream may in some cases result in a loss of 5 pct of the plant product. Efforts have

been made to recover as much as possible by screening, but screens of  $\frac{3}{4}$  and  $\frac{1}{2}$  mm size cannot recover a great deal more than can usually be settled in a reasonably sized drag conveyor sludge tank. This is sometimes a rather good coal and it may be beneficiated easily with a simple classifying process

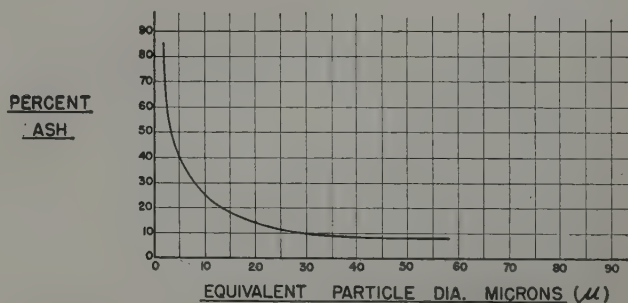


Fig. 9—Ash content - particle size curve for actual slurry sample.

Equivalent size based on 1.35 sp gr spherical particle.

because the high ash material is in the smaller size ranges. Fig. 9 is descriptive of this type of slurry solids.

The cyclone thickener is adaptable to this type of problem because it can classify large volumes of slurry in a small apparatus and produce a thickened clean underflow. Secondary benefits such as increased sludge pond life may be realized.

A larger unit is usually most adaptable to this



type of problem and the rate of flow through it may be changed to vary the classification point.

The elements of such a coal recovery scheme are simple. The thickener feed is removed from the plant sludge tank or from the underflows of dewatering screens or centrifugal driers and pumped to the thickeners. The underflow of the thickeners is usually fed to a filter for further dewatering and all or a portion of the overflow bled from the plant to a disposal basin. If only a fraction of the overflow is bled to waste then the thickener again becomes a part of the circulating water system and its effect on the system may be determined by the procedures discussed in the previous section. The only basic variation from the above discussion is in the modification of the effective thickener recovery values ( $r$ ) by the overflow bleed.

The solids removed from the circulating water are now the sum of the underflow and those bled to waste through the overflow fraction. Let  $p$  be the fraction of overflow bled to waste, then  $p\Delta\omega_{n2}$  is the fraction of overflow solids of size range  $n$  going to waste. ( $p\Delta\omega_{n2} + \Delta\omega_{n1}$ ) is the total weight removed of size range  $n$  from the system. Since  $\Delta\omega_{n2} = \Delta\omega_{n0}$  ( $1 - r_n$ ) and  $\Delta\omega_{n1} = r_n \Delta\omega_{n0}$  the above expression becomes  $\Delta\omega_{n0} \{p(1 - r_n) + r_n\} = \Delta\omega_n$ , where  $\Delta\omega_n$  is as before the required removal rate of size range  $n$  and  $\Delta\omega_{n1}$  was the portion of this removed through the thickener underflow.  $\Delta\omega_n/\Delta\omega_{n0} = r_n^1$  or the new effective recovery value for the combination of thickener and thickener bleed so that  $r_n^1 = p(1 - r_n) + r_n = r_n \{p(1/r_n - 1) + 1\}$ . The new value,  $r_n^1$ , is calculated from the selected bleed fraction  $n$  and the recovery value for the thickener  $r_n$ , are used in the calculation procedure in the same manner as illustrated in Table I.

A proposed solids recovery system operating on the slurry of a midwestern plant had the following characteristics.

The total feed flow to the thickeners was 800 gpm and contained 21 tph of solids. The concentration was 10 pct and the ash content 15.7 pct.

The underflow was recovered at 52 pct concentration. The ash content was 8 pct and it contained 64 pct of the total solids so that the sludge pond life was increased by about 3 times.

The overflow concentration was 4 pct and the ash content of the contained solids was 30 pct.

From the above data it may be seen that appreciable amounts of useful coal can be recovered and beneficiated and at the same time increased settling pond life realized.

This discussion is primarily intended to acquaint operators with the characteristics and possibilities of the cyclone thickener. With such knowledge it is possible to think of other applications unique to the individual washery. It is hoped that the paper will accomplish this result.

### Bibliography

M. G. Driessen: Cleaning of Coal by Heavy Liquids with Special Reference to the Staatsmynen Loess Process. *Jnl.*, Institute of Fuel. August 1939.

M. G. Driessen: The Use of Hydraulic Cyclones as Thickeners and Washers in Modern Coal Preparation. *Trans. AIME* (1948) 177, 240; *Coal Tech.*, Aug. 1947, TP 2135.

H. F. Yancey and M. R. Geer: Cyclones as a Thick-

ener of Coal Slurry. *Trans. AIME* (1948) 177, 262; *Coal Tech.*, Feb. 1948. TP 2351.

D. A. Dahlstrom: Cyclone Operating Factors and Capacities on Coal and Refuse Slurries. *Trans. AIME*, 184, 331; *Min. Eng.*, Sept. 1949.

T. Fraser, R. L. Sutherland and F. F. Giese: Performance Tests of an Experimental Installation of Cyclone Thickeners at the Shamrock Mine. *Trans. AIME*, 184, 439; *Min. Eng.*, Dec. 1949.

### Appendix A

#### Discussion of the Calculation Sequence for Table I:

The sample was analyzed yielding the size fraction weights in col. 1. The total weight of solids and water in the sample was 9.255 lb and the volume 1.075 gal so that the specific gravity was 1.034. The measured bleed volume was 1282 gpm so that the total weight of bleed slurry was  $1.034 \times 8.33 \times 1282 \times 60 = 663,700$  lb per hr. There were 10 pct solids by weight in the sample so the bleed must have contained 66,400 lb per hr of solids. The plant was running 700 tph with an average leaving moisture of 7.25 pct so that the water loss from this source was 101,300 lb per hr again containing 10 pct solid so that the total suspended solids removal rate was  $66,400 + 11,256 = 77,656$  lb per hr. These solids were distributed between the various size groups in the same proportion as found in the sample. As an example there were 0.0387 lb of 0-10 $\mu$  material in the sample so that  $\frac{77,656}{0.9252} \times 0.0387 = 3242$  lb per hr of 0-10 $\mu$  was being removed by coal, refuse and bleed.

We must now establish a tentative figure for the amounts of solids lost with coal and refuse after the thickeners are installed. We know that the solids loss will be somewhat less than a proportional figure for the concentration reduction from 10 to the estimated 3 pct. This will be about 2500 lb per hr instead of the former 11,256 lb per hr. Because of previous experience with this type of problem we will have a fair idea of the distribution of the 2656 lb between the various size groups. We will subtract these tentative figures from the weights in col. 2 and secure an estimation of the amounts removed through the thickener underflow, col. 4.

Using a trial flow ratio of 0.1 and an estimated feed concentration of 10 pct we will copy the recovery figures of col. 5 from the proper recovery curve. The process of dividing the amounts in col. 4 by the recoveries will yield the feed weights of col. 6. Subtracting col. 4 from col. 6 will give the weights of solids in the overflow in each size range.

If it is desired to maintain a 40 pct underflow concentration a total slurry weight of  $\frac{75,000}{0.4} = 187,500$  lb per hr must flow from the thickener underflows. The weight of water contained herein is the difference  $187,500 - 75,000 = 112,500$  lb per hr. For the flow ratio of 0.1, 1,125,000 lb of feed water must be circulated each hour. The total feed solids is the sum of col. 6, and this, combined with the feed water, gives a feed concentration of 8.4 pct.

There are other calculation methods that will eliminate the above trial procedure, but a discussion is too involved to present here. The example should serve to illustrate the principles.



# Chromite and Other Mineral Occurrences

in the Taştepe District of Eskişehir, Turkey

This paper is the first in a series which will describe geology, mining methods, and production costs of some of Turkey's more important minerals. In this paper the economically significant minerals—chromite, meerschaum, and magnesite—of the Taştepe district are described. The cost data presented are believed by the author to be the first made public on Turkish chromite mining.

by Ferid Kromer

**Geography:** The Taştepe district of the Vilayet of Eskişehir is about 20 miles northeast of the city of Eskişehir (approximately midway between Ankara and Istanbul) in western Anatolia. The area is a mountainous one, the highest peak being Taştepe Mountain (5200 ft) which is approximately in the center of the district. The mountains drop off to the deep valley of the Sakarya River on the north and to the plain of Eskişehir on the south.

*FERID KROMER, Junior Member AIME, is a Consulting Mining Engineer and General Manager, Baştaş Türk Maden Ltd., Istanbul, Turkey.*

*New York Meeting, February 1950.*

*TP 2629 H. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received Dec. 29, 1948.*

For the most part, the watershed is on the northern side of the mountain barrier, draining into the Sakarya River, which in turn empties into the Black Sea midway between Zonguldak and the mouth of the Bosphorus. The approximate area covered by the Taştepe district is shown in fig. 1.

Transportation to shipping points is available via the Istanbul-Ankara railroad. The station on this line nearest the mining district is Alpköy station, about 20 miles by road southwest of Taştepe Mountain. Interior roads within the district are poor. Being of dirt, the winter rains and snows render them almost impassable for trucks from about the middle of December until the end of March, thus presenting a considerable transportation problem. However, the roads from the Başören and Taştepe chromite mines to the railroad shipping point at Alpköy station have recently been repaired and will be maintained for all-weather truck transportation.

Detailed climatic data are not available. However, in general the spring, summer, and early autumn months are dry, and good weather may be expected from May until early November. Then the winter rains commence, and heavy snow is usual during January and February.

**Geology:** The mountainous structure of Taştepe belongs to basic rocks of serpentine (Variscan

Orogeny) which is in contact with Paleozoic schists at west, and an Oligocene outcrop of red clays in Margi-Sepetçi region (see fig. 1) at southeast.

The northwest and southwest borders of Taştepe district are, respectively, surrounded by Paleozoic schists and pebbly gray and yellow Neogene clays. More recent formations of alluviums overlay the plain of Eskişehir.

Dark basic rocks of trachytes with hornblende are visible on Türkmenbaba Mountain, at the west of Taştepe.

**Mineral Occurrences:** *Chromite:* the most important mineral found in any quantity in the Taştepe district. The alignment of the deposits of chromite is in general along the line Başören-Taştepe (see fig. 1). The first mines in the area were those of Taştepe and Başören, which were developed over 20 years ago with Swedish capital. Other deposits of chromite, more recently discovered and so far of less importance, are being worked at Kuruçör, Kömürcü, Gelinmezari, and Laçın (see fig. 1). Deposits average generally between 46 and 48 pct chromic oxide, with the exception of the Başören mine which averages 44 pct. However, a new lode, very recently plotted, in the Taştepe mine averages 50 pct  $\text{Cr}_2\text{O}_3$ , 4.6 pct  $\text{SiO}_2$ , and 7 pct  $\text{FeO}$ .

Geological character of the chromite occurrences in the Taştepe mine may be considered typical of most chromite lodes in this area. The indications are that the formations of ore lenses are developed by the segregation of chromite crystals intruded into the serpentinized rock, and exposed later to tectonic movement within the zone of crystallization. All lenticular masses are more or less regular in shape and follow each other in southeast-northwest direction and dip generally  $70^\circ$  NE. Ore lenses do not seem to persist in depth, average depth of two lenses is 60 ft below surface. Three lodes have been mined as open-pit. The average dimensions of individual lenses are as follows: pitch length, 100 ft; breadth, 27 ft; and width, 20 ft.

The lenses and their enclosing rocks are broken by parallel fractures in approximately east-west direction. These joints are filled, except in one lode, with cementing material, which gives to the ore a



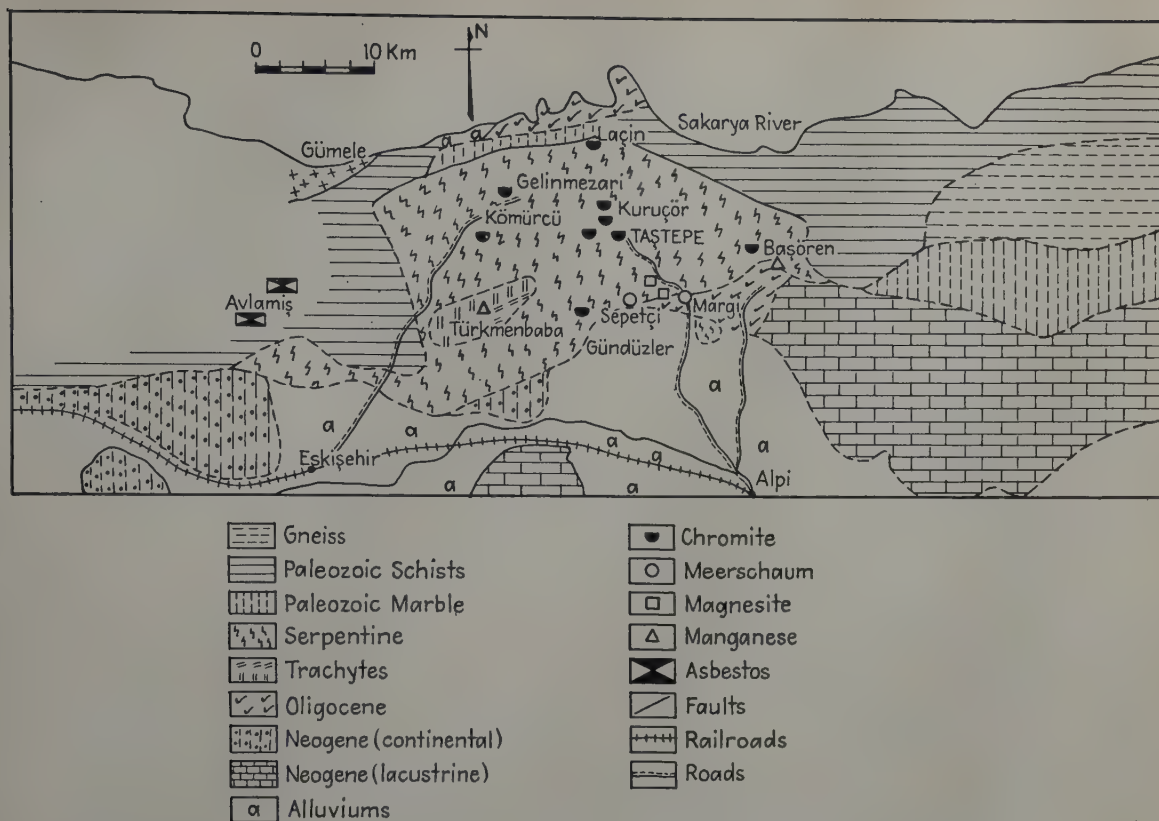


Fig. 1—Geological map of Taštepe District. (After Stchepinsky.)

more compact structure, thus permitting it to be mined with smaller pillars, larger openings, and higher efficiency in blasting. Some of the lodes are faulted. While the grade is maintained up to the faulted plane, it gradually fades out toward the peripheries in the direction of fracture planes. The fading out is sometimes accompanied by visible mineralogical changes in the cementing serpentine of ore.

The lenticular bodies which are oriented with their planes of greatest dimensions roughly in a northwest direction follow one another in that same direction. They do not seem to be connected with each other. However, the territory which falls within that direction, and the sublevels of the present workings need to be explored for determining the depth and extensions of the ore lodes.

Explorations conducted in 1948 have led to the discovery of 12,000 tons (metric) of visible chromite in Taštepe mine, and further investigations will be conducted in the same direction by next year.

At the Başören mine, farther to the east, the lodes are of a more sprinkled and spotted type of formation. The deposits extend in a southeast-northwest direction and reach a depth of about 150 ft below the surface. Present estimates indicate chromite reserves of about 25,000 tons in the Başören mine with an average grade of 44 pct of chromic oxide. Other deposits have recently been developed northwest of the Taštepe mine. No definite information is at present available indicating the ultimate reserves in this area. However, our estimate based on limited exploration made during the past summer indicates reserves as follows: Kömürçü, 2000 tons; Gelinmezari, 5000 tons; and Laçin, 5000 tons.

**Meerschaum:** second to chromite in importance

in this area. Meerschaum occurs as nodular lumps of varying sizes principally between Margi and Sepetçi (see fig. 1). It is found buried in a sandy seam of clay running in a southeasterly direction. The seam varies in depth from 20 to 150 ft below the surface, and is covered by an overburden of red Oligocene clays. The seam carrying meerschaum nodules is characteristically faulted; the throw along the fault varies between 10 and 30 ft with a maximum dip of 70° SE. No idea can be formulated at present about the extent of the meerschaum reserves. The area containing meerschaum appears to extend as far as 15 miles NW of Sepetçi, in the direction of Eskişehir.

**Magnesite:** of frequent occurrence on the eastern slopes of the Taštepe range, particularly between Margi and Taštepe Mountain. It is formed in veins associated with serpentine. The thickness of the beds varies between 6 and 15 ft. An analysis of a recent 2000 ton shipment from this area gave the following results: MgO, CO<sub>2</sub>, 95.5 pct; CaO, 1.5 pct; Fe<sub>2</sub>O<sub>3</sub>, Al<sub>2</sub>O<sub>3</sub>, 0.6 pct; and SiO<sub>2</sub>, 1.0 pct. Several outcrops are visible within the district. At the present writing, the most productive property has a reserve of 30,000 tons of good grade magnesite.

While other minerals such as manganese, iron, mineral water, and certain clays do occur in the Taštepe range, they have not yet been found in profitable exploitable quantities.

**Mining Methods and Mining Costs: Chromite:** All except Taštepe and Başören mines are worked as open-pit mines. However, most of workings in Laçin mine will be centralized in underground exploration and production by the end of this year. In Taštepe mine, for example, a vertical shaft for



individual lenses is sunk in a hanging wall, and the ore is reached by crosscuts. The drifts are 8 ft high by 8 ft wide and are untimbered. From bottom up, tops are robbed and stopes are filled with waste. Taştepe mine has a compressor, driven by a 25 hp Atlas diesel engine. Free air at 90 cfm is delivered at a pressure of 70 psi through a 3 in. pipe to a 3½ x 7 ft receiver in the compressor building, and from there through a 2½ in. pipe to the mine. The compressor building is about 600 yards from the mine. Drilling in the mine is done dry with Atlas jackhammers, ⅞ in. hollow hexagon steel rod with ⅞ x 3¼ in. shank and jackbits are used. Mining is done by three drillers and helpers per jackhammer working an 8 hr shift. Drilling and mucking are done by drillers. About thirteen 3 1/3 ft jackhammer holes are drilled per day, and eighteen sticks (25 mm, 75 g) of Polar blasting gelatin (82 pct nitroglycerin), are used in 13 holes, the rounds usually break 25 to 30 tons of material. Large boulders resulting from the blast are broken by sledging. The broken ore is loaded into an 800 lb hoisting bucket on a truck which is trammed to the shaft bottom over a track of 12 lb rails and 24 in. gauge. The bucket is hoisted up the shaft with a hoist driven by a 6 hp gasoline engine, and dumped into a 2½ ton side-dump car.

At Başören mine the underground water presents some difficulty, but the operations can be continued without interruption throughout the year. At all chromite mines, except Taştepe mine, drilling is done with hand drills. Their operations, however, are similar to those at the Taştepe mine.

Tables I and II show a breakdown of the actual production cost per ton of mining marketable chromite ore at the Taştepe mine. These are presented as representative figures for the operation of an efficiently managed, medium-size chromite mine in this area. At Başören mine cost per ton is somewhat higher than normal because of the presence of underground water and the inefficient use of man power. Open-pit mining costs are somewhat lower, however, the costs in the Taştepe region varying between 5 and 8 Turkish liras per ton of marketable chromite.

During the 1948 season the Taştepe mines will have sold approximately 12,000 tons of chromite, the bulk of which has gone to Austria.

**Meerschaum:** Meerschaum mining is done by sinking shafts vertically until the seam carrying the meerschaum is reached, and then drawing narrow galleries along the strike of the seam. The depth of workings is limited by underground water level. Mining is done by pick and shovel; ventilation is secured through the galleries connecting the shafts. The ground, which is of residual clays derived from serpentine, is solid, but it would necessitate timbering when large openings are driven. The present openings are narrow, 3 x 3 ft, and untimbered. Actually there is no centrally directed organization or exploitation of meerschaum mining in the Taştepe district. Peasants in villages near the mines work mostly during the winter season and are paid according to the quality and amount of meerschaum which they produce. Proper organization and mechanization of meerschaum mines might well constitute an interesting and profitable enterprise. Preliminary treatment of meerschaum consists of

**Table I—Chromite Mining Costs at Taştepe Mine.**

(1 Turkish lira = 36 cents)

Period Covered—June 8 to October 1, 1948	
Sorted chromite ore .....	2,250 Tons
Sorted waste .....	750 Tons
Output hoisted .....	3,000 Tons

**Underground Costs per Ton of Chromite Ore Sorted**

	Labor	Super- vision	Power	Explo- sives	Sup- plies	Total
Prospecting, exploration and development .....	0.53		0.20	0.22	0.44	1.39
Mining .....	1.10		0.60	1.00	0.80	3.50
Underground tramping..	1.70					1.70
Hoisting .....	0.30				0.10	0.40
General underground expense .....	0.10	0.40				0.50
Surface exploration ap- plicable to underground operation .....	0.40					0.40
Total underground expense .....	4.13	0.40	0.80	1.22	1.34	7.89
Surface sorting .....	0.60					0.60
Total direct mining .....	4.73	0.40	0.80	1.22	1.34	8.49

**Table II—Direct Mine Labor**

No. of Workers	Occupation	Average Rate, T.L.	Average Cost per Day
10	Drillers and Field Prospectors	3.20	32.00
13	Trammers	2.50	35.00
2	Hoistmen	3.00	6.00
7	Ore Sorters	1.69	11.83
4	Blacksmith, Motormen, Guardian	3.42	13.68
1	Supervisor	4.00	4.00
37			100.92

removing dirt, drying, and polishing. Price quotations are made according to standard-size cases. Polished meerschaum pieces are sorted into seven grades which are in turn classified into twelve groups. Shipment of the grades known as Lager, Grosse Baumwolle, and Kleine Baumwolle is made in standard 16.5 x 34 x 70 cm cases and of the grades known as Polierte Kasten, Geputzt, and Ciliz in standard cases 34 x 17 x 80 cm. The present price for Lager on the European market is 210.00 dollars per case. No figures are currently available on meerschaum mining costs.

**Magnesite:** Magnesite mining in the Taştepe district was recommenced this year after a long period of inactivity. Magnesite mines are worked as open-pit mining. The cost of mining magnesite is about 4 liras per ton, based on this year's production of 2000 ton. A furnace has been built with a capacity of 15 ton daily for calcined magnesite. This furnace will probably absorb the entire output of this district.

### Conclusion

The Taştepe district offers a great opportunity for extensive mining of chromite, meerschaum, and magnesite ores of good grade. While available reserves are not at present fully known, a rational exploitation of these resources with emphasis on prospecting and exploration, together with a capital investment adequate to permit mining and development by modern methods, should insure a profitable return on the investment.



Comparative

# Furnace Designs

for the Expansion of **Perlite**

by John B. Murdock and Herbert A. Stein

Perlite is a natural volcanic glass which expands into an insulation material upon the application of heat. This paper discusses the operating characteristics of eight different types of furnaces for the expansion of perlite. It also includes a section on the theory of perlite expansion which serves as the basis for the authors' analysis of the furnaces.

**A**N analysis of perlite expansion furnaces must be based upon one consistent theory which explains how and why perlite does expand when heated. There is more than one such theory, so to establish a basis for the rest of this paper, the authors will outline one theory and the data that support it before going into the construction details of different furnaces.

---

JOHN B. MURDOCK, Associate Member AIME, is President, and HERBERT A. STEIN is Secretary, of the Perlite Corp., Phoenix, Arizona.

San Francisco Meeting, February 1949.

TP 2736 H. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscript received Jan. 3, 1949.

---

**Why Perlite Expands:** Perlite is known to contain some dissolved or combined fluid, probably water, which is lost when the material is heated to fusion. It is generally agreed that when particles of this volcanic glass are heated to fusion in such a way that vaporization of this water into steam takes place at the same time as the glass becomes soft, the steam puffs up each particle into a mass of glass foam. It is a simple matter to expand perlite under a blow torch for demonstration purposes, but in commercial installations, the problem is made more complicated by the necessity of making efficient use of both raw material and fuel, and producing an expanded aggregate conforming to

close specifications for size distribution and density, using many different types of perlite rock. The ability of a given operation to fulfill these requirements depends upon the proper handling of four variables. They are: (1) the nature of the perlite rock used, (2) the heating time taken to get the perlite particle up to its highest temperature, (3) the size of the particle, and (4) the maximum temperature the particle reaches.

**Types of Perlite Rock:** Perlite may be classified in many ways, but the characteristics most easily correlated with its behavior in furnaces are its softening temperature, its effective water content, and its "liveliness." Actually there is a close correlation among these three characteristics so it is possible to describe a perlite using any one of them. Lively perlites always have low softening temperatures and high effective water contents, whereas the deader perlites have higher softening temperatures and lower effective water contents. These softening temperatures run from 1400°F to above 2000°F and the effective water contents run from over 2 pct down to 0.2 pct. Included in the category of lively glasses are the green pitchstone from the Edgar claim in Superior, Ariz., the red pitchstone on the Animas highway south of Lordsburg, N. Mex., and a gray glass from Beatty, Nevada. Examples of intermediate perlites (in order of decreasing liveliness) are the Guzman-Lobb perlite with obsidian



balls from Picket Post Mountain, Superior, Ariz. (formerly used by Rheem Research Products, Inc., Los Angeles), the Snow White perlite from the same area (used by the Great Lakes Carbon Corp., Torrance, Calif.), and the light gray granular perlite from Socorro, N. Mex. (also used by the Great Lakes Carbon Corp.). Deader perlites include the gray and black perlites of the Lady Frances mine, Deschutes River, Oregon (used by Dant and Russell, St. Helens, Oregon), and the green John-Clare perlite from near Texas Creek, Colo. (used by Alexander Film Co., Colorado Springs, Colo.). It should not be inferred that one of these perlites is "better" than another. Each rock is suited for use in a particular type of furnace, and in that furnace it makes a characteristic product.

The softening temperature, or expansion temperature, of a perlite may be determined by trial and error in electric muffle furnaces heated to different temperatures, or by optical pyrometer observations in continuous furnaces in which a bed of expanding material is accessible for viewing. Effective water content is tested by a "loss upon ignition test" where the sample is previously heated to some intermediate temperature around 700°F instead of to the conventional 220°F. Prospectors test the relative "liveliness" of different types of perlite rock by exposing half-inch pieces to the flame of a gasoline blow torch. Lively perlites expand easily and to a high degree, even when brought up to temperature slowly, whereas dead perlites subjected to the same treatment expand only slightly, often only at the edges, and sometimes not at all. Failure to expand under a blow torch is not an indication that a given type of perlite is unsuitable for expansion. There is one variety of dead gray glass which does not show even the slightest tendency to expand under a blow torch, yet when it is brought rapidly up to temperature in a properly adjusted furnace, it expands over twelve times. An acetylene torch is useful in determining whether an ore does actually expand if no expansion is obtained under a blow torch, as it allows much quicker heating to higher temperatures. Dead perlites must be subjected to higher temperatures than lively ones in order to obtain a given degree of expansion. This is due to the fact that a higher temperature provides the quick heating needed by dead perlites, as well as to the fact that the softening temperature of a dead glass is generally higher than that of a lively glass.

**Heating Time:** While many furnaces provide other than linear time-temperature curves, it is simpler and just as instructive to describe the time-temperature characteristics of a given operation in terms of two variables, the maximum temperature reached and the time taken to reach that temperature, which may be referred to as heating time. This heating time is usually not equal to retention time, which is the time the perlite remains in the furnace.

Early observers noted that if too long a time were taken to heat perlite up to its softening temperature, much of the dissolved water would seem to have escaped before the softening point was reached, and the final expansion of the glass would not be as great as when shorter heating times were used, even though the final temperatures reached were the same in each case. It was also noted that for

each perlite there was a lower limit to which the heating time could be reduced. As the heating time was decreased, the expansion ratio increased until finally the particles started breaking up under the sudden expansion. For very short heating times, these fragments when collected would show a very low bulk density, indicating that there had been considerable expansion. However, control had been lost over the size distribution of the expanded material, and the particles of expanded perlite were even smaller in diameter than were the grains of crude rock. It was also noted that the strength of the

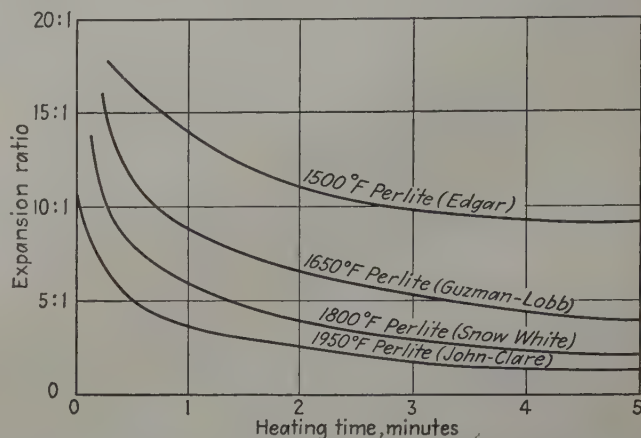


Fig. 1—Maximum possible expansion versus heating time for perlites which expand at various temperatures.

particles dropped quickly as the expansion ratio increased above a ratio of ten to one.

Impressed by the importance of heating time in determining the physical properties of the expanded product, the authors made a study of the exact relation between expansion ratio and heating time. Fig. 1 shows curves illustrating this relation for four perlites ranging from a lively glass which expands at a furnace temperature of 1550°F, to a dead glass which expands at 1950°F. These curves show that for heating times of several minutes, very small expansion ratios are usually encountered, whereas shorter heating times give more expansion. In the case of the Guzman-Lobb perlite, 40 sec was the longest permissible heating time to produce an expansion of ten to one, but with the green pitchstone from the Edgar claim, a ten to one expansion could be obtained *even with* a 4 min heating time. There were still other perlites such as the John-Clare perlite with which an expansion of ten to one could be reached *only with* a heating time of less than 5 sec. It should be noted that the perlites which require the shortest heating time to give good expansion are also the highest temperature perlites, whereas the perlites that can stand the longer heating time take a lower expansion temperature.\*

\* One known exception to this rule is a black obsidian which shows a very good expansion when raised to a temperature well over 2000°F with a heating time of several minutes.

**Size Distribution of the Feed:** Besides the heating time and the nature of the glass used, the size of the particles to be expanded is of importance. It takes more time for heat to be transferred into the interior of large particles, and large particles offer less sur-



face for heat transfer. Therefore, perlites which require short heating times for a given expansion cannot come in particles larger than a certain size. Using the same perlites mentioned above as examples, to give an expansion of ten to one the John-Clare perlite cannot be larger than 12 mesh, the Guzman-Lobb perlite cannot be larger than 6 mesh, whereas the Edgar perlite can come in half-inch pieces and still expand ten to one. This difference in permissible sizes is due to the comparison be-

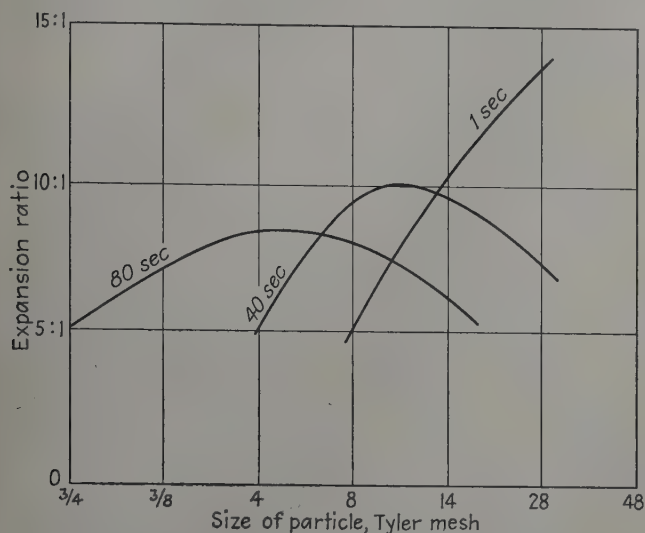


Fig. 2—Expansion ratio of 1650°F perlite versus particle size for different heating times.

tween the maximum heating time allowable for a given expansion of the particular glass and the minimum heating time necessary for heat to travel into the interior of the particles. The maximum size particle which can be expanded to a given degree for a given perlite is that particle which can just be brought up to its softening point in the same time which is the maximum allowable time for that degree of expansion.

Because of greater specific surface, the combined water in finer particles is more likely to be driven off before the softening point is reached. Therefore, if a mixture of coarse, intermediate, and fine perlite is subjected to a certain final temperature after a given heating time, the coarser particles and the finer particles may expand to a lesser degree than the intermediate particles. The coarse particles would fail to expand properly because the heating time was insufficient to bring them up to the temperature necessary for full expansion. The fine perlite would attain less than full expansion because much of its combined water was driven off before its softening point was reached. Fig. 2 shows the relation between expansion ratio and particle size for a 1650°F perlite containing a mixture of sizes and heated at three different rates.

**Maximum Temperature Reached:** The excess temperature attained above the original softening point of the glass determines the degree of expansion just as does the heating time. Most operators keep the furnace temperature as high as possible without fusing the perlite to the furnace wall in order to expand the coarser particles. This leaves no other means of controlling the expansion ratio

except by manipulating the heating time. Attempts to govern the expansion ratio by temperature control alone are not generally successful when a widely sized feed is employed.

**Cooling of Expanded Perlite:** It has been argued that particles of expanded perlite must be cooled rapidly so as not to allow the escape of the gases which produced expansion, thereby allowing the collapse of the bubbles. There is evidence, however, that expanded perlite is actually quite solid and free of stickiness even at temperatures above that of expansion. A type of perlite rock which expands at 1650° F was placed in a heated vessel and inserted into an electric muffle furnace previously heated to 1700° F. The perlite expanded ten times, presumably going through a soft state during expansion. The product was kept at 1700° F for 10 min, after which it was prodded with a rod. None of the particles were stuck together, and the individual grains were crisp and solid. A possible explanation for this observation is that the release of water from the glass during expansion elevates the softening point and causes the material to resolidify after the water is out of the glass and in the cells. This hypothesis would also explain why the bubbles in expanded perlite are of essentially equal size, whereas the pores in bloated shales are of greatly varying dimensions due to merging of bubbles in the soft mass.

**Other Variables:** Some believe that in addition to the variables enumerated above, the expansion of perlite depends upon the oxidizing or reducing character of the flame, or upon the pressure or vacuum in the furnace or discharge hood. There is no evidence to support these claims, since perlite expands quite well in an electric muffle furnace, in which the atmosphere is neither oxidizing nor reducing, and the pressure is exactly atmospheric.

### Furnace Types

The following furnace types will be discussed: the horizontal stationary furnace with and without a preheater, the vertical stationary furnace with four combinations of feed and draft, the short and the long cocurrent rotary furnaces, the counter-current rotary, the indirectly-fired rotary, and the multiple-hearth furnace. With the exception of the indirectly-fired rotary, only refractory-lined furnaces are discussed in this paper.

**Horizontal Stationary Furnace:** The horizontal stationary furnace (fig. 3) or flash furnace as it is sometimes called, consists of a tube 10 to 20 ft long and 6 in. to 2 ft in diameter. The tube is generally



Fig. 3—Horizontal stationary furnace.

refractory-lined and set at a slight angle to the horizontal. The burner and feeder are located at the upper end of the tube, and the lower end is connected by suction pipes to a settling system and a



suction blower. One type of flash furnace employs a settling tank, a cyclone, a blower, another cyclone, and a bag filter system, in that order. Perlite is dropped into the flame at the hottest point, expands, and is carried out in the exhaust gases. The settling

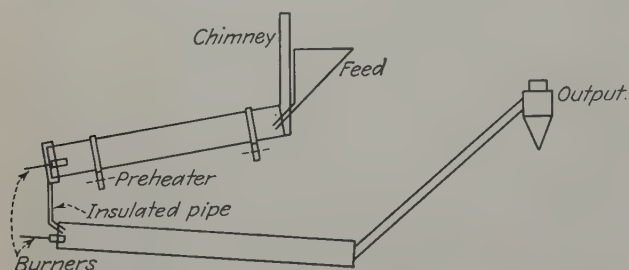


Fig. 4—Horizontal stationary furnace with preheater.

system is usually arranged so that unexpanded material drops out through a reject port, and various products ranging from coarse to fine are separated out in each successive settling device, making screening of the product unnecessary.

Since the heating time afforded by the flash furnace is never over 2 sec and is often only a fraction of a second, particles of perlite coarser than 14 mesh cannot be properly expanded. Even minus 14 mesh feed must be subjected to a furnace temperature much higher than the softening point of the glass in order to transfer the necessary amount of heat to expand the larger particles. At this high temperature, the finer components of the glass are liable to stick to the walls of the tube or to expand too much, producing an excess of "fly fines" which defy collection and have no large scale uses if caught. For these reasons, it is advisable to use a dead glass which expands properly with a short heating time, sized finely and closely, with neither plus 14 nor minus 48 components. The flash furnace is always a "snow machine" unless a very dead glass is selected.

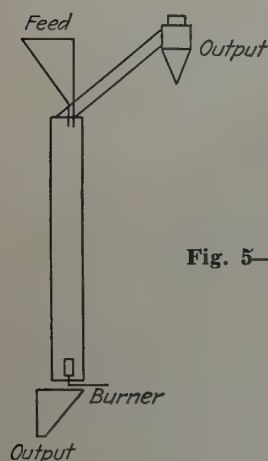


Fig. 5—Vertical stationary furnace.

The big advantage of the flash furnace is that the cost of construction for a given production rate is very low. The fuel consumption, however, is higher than that of any of the other directly fired furnaces, due to the fact that the retention time is too short

to allow proper transfer of heat from the gases to the perlite, and because the exhaust gases from the hot zone are not used to preheat the ore.

#### Horizontal Stationary Furnace with Preheater:

The flash furnace with preheater (fig. 4) is operated in the same way as the plain flash furnace, except that the perlite is preheated in a rotary furnace before being dropped into the hot flame of the flash tube. The rotary unit can be either cocurrent or countercurrent, lined with refractory or unlined, and its length can range from 2 to 30 ft. Retention times in preheaters range from 20 sec up to several minutes.

Preheating allows a broader size distribution to be used in the feed, and permits the use of livelier glasses. One possible explanation for these advantages is that preheating: (1) reduces the combined water content and elevates the expansion temperature of the glass, with greater effect on the fines than on the coarser particles, (2) diminishes the effect of thermal shock which causes unequal thermal expansion and mechanical breakage of the glass particles when they hit the hot zone, and (3) heats the larger particles to near the temperature required for expansion, thereby making them more likely to expand when they receive a short flash of high temperature. With proper temperature controls, a preheater offers an extra degree of control over the ratio of expansion, but with a manually set fuel feed, the preheater can actually be a cause of variation in the quality of the product. The use of preheaters with flash furnaces at present is gen-

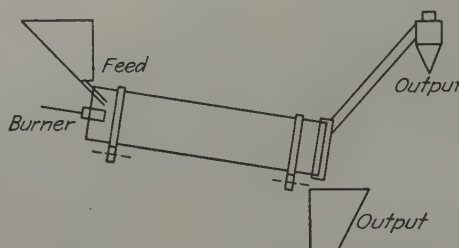


Fig. 6—Short cocurrent rotary.

erally limited to small units for the prevention of excessive fines in the product.

A small preheater fired with a simple, manually controlled gas burner does not add appreciably to the installation cost of a flash furnace, but larger, more effective rotaries can increase the investment by 50 pct.

**Vertical Stationary Furnace:** The vertical stationary furnaces discussed here are refractory-lined tubes placed in a vertical position. Many versions of the vertical furnace are possible because the burner may be placed at either the top or the bottom, and the perlite may be fed at either the top or the bottom. The top-fed, top-fired vertical has all of the characteristics of the horizontal stationary furnace, but requires elevation of the feeder and the preheater too, if used, which adds to the installation cost without yielding any appreciable advantages. With the bottom-fed, bottom-fired vertical, the perlite is shot up from the bottom and is supposed to float in the rising stream of hot gases until it is expanded, at which time it will be carried out the top with the exhaust gases.



The top-fired, bottom-fed vertical was tried but the plant did not get into production. The bottom-fired, top-fed vertical is pictured in fig. 5. The burner is at the bottom and the hot gases pass up the tube as in a chimney and leave at the top. The perlite is fed in at the top and drops through the

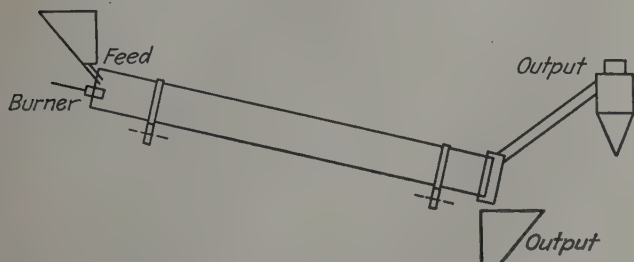


Fig. 7—Long cocurrent rotary.

rising hot gases. The perlite is heated and expanded as it falls until the density of the particles is low enough for them to be caught in the stream and carried back up the tube and out the top. Sometimes this furnace is operated so that part of the material falls on down past the burner. This makes two products: the light fine material which goes up with the hot gases, and the larger heavier particles which drop on down past the burner. Almost any ratio can prevail between these two products, depending upon how the furnace is being operated. Balance in this vertical furnace is very tricky. Large particles with less specific surface tend to fall faster through the rising gases than small particles with more specific surface, yet it is the large particles which should remain in the heat a longer time than the small ones. However, if the draft is set so that all particles eventually do return to the top of the furnace, then the larger particles are exposed for a longer time than the smaller ones because they travel further down before expanding and returning. If there is to be a wide size distribution of particles, the better type of operation for this furnace is for as much of the material as possible to leave the furnace at the top. Considerable difficulty with fusion of hot sticky particles to the walls of the furnace should be expected with this type of vertical. When a particle is in the act of turning around, it has no velocity and is very hot and sticky at that time.

Outside of these special characteristics, the vertical stationary furnace has all of the properties of the horizontal flash furnace. It is a short heating time furnace and, like the horizontal flash furnace, it requires a dead glass if the production of very light and fine material is to be avoided.

**Short Cocurrent Rotary:** The short cocurrent rotary (fig. 6) is essentially a rotating flash furnace. It is generally from two to four diameters in length and inclined about 1:8. The raw material is fed in at the upper end, usually dropping right through the flame. The hot gases carry the light fine product out of the furnace and it is handled in a cyclone and blower system similar to that used in stationary flash furnaces, but this type of furnace generally produces a fairly large percentage of coarse, heavier material which is not carried out in the gases but drops out at the discharge end of the rotary into a

mechanical conveyor which carries it to screens and bins. This coarse product is not caused by the design of the furnace but rather by the use of a coarser feed than is used in stationary furnaces. The retention time in the furnace is not sufficient to expand the coarse particles fully, hence the coarse aggregate made in the short cocurrent rotary is generally heavy, cracked, and discolored.

Due to the slightly longer retention time, the fuel efficiency in a short cocurrent rotary should be a little better than that of the stationary furnaces. Although high production rates are claimed for small furnaces of the cocurrent variety, the installation cost for a given production rate is bound to be higher for a rotary than for a stationary furnace.

**Long Cocurrent Rotary:** The long cocurrent rotary (fig. 7) is a variation of the short cocurrent rotary. Its length is usually about 8 diam although there is one furnace 13 diam long in production. This extra length allows the operator to play the flame down the tube so that the hot zone is some distance away from the feed end of the furnace. Then, if the perlite is dropped in under the flame, some preheating can be had before the perlite reaches the hot zone. This preheating does not utilize the exhaust gases, therefore fuel efficiency on a long cocurrent rotary will not be much better than on a short cocurrent rotary.

The longer hot zone usually available in the long cocurrent rotary allows the coarse particles of perl-

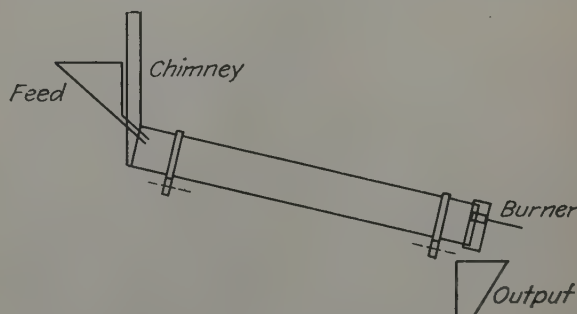


Fig. 8—Countercurrent rotary.

ite to be more thoroughly expanded, but the coarse expanded particles are nevertheless heavier than the fines. In the stationary flash and short cocurrent rotary furnaces, the fines expand rapidly and are swept out of the hot zone fast enough to prevent their sticking to the walls. But in the long cocurrent rotary where the perlite is fed under the flame, the fine component gets up to a softening temperature rapidly because of the high furnace temperatures which must be employed to expand the coarser particles in the short time available. These fine melted particles stick to the walls of the tube and start the formation of a ring, which gets hotter and hotter and finally picks up large particles as well. One remedy for this condition is to drop the perlite through the flame, as in a short rotary, but then most of the advantage of the length of the furnace is lost. Another alternative is to remove the finer component from the feed.

**Long Countercurrent Rotary:** The countercurrent rotary (fig. 8) is a conventional rotary furnace and



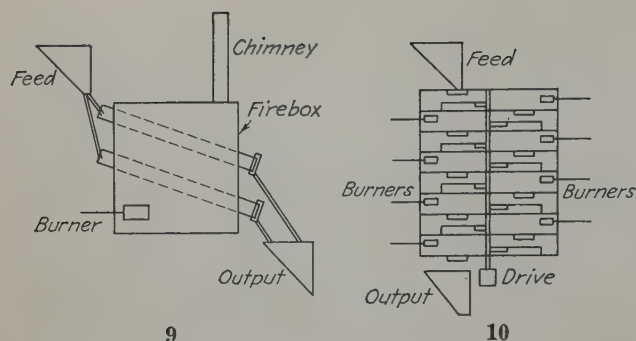


Fig. 9—Indirectly-fired rotary.

Fig. 10—Multiple-hearth furnace.

is operated in the conventional way. It is usually refractory-lined for more than 4 diam of its length, and set at a slope of 1:8. The burner is at the bottom of the tube, and the hot gases travel up the tube by convection and out a chimney at the top. The perlite is fed in at the top end, and is gradually heated by the exhaust gases as it rolls down the tube toward the hot zone, which is at the lower end. It expands in the hot zone and drops out at the bottom of the furnace. The exhaust gases at the top end are not over 800°F and their velocity is generally kept as low as possible to give good fuel efficiency and to prevent carrying any of the expanded perlite back up the furnace. In this furnace, heating time is about the same as retention time, and this time can be adjusted by changing the slope or the rotation speed of the tube, or both. Closer temperature control is possible with this type of furnace than with short heating time furnaces because the long heating time available obviates the necessity for furnace temperatures much in excess of the expansion temperature of the perlite. Due to ease of adjustment of retention time and good temperature regulation, the size distribution and density of the product may be closely controlled. Fusion of perlite to the walls is easily prevented, and the long retention time (20 sec up to several minutes) keeps the fines from flying apart into light dust. No dust separators are needed.

A big advantage of the countercurrent type of operation is its good fuel efficiency. As the hot gases travel up the tube from the hot zone, they transfer heat to the colder particles of perlite, so that by the time the perlite arrives at the hottest zone, it has already been preheated to almost the final temperature. At the same time, the hot gases which leave the tube at the upper end have transferred much of their available heat to the bed of perlite. Fuel consumption as low as 2½ million Btu per ton has

been reported with a countercurrent rotary of 8 diam long, and the efficiency could be improved a little if the tube were longer.

A disadvantage of the countercurrent rotary is that high expansion ratios can be obtained only with very lively glasses. With a 1 min heating time only relatively few types of perlite can be expanded to ratios greater than ten to one. Another disadvantage is the low production capacity for furnaces of a given size and cost. A countercurrent furnace 30 ft long by 3½ ft id is only good for two thirds of a ton per hour. The best use for a countercurrent rotary is in the expansion of perlite to pieces larger than ⅛ in. It is possible to expand some very lively glasses into pieces up to 2 in. long with a density of less than 10 lb per cubic foot.

**Indirectly-fired Rotary:** The indirectly-fired rotary (fig. 9) is an unlined tube made of heat-resisting but heat-conducting material, usually stainless steel. It is generally over 4 diam in length and capable of variable inclination. It is enclosed in a firebox which supplies the heat for one or more such tubes. Each tube has its own feeder, and as the perlite rolls down through the tube it is heated by conduction through the walls of the tube. Fusion of the perlite to the walls of the tube is avoided because none of the particles is overheated as is the case when perlite is fed directly into an open flame. Lack of extreme temperatures and fairly long heating times prevent the formation of dust. Retention time can be varied, and usually runs between 5 and 60 sec. Poor fuel efficiency should be expected in an indirectly-fired furnace, and difficulty with the steel tube will result if too high a temperature is reached. The installation cost of an indirectly-fired furnace is even higher than for a countercurrent rotary for the same production rate, and very lively glasses must be used to provide good expansion ratios.

**Multiple-hearth Furnace:** With the multiple-hearth Wedge-Herreshof ore-roaster, (fig. 10) perlite is fed in at the top of a series of hearths arranged one above the other, and progresses from one level down to the next with the aid of scrapers. Flames on the various levels provide any desired time-temperature curve, giving excellent expansion of coarse pieces of lively perlite. This type of furnace has all the advantages and disadvantages of the countercurrent rotary but in addition its installation cost is more for a given production rate.

## Summary

Table I summarizes this paper and the authors' opinions on each of eight furnaces with regard to nine operating characteristics.

Table I—Summary of Furnaces and Operating Characteristics

Detailed Evaluation of Performance of Various Types of Furnaces	Horizontal Flash	Horizontal Flash with Pre-Heater	Vertical Stationary	Short Cocurrent Rotary	Long Cocurrent Rotary	Countercurrent Rotary	Indirectly Fired Rotary	Multiple Hearth
Fuel efficiency	Fair	Fair	Fair	Fair	Fair	Good	Poor	Good
Freedom from waste dust production	Poor	Fair	Poor	Poor	Fair	Good	Good	Good
Production for a given construction cost	Good	Fair	Fair	Fair	Fair	Poor	Poor	Poor
Ability to make light plus 8 mesh product	Poor	Poor	Poor	Poor	Fair	Good	Good	Good
Ability to make light minus 8 mesh product	Good	Good	Good	Good	Good	Poor	Poor	Poor
Ability to expand low temperature ores	Poor	Fair	Poor	Poor	Fair	Good	Good	Good
Ability to expand high temperature ores	Good	Good	Good	Good	Good	Poor	Poor	Poor
Ability to avoid fusion of perlite to walls of furnace	Fair	Fair	Poor	Fair	Fair	Good	Good	Good
Control over size distribution and degree of expansion	Poor	Good	Poor	Poor	Fair	Good	Good	Good



## Geology of the Potash Deposits

of Germany, France, and Spain

Permian salt measures carry extensive lenses of soluble potash salts in north central Germany. Potash deposits of Oligocene age are found in the Upper Rhine Graben of Alsace (France), and in the Catalanian Basin of Spain. These deposits are found in a wide range of structural settings. Reserves are large, and the production of potash salts is an important phase in the economy of each of these countries.

by J. P. Smith

**D**URING the winter of 1946 to 1947 potash operations in Germany, France and Spain were visited by the author. The U. S. Department of Commerce, through its Field Intelligence Agency Technical, sponsored the inspection of German properties by representatives of the American Potash Industry.

Vast deposits of potash salts occur in the above-mentioned countries, and the production and marketing of these salts are important in their respective economies.

### Germany

An area of some ten thousand square miles of north central Germany is underlain by lenses of potash salts interbedded with greater thicknesses of sodium chloride or common salt. These salt measures are relict deposits from the evaporation of an arm of the Zechstein (Permian) Sea which formerly covered northern Europe. During the peak year of 1940, Germany produced salts containing 1,860,000 tons of  $K_2O$  and, next to coal, potash is its most important mineral raw material.

The German deposits are found in the synclinal areas flanking the Flechtingen, Hartz Mountain and Thuringian uplifts, and, to the north they lie beneath the plains of Hannover. The regional dip of the salt measures is to the north, hence, lenses of potash found at depths of 500 to 1000 m\* in the

Two general depositional provinces of potash concentrations are recognized: (1) the main basin, and (2) the subsidiary basin. The main basin embraces the producing districts of Stassfurt, Sud Hartz, Halle, and Hannover-Braunschweig. From

---

J. P. SMITH, Member AIME, is Chief Geologist, U. S. Potash Co., Carlsbad, New Mexico.  
New York Meeting, February 1948.

TP 2621 H. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950.  
Manuscript received May 26, 1948.

---

40 to 45 pct of the German production comes from this basin. The Werra-Fulda district, producer of 50 pct or more of the German potash, lies in the subsidiary basin.

**Salt Stratigraphy:** Five commercial potash zones or beds are recognized. The three beds present in the main basin are: The Stassfurt bed that occurs near the top of the older salt, and the Ronnenberg and Riedel beds in the younger salt occurring 200 to 300 m, respectively, above the Stassfurt bed. The Stassfurt bed is found over the entire main basin, while the Ronnenberg and Riedel are restricted to the Hannover area.

Two potash beds occur in the Werra basin. These are the Thuringian, or lower bed and the Hessen or upper potash bed that occurs 60 m above. These beds lie 210 and 150 m, respectively, below the position of the Stassfurt bed, and are restricted to the Werra basin. The upper bed is 2.5 to 11.5 m in thickness, and the principal ores are hartsalz with varying thicknesses of carnallite. The lower bed averages 3 m in thickness, and contains hartsalz. The upper bed is presently being mined, and the

---

\* For conversion of meters to feet see table at end of paper.

South Hartz or Stassfurt Districts should be expected at depths exceeding 3000 m in the vicinity of Hannover. However, local doming of the salt has brought the overlying potash beds upward, and, at the flanks of certain domes, the potash beds are 100 m and less beneath the surface.



lower zone is held in reserve. The younger salt is missing over the Werra basin.

More than thirty saline minerals have been identified from the German section of evaporites. The commercial potash beds are admixtures of sodium chloride (common salt) with varying proportions of the potash salts. These are termed raw salts. The principal raw salts mined are: (1) Sylvinite—sylvite ( $\text{KCl}$ ) and halite (sodium chloride), this raw salt averages 10 to 20 pct  $\text{K}_2\text{O}$ . (2) Hartsalz consists of sylvite, halite, kieserite (magnesium sulphate) and anhydrite (anhydrous calcium sulphate) and averages 10 to 15 pct  $\text{K}_2\text{O}$ . (3) Carnallite raw salts consists of carnallite (potassium and magnesium chloride) and halite, and averages 8 to 12 pct  $\text{K}_2\text{O}$ . The raw salts in the German mines are white, gray, or buff in color, and are distinguished with difficulty from the enclosing salt. Lateral gradation from sylvite to hartsalz to carnallite may suggest changes from marginal to basinward facies of deposition.

The deposits of the Werra district are relatively flat lying, and are characterized by regularity of thickness, tenor of  $\text{K}_2\text{O}$ , and distribution. Locally, these beds are cut by sills and dikes of basic igneous rocks.

The deposits of the Hannover district are steeply

and pillar system. About 60 pct of the ore is removed without backfilling. The mines visited showed little evidence of subsidence, despite the fact that the pillars were supporting from 1200 to 1600 ft of overburden. For an excellent description of the details of mining, you are referred to a paper by East.<sup>1</sup>

The Hannover deposits are mined by stoping and backfilling. Vertical hoisting shafts are sunk in the salt core, and drifts driven to the flanks. Much underground prospect drilling is necessary in following these beds. Because of high rock temperatures and pressures, little mining is done below depths of 800 m. Ground water conditions have discouraged mining operations at depths of less than 400 m, as several mines have been lost by flooding.

Waste salt disposal is a problem at the German potash plants. The law prohibits raising the chloride content of the streams above a concentration of 2000 parts per million, hence limited quantities of salt can be dumped into the streams. In the Werra district, waste brines are injected, through boreholes, into the laminated dolomite (Permian) which overlies the salt measures. From 1000 to 1300 gal per min of brine is disposed of in this manner at an individual plant. In the Hannover district, there is no porous formation to receive the waste

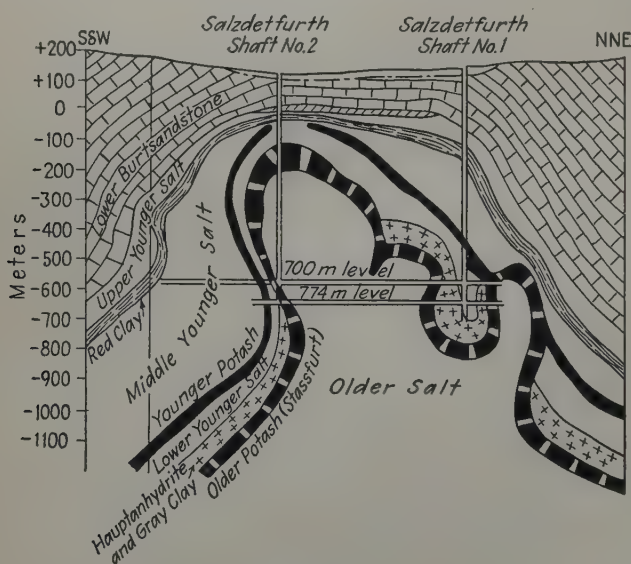


Fig. 1—Profile of salt stock, Bad Salzdetfurth, Germany. Cross-section showing deformation of salt and potash beds.

(After E. Fulda, 1935)

dipping, irregular in extent, thickness, and grade. In the upbuilding of the Hannoverian salt stocks, the more plastic carnallite, sylvite, and hartsalz have been squeezed into available voids. These potash salts are concentrated in the synclinal lobes on the flanks of the salt core. All variations, from conformable salt anticlines to extreme diapir or piercement type doming are represented in the German salt stocks. An example of piercement, on a relatively small scale, was seen at the Bad Salzdetfurth salt stock (fig. 1). On the northeast flank of this dome, the hauptanhydrite, a competent member of 75 m thickness, has been breached, the underlying Stassfurt (carnallite) bed, has been squeezed upward, piercing 100 m of salt, and has been brought into direct contact with the Ronnenberg (sylvinite) bed.

The flat lying Werra deposits are mined by room

brines, here the waste salt is partially dried, and sent underground for backfill or stock piled on the surface.

Prior to 1921 there were some 220 operating mines in Germany. In recent years, all but the low cost operations have been shut down. At present, only 25 mines are being operated. Under the present occupational zoning, the productive areas are divided as follows: Russian Zone, 61 pct; British Zone, 22 pct; American Zone, 14 pct; and French Zone, 3 pct.

The 10 most productive mines are in the Werra and South Hartz districts. Two of these are in the American Zone, and eight are in the Russian Zone. Kaiseroda (Russian Zone) is the largest mine, having a hoisting capacity of 7000 tons of raw salts per day. In terms of total  $\text{K}_2\text{O}$  processed, it compares with the individual mining operations in New

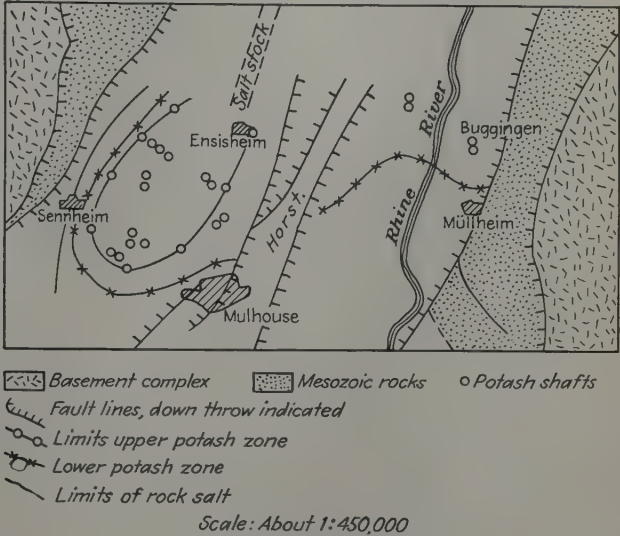


Mexico. The two mines in the American Zone are Wintershall and Hattorf. Both mines have hoisting capacities of 5000 to 6000 tons of raw salts per day. In December 1946, these plants were operating at 45 pct capacity. Hartsalz is the principal mineral exploited at these plants. The hartsalz averages 3 m in thickness and 11 pct  $K_2O$ . A carnallite bed of varying thickness, directly overlies the hartsalz at Wintershall. Metallic magnesium was formerly produced by electrolysis of molten carnallite at the Wintershall plant. During peak production years the output of the Wintershall operation was of the order of 6000 to 7000 tons of magnesium metal per annum.

The total reserves of potash in Germany have Buggingen (Baden Province, Germany) on the east side of the Rhine. The Buggingen deposits are the only known Tertiary potash deposits in Germany and are the richest sylvite beds in that country.

The Alsatian potash deposits are of (Sannoisien) Lower Oligocene Age, are contemporaneous with the Catalanian (Spanish) deposits and are much younger than the Permian deposits of New Mexico, Germany, and Russia. They occupy a fault basin or "graben" in the upper Rhine Valley. This part of the Rhine Valley is very flat, and is faced by the steep fault scarps of the Vosges Mountains in France and the Schwarzwald (Black Forest) of Germany. These mountains are capped with much more than ancient rocks (Jurassic), that represent remnants

Fig. 2—Map of Alsatian potash basin.  
(After F. Lotze<sup>2</sup>.)



been estimated as being of the order of 20,000,000,-000 tons of  $K_2O$ . This is sufficient to last for 1000 years at the former peak rate of production. Much of this reserve lies at depths too great to be economically exploited by present day techniques.

The solution and crystallization process of refining is used throughout Germany. Since the war, considerable experimental work on the flotation treatment of potash salts has been done. Potassium chloride containing 40 to 50 pct and 60 pct  $K_2O$  are the principal processed products. Sulphate of potash is an important product where hartsalz is mined. Many of the refineries produce bromine in limited quantities as a byproduct. Shortages of food, coal, equipment, and trained labor, with inadequate transportation facilities are responsible for holding present production to 60 pct of the capacity.

France

The Alsatian potash basin (fig. 2) is situated immediately north and east of the important manufacturing city of Mulhouse, in the upper Rhine Valley. The production area is oval in shape, and occupies an area of approximately 75 square miles. Two workable beds of sylvite raw salts are found over much of this area. The upper bed averages about 1.5 m in thickness and covers about half of the productive area. The lower bed averages 4 to 4.2 m in thickness and covers an area of about 65 square miles. An extension of the lower bed is found near

of the anticlinal arch that once was continuous across the Rhine Valley. The present valley is floored by Quaternary sands and gravels. These are underlain by Tertiary rocks that carry rock salt and the potash beds.

The section at the Marie Louise Shaft follows:

Quaternary	Valley Fill	0 to 22.3 m
Upper Oligocene	Limy Sand	22.3 to 77.4 m
Chattien	Fish Shale	
	Marl	
Middle Oligocene	Gypsum	77.4 to 236.4 m
Stampien	Limnae Zone	
	Dolomite	
	Anhydrite and	
	Rock Salt	
Lower Oligocene	Upper Bituminous	236.4 to 700 m
Sannoisien	Zone with Rock Salt	
	and Potash	
	Fossiliferous shale	
	Lower Bituminous	
	Shale and Rock Salt	
	Conglomerate Zone	
	with Rock Salt	

More than 200 borings have been made in the general area, only four of which have been drilled through the Oligocene. The greatest thickness of Oligocene penetrated was in the boring D.P.II near Schoensteinbach. This well entered the Sannoisien at 40 m and bottomed at 1650 m in rocks carrying



salt in the basal Sannoisien. Shales and marls, rich in marine-type fossils are found intercalated with rock salt short distances above and below the potash beds. The area has had a complex depositional and structural history.

The potash beds are prominently banded. An exposed face shows alternating bands, more or less wavy, of white and red sylvite salts. The banding is accentuated by paper thin shales that separate the varicolored salts. The top and bottom of the workable potash zones are demarked by prominent dark shale beds. The banding and layering aids the breaking and extraction of the ore. It also adds some element of hazard, as rock falls are common.

The saucer like basin is roughly oval, bulging to the south, and restricted as a relatively narrow neck to the north. The dips are to the center of the basin from east and west, with gentle overall tilting to the north. The north end of the basin is further complicated by a narrow north-south trending salt anticline or "Sattel." The potash beds are not continuous across this saddle.

The regularity and gentle attitude of the beds favors large scale exploitation. Mining is done by longwall and modified room and pillar systems.

Fifteen shafts and six refineries were in operation in early 1947. For the current year production equaled or exceeded that of the peak years 1941 to 1943, and was of the order of 650,000 tons of  $K_2O$ . Processing by solution and crystallization is used by all the refineries. Flotation equipment has been purchased, and future plans call for the use of this method in some of the plants. Potassium chloride of 50 and 60 pct  $K_2O$  grades are the principal products of the refineries. Limited amounts of bromine are produced as a byproduct. Reserves of the Alsatian field, including the deposits of Buggingen, are of the order of 300,000,000 tons of  $K_2O$ .

The potash deposit at Buggingen (Baden), Germany, is an eastward offshoot of the main Alsatian basin. The workable zone covers an area of 0.56 square miles, and varies from 0.5 to 4.0 m in thickness. The principal bed is exploited through two vertical shafts, and the main level of mining is 800 m below the surface. These deposits dip from 18 to 20° to the northwest, and are mined by modified room and pillar system. The production is small, as the hoisting capacity is of the order of 200 tons of raw salts per day.

Lenses of potash salt, of Triassic age, are found in southwestern France near Bayonne and Dax. More than thirty boreholes have been sunk in this area. The salt and potash beds are highly contorted. A record of one borehole south east of Dax shows a 7 m zone (from 762 m to 769 m) that averages 10.77 pct  $K_2O$ . The small mining operation in this district has recently been discontinued because of the erratic occurrence of the beds and the high cost of operation.

### Spain

The principal Spanish potash basin is some forty miles north and west of the Mediterranean seaport of Barcelona. The potash beds are of lower Oligocene age and occur near the top of thick salt measures. Outcrops of Eocene rocks in the vicinity of the towns of Vich, on the east, Balaquer to the west, Isona and Berga to the north, and Manresa on the

south delimit this basin. The total area of this basin is some 1200 square miles, but the workable potash beds are confined to the depositional center. Several east-west trending anticlinal axes extend across this basin and prospecting has been confined to these anticlines. Claims covering 350 to 400 square miles have been granted to concessionaires.

Overlying the potash beds are varying thicknesses of rock salt and gypsum, followed upward by marls, clays, and siltstones. In the deeper part of the basin the potash beds are overlain by 150 to 200 m of salt, in the shallower portions of the basin, the potash beds occur near the top of the salt measures. A basal anhydrite member separates the evaporates from the fossiliferous marls and limestones of the underlying Eocene rocks.

This interior basin is bounded on the west by the midland plateau of Spain. To the east it is bounded by the rugged coast range, an upfaulted mass of Mesozoic and Paleozoic rocks, and to the north it is bounded by the upfolded older rocks of the Pyrenes Mountains. According to European geologists, the forces that gave rise to the building of the Pyrenes Mountains began in the Eocene and reached a maximum during Oligocene times. During Oligocene, the Catalanian basin was occupied by a gulf or continental sea connected by narrow waterways to the ocean. The evaporation of these waters has produced the salts of the present basin. Salt and potash deposits of Triassic age are found near Bilbao and elsewhere in Spain. It has been suggested that leachings from these ancient salt measures may have contributed increments of potash to the brines that produced the evaporites.

Doming and faulting have brought the salt to the surface at Cardona. The famous salt mountain represents a local eruption superimposed on the anticlinal axis. Thrust faulting has brought the potash beds to 50 m below the surface at Suria. Anticlinal folding has brought the potash beds upward at Sallent.

A study of the Spanish deposits is a classic example of the behavior of plastic rocks that have been subjected to various intensities of folding. There are numerous examples of gliding and wrinkling of the more plastic carnallite and sylvinite beds. Sr. Marin has suggested that the potash salts are not only plastic but may also be considered as semi-dynamic or catalytic, that is, where present in considerable thickness they may serve to localize or initiate structural activity.

Two zones of soluble potash minerals are recognized in the moderately disturbed section at Sallent. At the top is a carnallite zone 8 to 10 m thick, carrying from 40 to 60 pct of carnallite. Directly underlying the carnallite is the sylvite zone. This consists of two beds of sylvite given in descending order as follows:

	Thickness	$K_2O$ , Pct
Bed B sylvinitic	0.8 m	23.0
Salt Bank	2.5 m	
Bed A sylvinitic	2.7 m	23.8

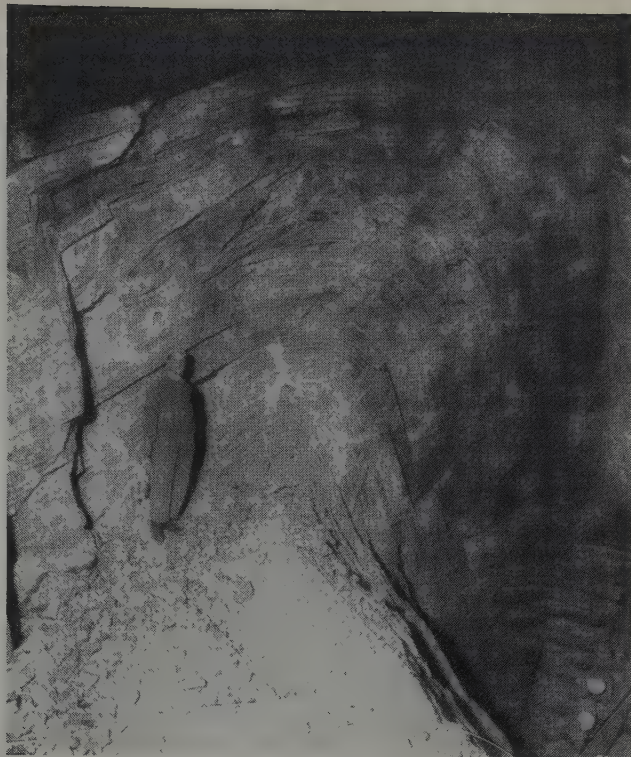
The potash salts are red in color and are prominently banded.

Folding at Sallent is moderate and is of open type (fig. 3). Folding at Suria and Cardona is intense, and is complicated by overturned folds, shearing,



**Fig. 3—Interior of mine showing local folding in sylvite.**

(Courtesy of Potasas Ibericas, S.A., Sallent, Spain.)



and faulting. The potash beds at Suria and Cardona have been thickened by duplication of beds. Local impoverishment and enrichment is characteristic. Mining at Sallent is by modified room and pillar system with backfill. The salt bank, referred to above, is first mined and used for pack walls, the upper sylvite bench is then taken, and, finally the lower bench. Refinery tailings and barren salt are used for backfill. At Cardona and Suria variations of stope mining with backfill are employed.

Three companies are in operation at present with total daily production of 2900 tons of raw salts. These are Potasas Ibericas (Sallent), Union Espanola de Explosivos (Cardona), and Minas de Potasa de Suria (Suria). A fourth operation, Explotaciones Potasicas (at Balsareny) was scheduled to begin hoisting by the end of 1947. The total annual production of processed salts is of the order of 150,000 tons of contained  $K_2O$ . Shortages of coal and equipment, inadequate transportation and nominal domestic consumption of potash have served to hold

down the production of potash. With minor changes, the Spanish operators could double the present output.

The combined reserves of these four concessions has been estimated at 143,000,000 tons of  $K_2O$ . A total of 1,800,000,000 tons of  $K_2O$  is claimed for the entire basin.

Solution and crystallization is the important method of refining used throughout Spain. Within the past six months, a flotation plant capable of producing 25 tons per day has been operated by Potasicas Ibericas at Sallent.

#### References

- <sup>1</sup> J. H. East: Potash Mining in Germany. Bur. of Mines I. C. 7405 (1945).
- <sup>2</sup> Franz Lotze: Steinsalz and Kalizalze Geologie. Verlag von Gebruder Borntraeger, p. 606, 1938, Berlin.
- <sup>3</sup> D. Augustin Marin: Historia de una Molecula de Potasa. Notas y Comunicaciones del Instituto Geologico y Minero de Espana, Ano V. Numero 5. Graficas Reunidos, S. A., 1935, Madrid.
- <sup>4</sup> Vincoslav Maikovsky: Contribution a L'Etude Paleontologique et Stratigraphique de Bassin Potassique D'Alsace. Clermont-Ferrand, 1941, Paris.
- <sup>5</sup> Ernst Sturmfels: Das Kalizalzlager von Buggingen (Sudbaden) Sonderdruck aus dem Neuen Jahrbuch fur Mineralogie etc. Abt. A. Bd. 78, 1943, S-131-216.
- <sup>6</sup> Hoyt S. Gale: The Potash Deposits of Alsace. U. S. Geol. Sur. Bull. 715, (1920) and The Potash Deposits of Spain. U. S. Geol. Sur. Bull. 715 (1920).
- <sup>7</sup> G. S. Rice and J. A. Davis: Potash Mining in Germany and France. U. S. Dept. of Commerce Bureau of Mines Bull. 274 (1927).
- <sup>8</sup> R. W. Mumford, M. H. McAllister, J. P. Smith, A. M. Into, and G. H. Gloss: Mining and Refining of Potash in the American and British Zones of Germany. U. S. Dept. of Commerce F.I.A.T. Final Report 1045. March 1947.

#### Conversion of Meters to Feet (rounded off to nearest tenth)

Meter	Feet	Meter	Feet
0.5	2	100	328
1.5	5	150	492
2.5	8	200	656
3	10	210	689
4	13	236.4	776
4.2	14	300	984
7	23	400	1,312
8	26	500	1,641
10	33	700	2,297
11.5	38	762	2,400
22.3	73	769	2,523
40	131	800	2,625
50	164	1,000	3,281
60	197	1,650	5,414
75	246	3,000	9,843
77.4	254		



# California Talcs

by Lauren A. Wright

The principal talc deposits in California are in a 200-mile belt paralleling the state's eastern border. The southernmost deposits represent selective alteration of early pre-Cambrian (?) carbonate strata, and are associated with granitic rocks. Deposits in the Death Valley area are at or near contacts between late pre-Cambrian carbonate rocks and diabase sills. Those in the Inyo Range area are chiefly alterations of Paleozoic carbonate rocks and silica rocks. Most of the state's talc output is used in ceramics.

SINCE the early nineteen-thirties the production of talc\* in California has increased five-fold to a yield in 1947 of about 76,000 tons (fig. 1); conse-

\* Unless otherwise qualified, the term "talc" in this paper will imply a mixture of the pure talc with such related minerals as commonly occur in the commercial material.

quently the state is now one of the nation's three principal talc producers. The 1947 output, which represented about one-sixth of the national total for that year, was approximately equal to production from Vermont, and was second in value to that from New York. New York has an annual output of about 130,000 tons.

Three factors have been mainly responsible for the expansion of talc production in California. (1) Technical advances and innovations, particularly in the ceramic industry, have increased the demand for certain California talcs. The nation's high-quality

the industry, mainly through demand for the urgently needed steatite. Fibrous talc, used as a paint extender by the armed forces, also was produced in quantity. (3) Expanding western industry has continued to draw heavily on California talc deposits, and has extended the upward production trend into the postwar period.

Considerably more than half of California's current talc output is used as ceramic raw material. The paint, rubber, paper, and textile industries are the more important consumers of the remainder. Smaller amounts are used as insecticide carriers, and in cosmetics, and various minor industrial applications. During World War II the principal steatite deposits were studied in detail by members of the U. S. Geological Survey; other sources of commercial talc in the state have in the past received but scant geologic attention. The following discussion is essentially a preliminary report based on research currently in progress by the California Division of Mines, and includes a brief outline of previous steatite studies by the U. S. Geological Survey.

**Distribution of California Talc Deposits:** The principal talc resources of both California and Nevada are confined to an elongate belt that extends from the vicinity of Baker, in north-central San Bernardino County, northwestward to include the southern Death Valley region, the Inyo Range, and the Palmetto-Oasis district of western Nevada (fig. 2). This belt, of which all but the northernmost tip lies in California, is approximately 200 miles long and 30 miles in average width.

Included in the belt are four contrasting areas. The deposits within each area are geologically similar and yield talcs that have characteristic properties and applications. Three of these local "talc provinces" lie within California's portion of the talc-

LAUREN A. WRIGHT, Junior Member AIME, is Associate Geologist, California Division of Mines, San Francisco, Calif.

Los Angeles Meeting, October 1948. San Francisco Meeting, February 1949.

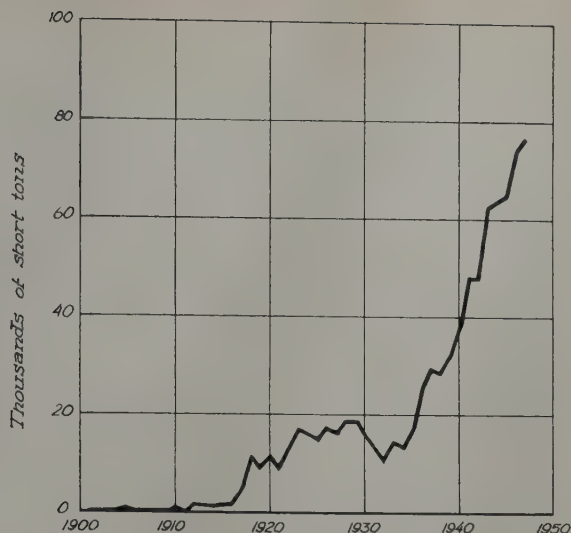
TP 2622 H. Discussion of this paper (2 copies) may be sent to Transactions AIME before Feb. 28, 1950. Manuscripts received Dec. 7, 1948; revision received Aug. 29, 1949.

This paper is Contribution No. 508 of the Division of Geological Sciences, California Institute of Technology, Pasadena, Calif.

ity steatite, used in the manufacture of high-frequency insulators, is obtained largely from sources within the state, as are the tremolitic talcs that are used as the principal ingredients in nearly all of the floor and wall tile manufactured on the Pacific Coast. (2) World War II brought added impetus to



Fig. 1—Talc production in California.



bearing belt. The southernmost is centered north and east of Silver Lake, in north-central San Bernardino County. A second, immediately to the northwest, includes the southern Death Valley area and the Kingston Range, and lies in both San Bernardino and Inyo Counties. Deposits of the third province are best represented by a group of mines east of Keeler, in Inyo County, but similar deposits are scattered on both sides of the Inyo Range to the north. These three areas are here referred to as the Silver Lake, Death Valley, and Inyo Range talc provinces.

The fourth and northernmost province is in Esmeralda County, Nevada, east of the California-Nevada line. The talc deposits are confined to a relatively small area that is crossed by the highway connecting Oasis, California, with the old Palmetto mining district, Nevada; the name "Oasis-Palmetto talc province" has been applied to the area. Most of these deposits are alterations of dolomite; the talcs are used chiefly as cosmetic material and are milled in southern California.

A second talc-bearing belt comparable in size to the one described above and with a similar trend lies within the western foothills of the Sierra Nevada. For many years talc and talcose material have been produced under the name "soapstone" from numerous small operations in this area. The output has been consumed largely in the manufacture of roofing material, and has been produced in much smaller quantity than the talcs of eastern California. A small talc and soapstone production has also been obtained from other, widely scattered California deposits.

The history of California's talc industry to date has evolved largely around three properties, each in a different talc province of the eastern belt. These, from south to north, are the Silver Lake, Western, and the Talc City mines (fig. 2). All were opened during the period 1915 to 1917, and since that time they have yielded more than 500,000 tons of talc, or approximately two-thirds of California's total talc and soapstone production. According to records of the California Division of Mines, the remaining tonnage has been obtained from 53 other

properties, 30 of which are in the eastern belt, 17 in the western Sierran foothills, and 6 at scattered points throughout the state.

At present mining is centered in the eastern belt, where 11 properties, including the three previously mentioned, have been in relatively continuous operation; 13 others were worked earlier or have been intermittent operations. At least three deposits are scheduled to be worked for the first time within the next year or two; and the exploitation of still others must await more favorable economic conditions than now exist. Space does not permit discussion of each property here, but the following brief descriptions of the Silver Lake, Western and Talc City mines should serve to outline the geological features that are characteristic of most in the respective provinces.

#### Descriptions of Representative Mines

**Silver Lake Mine: Location and Production History:** The Silver Lake mine workings have been developed in a group of elongate talc-tremolite lenses that trend eastward. These are exposed along a narrow, two-mile zone in the hills east of the Silver Lake playa and about ten miles north-northwest of Baker, San Bernardino County. Since 1918, when the property was first worked, a total of approximately 140,000 tons of talc and talc-tremolite rock has been produced. Early mining was confined largely to the western 5000-ft portion of the zone. More recently, a talc body at the easternmost end of the zone was outlined by diamond drilling, and subsequently this body has yielded about 30,000 tons of commercial material.

**Geology:** The principal talc-tremolite concentrations lie within a relatively thin, but persistent diopside- and feldspar-rich member of a series of highly metamorphosed sedimentary rocks (fig. 3). The series has been briefly described by W. J. Miller<sup>1</sup> who has tentatively dated it as early pre-Cambrian. The more prominent rock types include vitreous quartzite, quartz-muscovite, and quartz-biotite schists, limestone, and granulite. The diopside-feldspar rock which encloses the talc-tremolite bodies also includes phlogopite, quartz, calcite, and



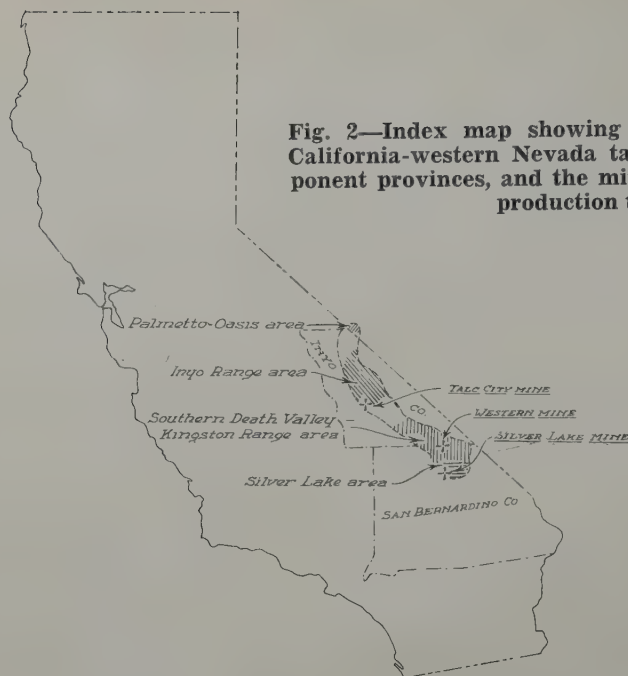


Fig. 2—Index map showing location of the eastern California-western Nevada talc-bearing belt, its component provinces, and the mines that have the largest production to date.

garnet as locally prominent mineral constituents. These minerals, in differing proportions and textural relations, form granulitic, schistose, gneissic, and quartzitic rock types as variants of the member.

In the western portion of the talc-bearing zone the metasediments dip moderately to steeply southward, and are successively offset by a series of cross faults. Toward the eastern end of the zone the metasediments are, in general, intricately faulted. The easternmost talc bodies are in a structural block characterized by gentle folds, but bordered on the west, south, and east by highly brecciated and faulted rocks.

The relatively simple structural features that characterize most of the talc-bearing zone, however, have been greatly complicated in detail by a wide variety of igneous rocks in the form of dikes, sills, and irregular masses. These rocks range in composition from mafic to felsic. Quartz-diorite is the most widespread type but lamprophyric rocks, in both dikes and larger, irregular masses, also are common. Small pegmatitic bodies are abundant, especially in the diopside- and feldspar-rich unit that encloses the talc-tremolite masses.

The main talc-tremolite bodies in the western part of the area are remarkably uniform in their internal structure, as well as in their stratigraphic position. They represent a nearly complete selective alteration of carbonate rocks, presumably of dolomitic composition. A persistent massive limestone in the associated metasedimentary sequence is barren of commercial talc or tremolite, but does contain abundant, though dispersed, olivine, chondrodite (?) and talc. Diopside has locally replaced this limestone along fractures.

The commercially exploitable talc-tremolite masses are generally 10 to 15 ft thick and as much

as 800 ft long. Two of these commonly parallel each other, and are separated by a 5 to 10 ft thickness of diopside wall rock. All the talc-tremolite masses in the western area dip about  $65^\circ$  southward. Some are terminated by cross faults, but more commonly they lense along the strike into talc- and tremolite-poor rock or end abruptly against granitic rocks. The contacts separating talc-tremolite rock from diopside- and feldspar-rich wall rock are very sharp in most places. The steeply-dipping deposits have proved disappointingly shallow; the longest mass, which is 800 ft long, was bottomed in a granitic rock at a depth of less than 200 ft.

Tremolite is a major constituent of most of the material currently marketed, and is commonly more abundant than the mineral talc. This is particularly true of the rock that occurs near the hanging walls of the steeply-dipping bodies. Most of the tremolite-poor talc concentrations in these deposits are adjacent to the footwall and are only several feet thick. Smaller and less continuous seams of the mineral talc occur within the tremolite-rich material and are locally adjacent to the hanging wall. The greatest thickness of relatively pure talc, however, was removed from bodies in the gently folded metasediments to the east. Here this talc is reported to have had a maximum thickness of 40 ft.

**Mining Methods:** Mining operations at Silver Lake have followed a relatively simple pattern. The steeply-dipping deposits have been developed at the six localities where surface exposures of talc-tremolite rock were most extensive. In common procedure, a shaft is sunk to the keel of the deposit. Drifts are driven to the along-strike limits of the ore body at several levels, and the talc is removed by overhand stopping and hoisted to the surface.



Much material has, however, been removed from the largest body by tramming from an adit. The ground is unusually solid and little timbering is required (fig. 4). In the gently dipping deposits to the east a modified room and pillar method of mining was employed and, recently, a considerable tonnage has been removed from here by open cut methods.

**Uses:** The earliest production, intended mainly for the cosmetic market, was from the tremolite-poor, talc-rich portions of the deposits. Later, the paint and rubber industries were the principal consumers. Currently most of the ore is marketed as a ceramic material. As tremolite has proved a desirable wall-tile ingredient, the high-tremolite rock, formerly avoided, has become commercial.

**Other Active Properties in Silver Lake Area:** The two other active mines in the Silver Lake province are in the vicinity of Yucca Grove, on the Baker-Las Vegas highway, and are 12 airline miles north-northeast of the Silver Lake mine. These, the Calmasil and Pomona mines, were opened within the past ten years. Here, as at the Silver Lake mine, talc is an alteration of carbonate strata in a terrane of steeply dipping metasedimentary rocks. Here, also, granitic rocks are extensively exposed. The talcs, too, are tremolitic and are marketed principally as ceramic materials.

**The Western Mine: Location and Production History:** The Western talc mine is about 25 miles due north of Silver Lake and seven miles east-southeast of Tecopa, a small settlement in Inyo County. The mine workings are immediately south of the northern San Bernardino County line. Production, which dates from 1917, now totals well over 150,000 tons, most of which has come from a single ore body.

**Geology:** The geologic setting of the Western mine (fig. 5) closely resembles that of all of the numerous talc deposits scattered throughout the 1000 square miles of the Death Valley talc province.

All of the commercial deposits thus far examined by the author involve similar rock types, and all have been formed at nearly the same stratigraphic position. The only noteworthy differences are in structural and mineralogic details. The talc bodies are confined to the lower portion of the lowermost carbonate member of a late pre-Cambrian sedimentary sequence. At each of the observed properties a diabase sill also is present, and is either in contact with or proximate to the talc bodies.

The carbonate member, characteristically a dolomite, is part of the Crystal Spring formation as defined by Hewett.<sup>2</sup> At the Western mine this member is underlain by about 1000 ft of vitreous quartzite and shale, which in turn overlies, with probable fault contact, pre-Cambrian schist and granitic gneiss. The lower members of the Crystal Spring formation at this locality (fig. 6) are representative of the sedimentary units exposed at all of the active talc mines in the southern Death Valley-Kingston Range area.

Current operations at the Western mine are confined to a north trending talc-tremolite zone that is continuously exposed for a length of more than 5000 ft and a width of 20 to 90 ft. The deposit lies almost entirely along the contact between the diabase sill and the overlying siliceous dolomite; masses of green altered rock, however, form local septa between the talc and the diabase near the south end of the deposit. All units dip moderately eastward. The Western deposit is the largest California talc body that has been extensively developed. Talcose zones of comparable or greater length are commonplace elsewhere in the Death Valley talc province, but none of these are known to have as continuous a body of commercial material.

Parts of the Western deposit consist chiefly of tremolite. Like the tremolite-rich material of the Silver Lake deposits, this rock became marketable only when its value as a ceramic material was

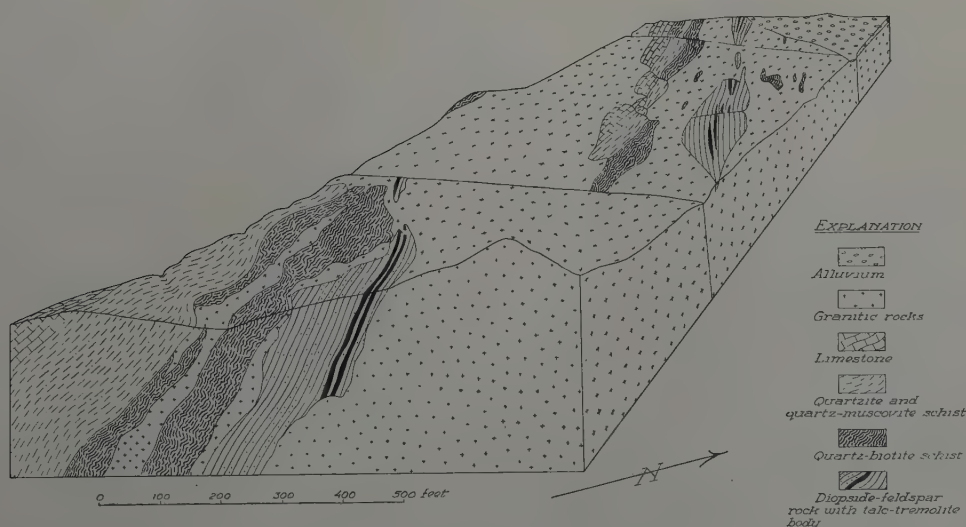


Fig. 3—Generalized block diagram of the western portion of the Silver Lake talc-bearing area.



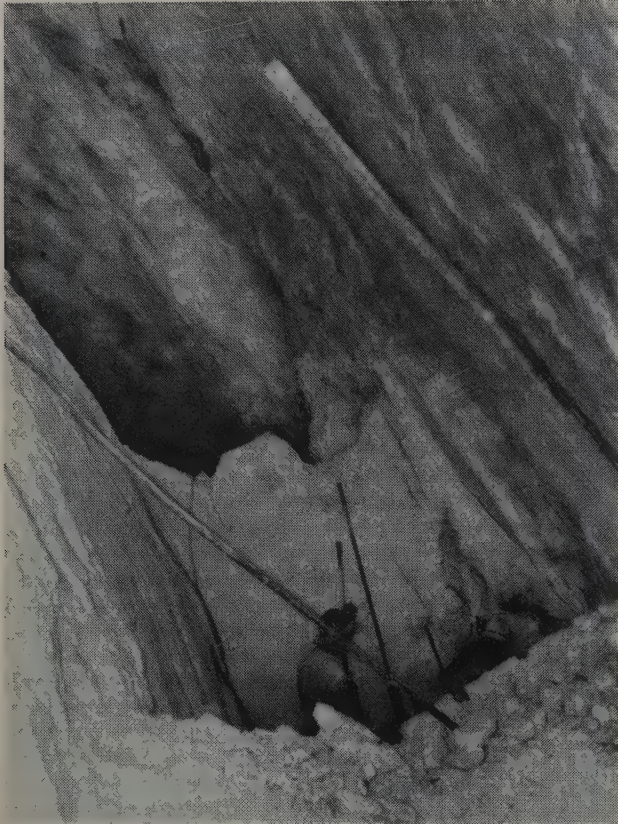


Fig. 4—Typical stope in the Silver Lake mine.

realized. At the Western mine there are three principal talc types which are mined separately, but are blended in the mine bins prior to shipment. The blended talc is used largely for ceramic purposes.

One talc is compact and evenly laminated, consisting of tiny unoriented tremolite needles; one is

nearly pure talc; the third is a mixture of talc, tremolite, and serpentine.

The compact talc occurs in a well defined zone that, where present, lies next to the diabase footwall with a knife-like contact. This talc is currently mined from a mass 1200 ft long and averaging 10 ft wide in surface exposure.

The other talc types are irregularly distributed, and commonly occupy the entire width of the ore body. Fingers of these types extend into partly altered hanging wall dolomite. Elongate masses of altered wall rock are common as inclusions in these talcs.

Though the structure of the Western deposit itself is rather simple, the area, as described by Noble<sup>3</sup>, has been subjected to intensive, late Tertiary faulting. In the Western mine area, fault movements have been partly along the relatively weak talc-rich layers. Consequently the less competent portions of the main ore body and the dolomite near the hanging wall are ordinarily sheared and brecciated. Mining, therefore, has been mainly in "heavy ground" that has required much timbering.

**Mining Methods:** The earliest work was largely a gophering operation, carried on at a time when uncertain markets necessitated sporadic mining. In later years production has been obtained through inclined shafts spaced at intervals of approximately 1200 ft along the ore body. The deepest of these has been extended 350 ft along a 52° slope. From these and appended crosscuts, drifts have been driven along the deposit in both directions. Most of the talc has been removed by overhand stoping. Talcs from the Western mine have been used for a variety of purposes in the past. Almost all of the current output is being absorbed by the ceramic industry, principally in the manufacture of wall tile.

**Other Active Properties in the Death Valley Area:** Other active properties in the Death Valley talc province include the Excelsior and Smith mines on the east and west flanks, respectively, of the Kingston Range; the closely grouped Monarch, Pleasan-

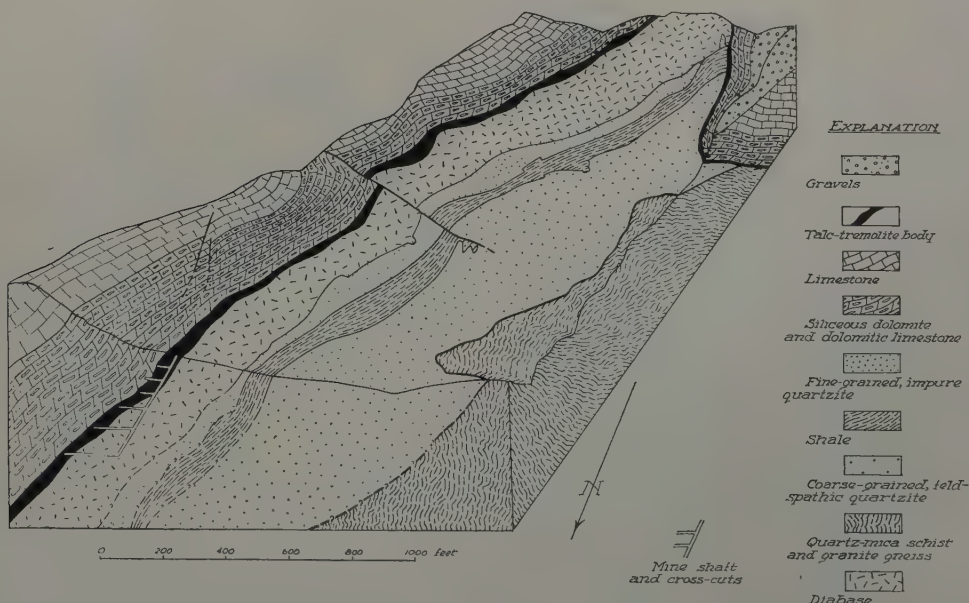


Fig. 5—Generalized block diagram of a portion of the Western mine area.



ton, and Ibex mines near Ibex Springs at the southern end of the Ibex Hills; the Superior and White Cap mines near Saratoga Springs at the southernmost tip of Death Valley; the Warm Springs mine in Warm Springs Canyon on the east flank of the southern Panamint Range; and the Western Atlas mine near the base of the north flank of the Avawatz Range. At each of these the general rock units exposed at the Western mine are duplicated. At the Smith mine area, however, a rhyolitic volcanic rock also is exposed. It forms extensive intrusive masses in the sediments, diabase, and talc bodies. The commercial talc bodies at the Superior and White Cap mines lie beneath, instead of above, the diabase sill.

The talc now being mined at Warm Springs (fig. 7) is in the lowermost of three parallel talcose zones that are separated by silica-rich calcareous layers. The deposits at this locality, though poorly exposed and but recently developed, may well be among the largest in the state. The talc bodies dip gently and are mined by closely-spaced parallel adits driven from a canyon wall. The adits are joined by low-angle raises that are progressively enlarged to overhand stopes. Talc is removed by tramming and practically no timbering is required. This mining procedure is the simplest of any talc mine currently operating in the eastern belt. Development thus far has been confined to a relatively small portion of the talcose zone, which is approximately two miles long.

Despite broad similarities, the deposits of the Death Valley province do differ markedly in mineralogic detail. The compact, tremolitic talc of the Western mine, for example, is not a widespread rock type. In many deposits even a talc-tremolite aggregate is uncommon or absent altogether, and fibrous or micaceous forms of the mineral talc predominate. The fibrous and micaceous talcs are used with tremolitic talcs in ceramic blends, or are marketed unblended for consumption in the paint, rubber, textile, and paper industries.

**Talc City Mine: Location and Production History:** The Talc City mine, near the southern end of the Inyo Range, is 13 miles southeast of Keeler, in Inyo County. This mine, which was first operated in 1915, has yielded more than 200,000 tons of talc, the largest total of any California property. The product, like that from several other mines in the Inyo province, is unusually pure and was considered a critical material during World War II. The area drew the attention of the U. S. Geological Survey in 1942, when a detailed study was made under the direction of Dr. B. M. Page. The following description is based in large part upon data kindly supplied by Dr. Page<sup>4</sup>.

**Geology:** The talc bodies of the Talc City mine area are contained in an irregular mass of dolomite, many hundreds of feet in maximum dimension. Paleozoic limestone, from which the dolomite appears to have been developed by hydrothermal alteration, both surrounds this mass and is included in it as residual "islands." Irregular bodies of a quartzite-like rock are also included in the dolomite, but their origin is obscure. The largest of these lie along the eastern borders of the limestone "islands," but some are themselves individual islands in the dolomite.

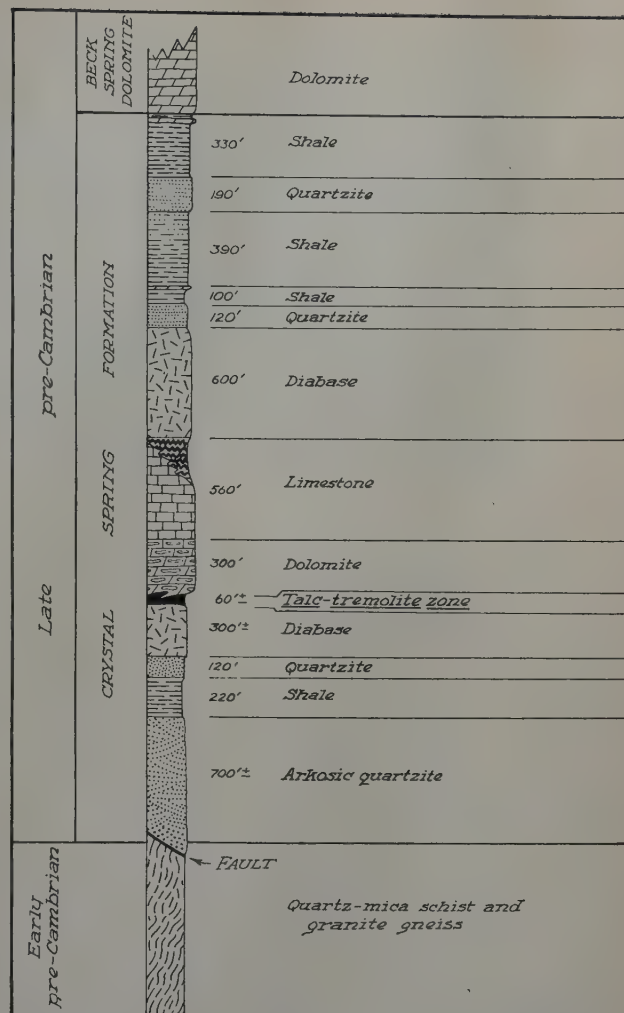


Fig. 6—Columnar section of the pre-Cambrian rocks exposed in the Western mine area.

The principal talc bodies characteristically occur along limestone-dolomite contacts that form the eastern borders of "islands." Others lie completely within dolomite. Much of the talc alteration appears to have been guided by shear zones. Nearly all of the talc at the Talc City mine was formed at the expense of dolomite, though at nearby mines and elsewhere in the Inyo province silica-rock has been extensively altered to talc. Four major ore bodies are exposed at Talc City; all are elongate and somewhat lenticular, but are irregular in detail. The two with the largest surface exposures are slightly more than 500 ft long and 50 ft in maximum width.

**Mining Methods:** Early operations were based on glory-hole mining, but an underground caving method was later employed. More recently the ore has been removed from overhand stopes that are supported by timbers where necessary. Systematic exploration since 1942 has outlined a downward and lateral extension of the largest talc body. Consequently the mine's proven reserves are now greater than at any time in its history.

**Uses:** Early production was used largely in the manufacture of paint, soap, paper, and cosmetics.



Fig. 7 — A portion of the Warm Springs Canyon talc zone. Talc bodies lie within siliceous limestone near the upper contact of a diabase sill.

The blocky nature of the talc which generally distinguishes the "steatite" variety made it suitable for the manufacture of carved electrical insulator bodies. As the demand for such bodies grew, particularly for use in high-frequency and ultra-high-frequency electrical equipment, manufacturing methods were developed which employed extrusion or molding. Thereafter "steatite" did not necessarily imply a carveable material, and the term, as used in the ceramic industry, has consequently come to mean any talc of sufficient purity ( $\text{CaO}$  and  $\text{Fe}_2\text{O}_3$  contents each less than 1.5 pct) that is adaptable to the currently-used techniques of insulator manufacture. It has been found, however, that talcs that grind to equant grains are more easily molded or extruded than talcs that grind to fibrous or micaceous grains. Moreover, the blocky "steatite," from which insulators previously had been carved, has been found to have equant grains in both natural and powdered form. The term "steatite," therefore, has not been applied generally to fibrous or micaceous talc even though they are sufficiently pure.

Early in World War II the national importance of the Inyo talc province as the principal domestic source of high-quality steatite became apparent. Heavy demands were placed on mines producing talcs of proven grade, and stringent government restrictions were placed on the use of such talc for any purpose other than the manufacture of electrical insulators. The Talc City mine proved to be the major source, but the White Mountain mine, on the southeast flank of the Inyo Range, shared a large portion of the wartime burden.

*Other Talc Deposits of the Inyo Range Area:* Talcs of steatite grade also occur at a number of smaller deposits in the Talc City area. These include the Alliance, Irish, Frisco, East End, Victory, Viking, Trinity, and White Swan mines, most of which were operated during World War II. Other deposits on the east and west flanks of the Inyo Range also contain steatite.

Noteworthy among the talc deposits of the Inyo province is the White Eagle<sup>5</sup>, on the east flank of the Inyo Range above northern Saline Valley. Here a large, irregular talc body has formed principally through alteration of quartz monzonite.

Quartzite and dolomitic limestone of Paleozoic

age border portions of the deposit and, near the contact, have themselves been altered to talc. The main talc body, however, is bordered by quartz monzonite and contains many quartz monzonitic residua. In many places the quartz monzonite-talc transition is complete within a few inches and is characterized by a decomposition of hornblende and biotite, and the replacement of feldspar and quartz by talc.

In recent years the Talc City and White Mountain mines have been the only ones continuously operated. The Lane and Nikolous mine in Eureka Valley, however, has been a substantial producer, and numerous other mines have been operated intermittently. With the end of World War II, restrictions on the use of high-quality steatite were lifted, and the material is now being employed in the manufacture of refractories, paint, paper and cosmetics, in addition to the manufacture of insulators.

#### Acknowledgments

The current studies of California's talc resources have proceeded at the instigation and under the helpful guidance of Dr. Olaf P. Jenkins, Chief of the California Division of Mines. Drs. Ian Campbell, R. H. Jahns and A. E. J. Engel of the California Institute of Technology also have been particularly helpful, both in offering suggestions in the field and in critical reviews of this manuscript. Valuable field assistance was rendered by Robert S. Orr, who voluntarily aided field studies at the Talc City mine, and by William E. Ver Planck and Mort D. Turner of the California Division of Mines.

#### References

- <sup>1</sup> W. J. Miller: Crystalline Rocks of Southern California. *Bull. Geol. Soc. America*. (May 1946) **57**, 499.
- <sup>2</sup> D. F. Hewett: New Formation Names to be used in the Kingston Range of the Ivanpah Quadrangle, California. *Jnl. Wash. Acad. Sci.* (1940) **30**, (6).
- <sup>3</sup> L. F. Noble: Structural Features of the Virgin Spring Area, Death Valley, California. *Bull. Geol. Soc. America* (1941) **52**, 941-1000.
- <sup>4</sup> B. M. Page: Some California Talcs of Steatite Grade. *Min. and Indus. News* (1948) **16** (1) 12.
- <sup>5</sup> L. A. Wright: The White Eagle Mine, an Example of the Steatitization of Granite (abst.). *Bull. Geol. Soc. America* (1948) **59**, (12) pt. 2.



## Mine Mechanization Program

by C. Kremer Bain

To keep shovels working to capacity in the smaller and more scattered ore bodies necessitated the development of a mobile jumbo drill rig, utilizing the V-cut round drilled to a template.

The paper shows the steps taken, and the difficulties encountered in accomplishing an increase of 38 pct in tons per drill shift over the method previously used, with probability of more efficient operation to come.

**U**NDER present economic conditions the necessity of mechanizing the mines of our country has become a very important problem. More and more mines are looking toward increased or complete mechanization of their operations, so in the writer's opinion it is quite pertinent that some of the associated problems that accompany such a program be pointed out.

The following generalization on attempting a comparatively slight change in mining methods at the Southeast Missouri properties of the St. Joseph Lead Co. is an attempt to point out to anyone faced with a mechanization program that much time and

C. KREMER BAIN, Member AIME, is Assistant to General Manager, St. Joseph Lead Co., Bonne Terre, Mo.

AIME San Francisco Meeting, February 1949.

TP 2743 A. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received Dec. 27, 1948; revision received Oct. 13, 1949.

effort can be saved, and many headaches avoided, if company policy can be established whereby everyone in the organization from the top down is desirous of improvement and is working towards the common objective. Any changes in mining methods, machinery, and so on, must entail co-ordination of all the other phases of mining as they all have to mesh as accurately as a smooth running chain of gears if satisfactory results are to be obtained.

The purpose of this paper is to point out some of the problems which have to be surmounted when any mechanization program is attempted, rather than to go into the mechanical details and actual operating data of the equipment employed. After all, machines have to be adapted to best suit the local conditions, so no general formula can be given that would apply to other properties.

Although St. Joseph Lead Co. has been mechanically-minded for a number of years, as shown by the fact that over 110 million tons have been loaded mechanically since 1922, any further mechanization

or changes always present a problem. When our ore bodies were of great height and area the breaking and loading were developed to a high degree of efficiency according to known standards. But in recent years, when our ore bodies became smaller and more scattered, the amount of tonnage we were able to break was not sufficient to supply our shovels with enough rock to keep them working to capacity. Our first step in attempting to rectify this situation was to develop a technique of moving the shovels to the broken ore in various stopes, which sometimes entailed a move each shift. A shovel is not producing when being moved, so within the last two years we have been endeavoring to devise some means of getting more rock broken in these smaller stopes and it was then that we became interested in jumbo drilling.

To illustrate the fact that mechanization is a process of a chain of developments rather than a stroke of genius can best be shown by a general summary of conditions and obstacles which were encountered in our present attempt to establish template drilling of a standardized round. Our first jumbo was more or less the conventional type with two machines mounted on a crossbar. This arrangement proved to be impractical to duplicate satisfactorily the regular routine of slabbing in a more or less radial pattern as carried out by column drilling. One machine was either too close to or too far from the face and there was another disadvantage in that one drillman was frequently delayed while waiting for his partner to bottom a hole. Our next step was to eliminate the crossbar so the machines could operate independently of each other, but we still encountered the same objections mentioned above except that one drillman was free to drill three holes instead of one in a vertical plane independently of his partner.

Time studies while trying to adapt the jumbo to the regular slabbing practice as accomplished by column drilling, revealed that twice as much time was spent in maneuvering the jumbo into position and aligning the holes as was consumed in drilling.

We tried side swinging to eliminate some of this moving and were then confronted with a blasting problem. In an ordinary size heading it would take at least seventeen 3-hole settings to advance the stope from pillar to pillar. Blasting with fuse was impractical because in order to get the proper sequence of blasting, starting with a 6 ft fuse would require a minimum of 19½ ft of fuse for the last hole. With the 10-delay electric caps available on the market, we tried series blasting of five 3-hole settings at a time but inevitably the lead wires were broken by the concussion of the previous blast, making it necessary to go back into the smoke to reconnect, which is impractical in every way and is against good safety practice.

During this period we decided to get an accurate record of our standard drilling as practiced from

Block models (scale 1 in. = 2 ft.) were made of each day's drilling and advance was colored for identification and the day-by-day progress studied closely. From these block models the present standard round was developed, applying the best results from column drilling as illustrated in fig. 1. We took sections marked 1 and 2 and placed them together, as shown in fig. 2, so we could incorporate more of the advantages of jumbo drilling than had heretofore been possible; namely,

1. Every man was free and independent of his partner for the entire shift's drilling.
2. Jumbo could be moved in on center line at the beginning of the shift and would not have to be moved after the original setting.
3. Permitted accurate spotting of holes as well as governing the depth of the holes to prevent over

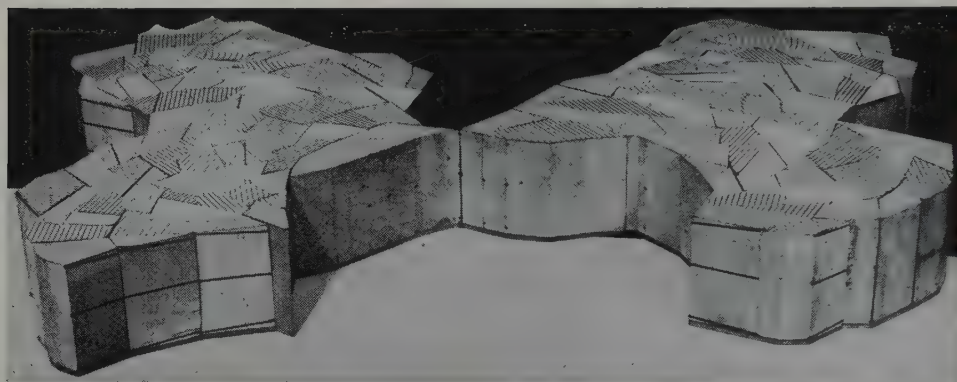


Fig. 1 — Block model showing conventional method of advancing a stope in Southeast Missouri room-and-pillar operations.



Fig. 2 — Same model as shown in Fig. 1 with sections 1 and 2 (which represent a part of four different shifts' drilling) removed and placed in foreground to compare with standard center-cut round.

columns. Each mine foreman selected a better than average driller and a better than average stope. These men were selected not only because of their ability as drillers but because they possessed a temperament that would not resent a close check upon their work. Each stope was surveyed daily and the advance plotted. The collars of each hole were located accurately along the hole alignment. This was checked the next shift by locating the points of the holes drilled the previous shift and measuring the bootlegs. From these data we gained some really valuable information as to the advance, burden, breaking angle, powder factor, percentage of holes broken and tons broken per foot of height.

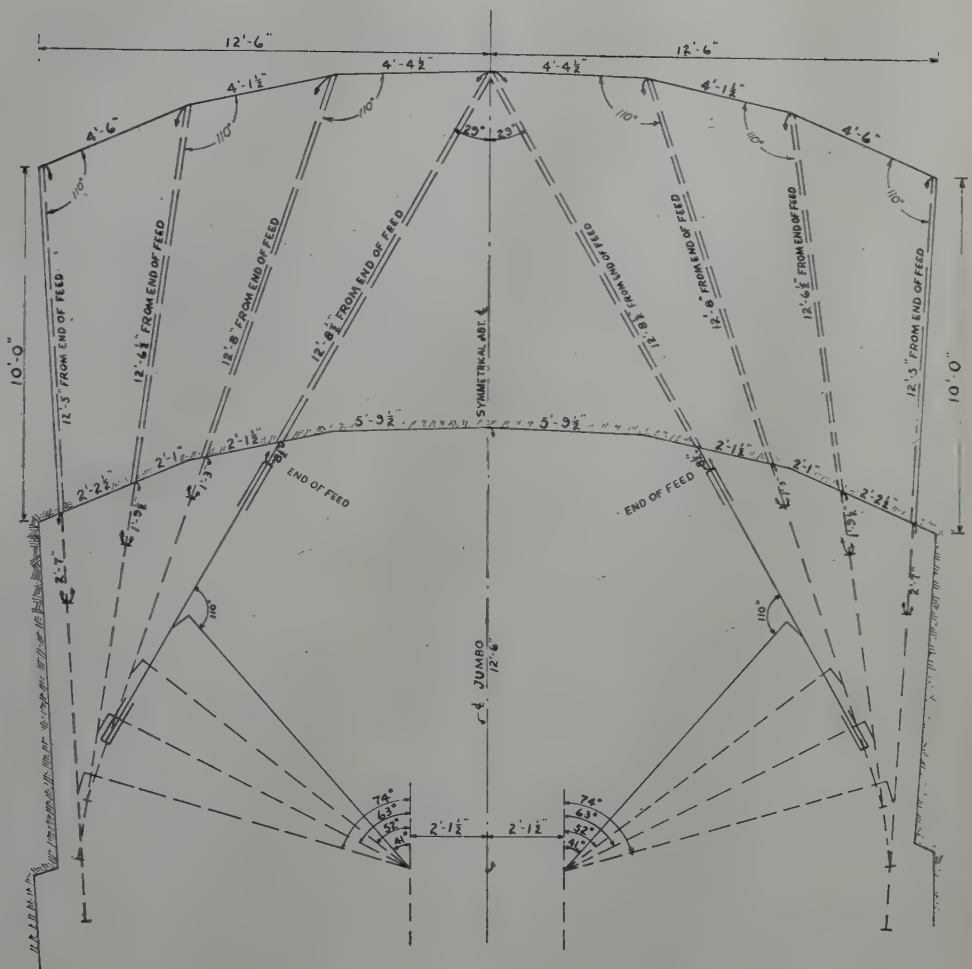
drilling, thereby minimizing the amount of bootleg.

4. Permitted use of the jumbo as a template to minimize lost motion and physical effort.

The solid block of ground remaining between the sections 1 and 2 left us with the problem of developing a dependable cut which would remove this "V" and still not deviate too far from our present drilling practice. The original efforts with this round met with only average results due to the inconsistency of pulling the cut. It took a lot of time and patience to develop the round to the present state, during which time we experimented with the size of the hole, kind and size of powder, and length of the powder column in relation to the



Fig. 3—Plan view of standard round showing location of the center line of rotation of the jumbo arms and how it determines the depth of hole drilled to conform with the breaking angle of the rock. Eight setting round. 21.9 tons per foot of height.



length of the hole. During this time we had the cooperation and help of the technical representatives of powder companies as well as the miners themselves.

The present V-cut round, as shown in fig. 3, is the result to date of this study and experimentation. We think it has the advantage of using standard length of steel to control the depth of drilling. It will be noted from the diagram that all holes are bottomed at approximately the same required depth when the machine is at the same place on the feed.

In breaking the first 187,000 tons by Jumbo drilling, including all mechanical changes and the development of the V-cut round, we averaged only 43 pct of the tonnage which reasonably should be expected from a shift's drilling, and because of mis-breaks, and so on, only 63 pct of the calculated tonnage was obtained from the round. This would have been most discouraging except for the fact that in spite of this poor showing our average tons per driller shift was 38 pct more than the average from column drilling during the period. At the present time our results are much more favorable and we have demonstrated that it is possible to get approximately 22 tons per foot of height in a face 25 ft wide, so we are quite confident that our efforts will confirm the feasibility of these methods.

During November 1948, some gratifying results were recorded. On one jumbo an average of 93 tons per driller shift was obtained for the 4-week period

ending November 29, using the V-cut round. Another jumbo in a different mine obtained an average of 112 tons per driller shift for the last week in November with the V-cut round. This is the highest average per jumbo-driller shift obtained to date. It is interesting to note that the same crew drilling with the same jumbo in the same height of ground, but advancing the stope in the conventional way as employed in column drilling, averaged only 54 tons per driller shift during the month of October 1948. The change in method to the standard V-cut round, using the jumbo as a template, with somewhat better car service, was the only difference in conditions affecting these two results.

The development of the round drilled to a template has thus accomplished the original objective of providing sufficient broken ore in the smaller stopes to equal a shovel average that was obtained in the stopes of greater height and area. It has eliminated the excessive movement of shovel equipment but has necessitated closer supervision, rearrangement of haulage to provide adequate car service for the concentrated loading operations, and long range planning to coordinate loading and drilling so that the broken ore available for loading would not interfere with the operation of the jumbo.

In conclusion, we wish to emphasize the statement made earlier that changes in mining methods, machinery, and so on, must entail changes in and coordination with the other phases of mining.

# Health and Safety Practices at Pioche

by S. S. Arentz

An organized safety program has reduced accidents at Pioche because effort is first devoted to arousing and maintaining interest in safety, followed by training in accident prevention, assigning responsibility for the detection and removal of safety hazards, and finally, by keeping score through an adequate system of records. A health program is integrated with the safety program.

**P**LANNED health and safety programs have become an essential part of American industry because such programs lead to increased operating efficiency, improved labor relations, better public relations, and to substantial savings in compensation insurance. Those of you who have had the unpleasant duty of informing the wife or widow of one of your men of his serious injury or death while on the job, know that all the benefits of a successful safety program do not show on the balance

---

*S. S. ARENTZ, Member AIME, is General Superintendent, Nevada Operations, Combined Metals Reduction Co., Pioche, Nevada.*

*AIME San Francisco Meeting, February 1949.*

*TP 2741 A. Discussion of this paper (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received Jan. 6, 1949.*

---

sheet. These programs are of particular importance to the mining industry because mining's reputation as an unusually hazardous industry and the commonly isolated location of mining operations tend to focus attention on these problems.

**Description of Operations:** Before proceeding with a discussion of our health and safety programs at Pioche, it may be proper to give a brief description of Pioche and of our operations there. Pioche is one of the early Nevada mining camps. It was founded shortly after the discovery of high grade silver ore in 1863 and mining has continued with more or less regularity to the present day. In an era of lawlessness, Pioche was notorious. The story persists that 75 men died with their boots on before one died a natural death, and old payroll records show that nearly as many gunmen were employed to stand off claim jumpers as there were miners working the mine. That was probably as close to a safety program as the times permitted. Pioche is situated in southeastern Nevada on the main highway between Ely and Las Vegas. The camp is on the flank of "Treasure Hill," near the original silver discovery, at an elevation of about 6000 ft. The present day population of about 2000 is primarily dependent upon the mines of the area, although Pioche also serves as the county seat of Lincoln County and as the center of the surrounding livestock industry. The camp is served by a branch of the Union Pacific Railroad and receives power from the generators at Hoover Dam.

The Pioche operations of the Combined Metals Reduction Co. were started in 1923 when the first complex lead-zinc ore was shipped to the company's mill at Bauer, Utah. The modern mill at Pioche was completed in 1941. The operations are medium sized in the nonferrous field, employing an average of 350 men in the mine, mill, and related works. The complex lead-zinc ore is mined from replacement deposits in a comparatively flat, extensively faulted, limestone horizon. Mining methods vary from stull-supported open stopes to filled square-set stopes. The thin bedded limestone and shale overlying the ore is allowed to cave as areas are mined out and caving frequently follows closely upon ore extraction. The relatively heavy ground and the numerous faults add to the problems of safe mining. The mine is well mechanized and the mill and surface plant are modern and well equipped. Labor is organized in a C.I.O. union and labor-management relations have been unusually harmonious.

During most of the period since 1923 a competent supervisory staff worked to reduce safety hazards but the primary responsibility for safety rested on the individual workman. Accidents happened and all too frequently they were regarded by all concerned as unavoidable.

In October 1939, the late Robert L. Dean became superintendent at Pioche. Most of his previous experience had been in the fields of iron and coal mining and from that experience he brought the concept that no accident is unavoidable. Many of the features of our present health and safety programs were initiated by Mr. Dean during his term as superintendent.

**Health Program:** Our health program centers in Dr. Q. E. Fortier and his new, well-equipped, and well-staffed, modern hospital in Pioche. The program starts with a thorough pre-employment physical examination and is followed by yearly re-examinations at the expense of the company. The Pioche Mutual Benefit Association, to which all Pioche mine operators and employees belong, pays benefits covering hospitalization and surgery expense incurred by employee members and their families. The Association is governed by a board of directors elected by its members. The mine operators of the district donated the original capital and pay the monthly dues of the employee members. The employees pay the dues covering members of their families. Though not strictly a part of the



health program, all employees are covered by group life insurance with the premium paid by the company. The union supplements the program for its members by the payment of compensation benefits to men losing time due to illness or nonindustrial accident.

The Pioche operations offer no unusual health hazard. Both the mine and mill are well ventilated with suitable dust-control equipment and practices. The underground temperature is roughly equivalent to the mean annual surface temperature and most workings are dry. Well-equipped change rooms, a clean boarding house, and above average bunkhouse facilities maintained by the company contribute to the health program.

**Safety Program:** Our safety program is the direct responsibility of our safety inspector, D. T. Sullivan, but all supervisors are responsible for the safety of the men under their supervision, and the union safety committee works closely with the management on matters relating to safety.

**Inspection:** Accident prevention starts with the detection and removal of safety hazards. The workmen, the supervisors, and the engineers are continually making informal safety inspections in the course of their regular duties and they automatically correct many hazards as they are noted. But some hazards are overlooked and within a short time become so familiar a part of the daily scene that they are completely disregarded. Other hazards are assumed to be the responsibility of someone else and are allowed to persist until they result in a serious accident. It is therefore essential that regular safety inspections be made and that responsibility for certain types of inspection be definitely assigned to individual members of the crew.

At Pioche we maintain the following safety inspections and the results of each inspection are entered on signed reports:

1. Daily inspection of all hoisting equipment including hoist ropes, safety and control devices, cages, skips, and hoisting compartments.
2. Semiweekly inspections of all active mine workings by safety inspector.
3. Semimonthly inspections of main shafts by maintenance crew.
4. Regular inspections by state mine inspector or his deputy.
5. Monthly inspection of all mine workings by employees safety group headed by the chairman of the union safety committee assisted by two experienced miners appointed by the management. The group is accompanied by the safety inspector, the mine foreman, or a shift boss.

6. Regular mechanical inspection covering condition of equipment, safety guards, and so on.

In addition to the above-listed, regular safety inspections there are numerous special inspections including the investigation of all lost-time accidents.

Safety inspections are of value only to the extent that they are followed by action designed to remove the hazards detected. At Pioche the reports on the various safety inspections are read by the department head concerned and by the plant superintendent and are then posted on the plant bulletin

boards. In most cases the hazards noted are corrected immediately and a note is made on the report showing the action taken. Occasionally a recommendation in a report involves a material change in method, equipment, or policy, and such cases are considered by the plant and company management before action is taken. But in all cases some definite action is taken and noted on the posted reports before the reports are relegated to the files.

All lost-time accidents are investigated by an employee's committee appointed from the men working in the department in which the accident occurred, and by the safety inspector and the department head. The reports on the investigations show cause of accident, fix responsibility for the accident, and make recommendations for avoiding similar accidents. The reports are posted on the bulletin board and are distributed to the general manager, the department heads, and the local union. Our experience indicates that these reports have done much to reduce accidents and that the employees' committees take the job seriously and are fair in fixing responsibility.

**Accident Prevention:** Attention is given to the prevention of accidents through good housekeeping, the proper installation and use of equipment, and the use of protective equipment, warning signs, and other safety devices. The following details may be noted:

1. Machinery is equipped with safety guards and electrical installations are grounded.
2. Warning signs and signals are maintained. Scotchlite reflecting tape is widely used for signs underground and for indicating obstructions such as projecting chute lips. Patches of the tape on men's hats have prevented several possible haulage accidents.
3. All underground men wear hard-boiled hats, steel-toed boots, and battery lamps. Candles and matches are provided at convenient locations for use in testing air in workings off ventilation courses.
4. Entrances to abandoned workings are barricaded. Signs indicating nearest exit are posted at raise stations and drift junctions.
5. Well-equipped, well-maintained first aid stations are located on all levels underground and at central points on the surface.
6. Scaling bars and similar equipment are painted a bright yellow to aid in identification and as a reminder of their regular use.

**Training:** Training is an essential part of a safety program. We have made good use of the excellent training courses offered by the U.S. Bureau of Mines and have supplemented these courses with a training program conducted by our plant safety inspector. All Pioche supervisors and many key employees have completed the Bureau of Mines course in accident prevention and about 75 pct of our crew have completed the course in first aid to the injured. Many of our supervisors have instructor's certificates in first aid training. Attendance at accident prevention and first aid classes is voluntary and on the employee's own time.

We give regular training in mine rescue work to a picked crew, and annual classes in this subject are



conducted by a Bureau of Mines instructor. This training is given on company time.

The course conducted by the plant safety inspector covers accident prevention, plant safety rules, discussion of accident investigation and mine safety inspection reports, and demonstrations of safe methods of doing typical underground jobs. The classes are held on the main shaft stations underground just before a shift is hoisted to the surface after work. Classes are held biweekly for each crew and men attending class are delivered to the shaft station about 20 min before the end of the regular shift.

All employees receive a printed handbook containing the plant safety rules before reporting for work. In the past it was customary to penalize men for safety infractions with from one to ten days off without pay but under present conditions we find it better to caution men for minor infractions and to discharge them for repeated or flagrant violations. The rules were compiled and are occasionally revised by a committee representing both labor and management.

**Accident Reports:** Detailed records are necessary for a well planned safety program. Such records show progress made in safety, give a comparison of results achieved with those obtained at other mines or with past performance at the plant, and frequently reveal unsuspected hazards or dangerously careless employees.

At Pioche our records start with a foreman's report of accident on all injuries, even minor cuts and bruises. The reports are made in duplicate with one copy for the injured employee to give the doctor treating the injury, and the other for the department head and plant superintendent for their information before it is turned over to the safety inspector for his permanent files. A stub attached to the doctor's copy must be filled in showing the employee's release by the doctor before the injured employee may return to work. Failure to report injuries by the end of the shift in which they occur subjects an employee to immediate discharge. Prompt treatment of even minor cuts and scratches has eliminated infection following injury.

Other records include the various accident reports, safety inspection reports, reports on man-shifts worked in the various departments and reports on days lost due to accident. From this information the safety inspector compiles records showing the location, cause, and extent of injury of each accident, the severity and frequency rates of lost-time accidents, and individual records covering the accident history of each employee. Monthly and yearly reports are prepared giving detailed information on the progress of our safety program with graphs showing comparisons with past periods at the plant and with national averages. The data are prepared to conform with the similar figures released by the U.S. Bureau of Mines.

**Maintaining Interest:** The problem of arousing and maintaining interest in safety is a most difficult one and its solution can assure a successful program. The training programs, the safety inspections, and accurate records showing progress made, all contribute to arousing and maintaining interest in

safety, but in our experience it takes some additional incentive to keep interest alive over any extended period of time. We have found that if the interest of the supervisors can be aroused and maintained, the problem is more than half solved. One method we employ with success is a bonus system under which all supervisors completing a month without a lost-time accident among the men in their crew receive a cash bonus. The amount of the bonus varies depending upon the number of men supervised and the hazard rating of the department in which they are employed. A variation of this system has been a contest between the three mining operations and between the two mill and surface operations of the company. The contests run for periods of four months with a substantial cash bonus for the foreman and shift bosses at the mine and mill having the lowest lost-time frequency record. As the relative standings of the several operations are posted monthly, the contest serves to maintain a lively interest.

It is equally important and even more difficult to arouse and maintain the crew's interest in safety. Our methods have ranged from essay contests in the local schools to safety parties and dances. For the past few years we have had success with the following method:

All plant employees are divided into four groups with the men from the several departments distributed equally through each group. The men in any group which does not have a lost-time accident during the month receive free show tickets for themselves and their families. Needless to say, the man that knocks his group out of show tickets is unpopular with the group.

A second method regularly employed at Pioche is a bonus system in which the company contributes a substantial sum to the employees entertainment committee whenever the plant completes sixty days without a lost-time accident and a similar amount for each thirty days the accident-free period is extended.

A primary requisite to any of these methods is that there be no misunderstanding as to what constitutes a lost-time accident. We leave the decision up to the doctor on the understanding that the man does not return to work unless he can perform his regular duties without serious discomfort or danger to himself or others. We sometimes change the duties of injured men to allow them to draw wages instead of compensation, but all such cases are carried as lost-time accidents.

Attractive bulletin boards, posters, and signs all contribute to improved plant safety.

**Achievements:** Even though we have given a great deal of attention to the problem of preventing accidents, we are still far from our goal of an accident free year or years. We cannot fail to be impressed by the outstanding records achieved by mining operations placing high in the national safety competition. The methods outlined above have substantially improved our safety record. Experience has shown that there is no easy way to achieve a safe operation. Safety, like all other good things in life, is had only in proportion to the brains and effort devoted to it.



# Faster Calculation of Plane Triangulation Systems

## by Calculating Machine and Semigraphical Methods

by Richard Hamburger

Calculating machines permit the use of the more rapid cotangent and semigraphic solutions of plane triangulation. The results of these methods are as accurate as those of other methods. Simple adjustment of the net, and ease of calculation encourage more complete field work. Errors in field work or in computation are easily localized.

**(1) Introduction:** The purpose of this paper is to call attention to certain methods which will shorten and simplify the calculation of plane triangulation. These methods, though not new,<sup>1,2</sup> do not appear to have been properly publicized in the United States.

The cotangent (or tangent) solution of a triangle permits calculation of coordinates in a smooth series of operations of the calculating machine with a minimum of notation. The semigraphical solution permits the simultaneous solution of a number of intersections, at one station, without the added burden of solving additional triangles. This in turn encourages more numerous field checks and more complete office computation than would normally be undertaken.

RICHARD HAMBURGER, *Junior Member AIME, is Junior Engineer, Inspiration Consolidated Copper Co., Inspiration, Arizona.*

TP 2798 A. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950.

Tables I and II show the distinct advantage which the combined use of the cotangent and the semigraphical solutions hold over, what might well be called, the standard solution.

It will be shown that the positions for the entire net in fig. 1 may be calculated without computing any distances, thus saving much computation.

**(2) Comparison of Cotangent and Standard Solutions:** To compare the cotangent method with the standard method, the notation involved in the solution of triangle I-X-4, fig. 1, by both methods is shown. Fig. 2 shows the given information, that is, the coordinates and azimuths of Stations 1 and 4, and angles of the triangle or the azimuths from Stations 1 and 4 to Station X. (Azimuths are used here because the author feels that they are easier to handle, but bearings may be used by those who prefer them).

Cotangent Method			
	Eastings	Northings	
Sta. 1	2,151.62	3,400.52	Cot 192° 33' 30" +4.489074
Sta. 4	6,181.59	1,125.70	Cot 142° 00' 25" —1.280262
			(See Sec 3) +5.769336
Sta. X	2,651.608	5,645.00	
Standard Method			
Sta. 1	2,151.62	3,400.52	Distance Sta. 1 to Sta. 4 by
Sta. 4	6,181.59	1,125.70	Pythagorean theorem on
Diff.	4,029.97	2,274.82	machine:
			4,627.68
	(sin 22° 33' 48")		
4,627.68	×	0.383704	=
		0.772195	2,299.50
	(sin 50° 33' 05")		
	(sin 106° 53' 07")		
4,627.68	×	0.956888	=
		0.772195	5,734.52
	(sin 50° 33' 05")		
	(sin 192° 33' 30")		
	0.217433	×	2,299.50 = +499.99
	(cos 192° 33' 30")		
	0.976075	×	2,299.50 = +2,244.48
Sta. 1	2,151.62	3,400.52	
	+499.99	+2,244.48	
Sta. X	2,651.61	5,645.00	
	(sin 142° 00' 25")		
	—0.615566	×	5,734.52 = —3,529.98
	(cos 142° 00' 25")		
	0.788085	×	5,734.52 = +4,519.29
Sta. 4	6,181.59	1,125.70	
	—3,529.98	+4,519.29	
Sta. X	2,651.61	5,644.99	

It will be seen from the above comparison that the advantages of the cotangent solution of a triangle are: (a) rapidity, (b) full use of the calculating machine, (c) the small number of

functions to look up, and (d) the small amount of notation involved. There is, however, one great disadvantage also apparent above. The cotangent solution, within itself, contains no checks. This disadvantage must be overcome by careful notation, and accurate extraction of functions.

(3) **Solution of the Triangle by the Cotangent Method:** The machine calculation of a triangle by

once selected, are maintained throughout the calculation. It is advisable to set  $E_A$  as far to the left, and  $\cot \alpha$  as far to the right as is possible on the calculator. The exact position will vary with the capacity of the machine at hand. The selection of the decimal point positions should permit the value of the last significant number of interest to be no longer affected by further operations of the calculating machine. By operating the calculator is meant the turning of the crank or other motivating device in either a positive or a negative direction.

2) Operate the calculator so as to change  $E_A$  on the revolution dials to read  $E_B$ .

3) Clear the keyboard only and set ( $\cot \alpha - \cot \beta$ ) on the keyboard.

4) Operate the calculator so that the product dials read  $N_B$ . Read  $E_C$  on the revolution dials and write it down. During this step carry  $E_C$  one or more places beyond the last significant number desired, and use this value in calculating  $N_C$  in step 7. In performing steps 4, 5, and 7 on calculating machines having no carryover mechanism on the revolution dials, care must be exercised so as not to allow numerals of opposite sign to appear on the revolution dials. Thus if black (positive) numerals are being used, the machine must be operated so as to maintain these numerals black (positive) throughout the calculation. Since this step is a division it should start at the left hand part of the revolution dials and progress to the right (that is, the carriage moves to the left). To maintain numerals of the same sign, on calculating machines with no carryover mechanism on the revolution dials, may require some juggling of the carriage between

two adjacent columns. Thus if the machine is being operated in a subtractive manner and it is found that, say, 30 (black) is too large a number on the revolution dials and that 20 (black) is too small a number on the revolution dials to allow the numerals on the product dials to approach the value of  $N_B$ , make the numerals on the revolution dials read 29 (black) and proceed from there to find that number, between 30 (black) and 20 (black), which best satisfies the condition sought. A similar operation may be performed if the machine is being operated in an additive manner and

Table I. Comparison of Methods in Single Solution

To Solve Triangle and Check	Standard Solution	Cotangent and Semigraphical Solution
Number of functions looked up .....	7	4
Operations of machine .....	11	7
Number of series of operations .....	5	3
Sum = Chance of error and speed of operation ..	23	14

The addition of one extra observation brings the possible answers to three instead of one, and gives table II.

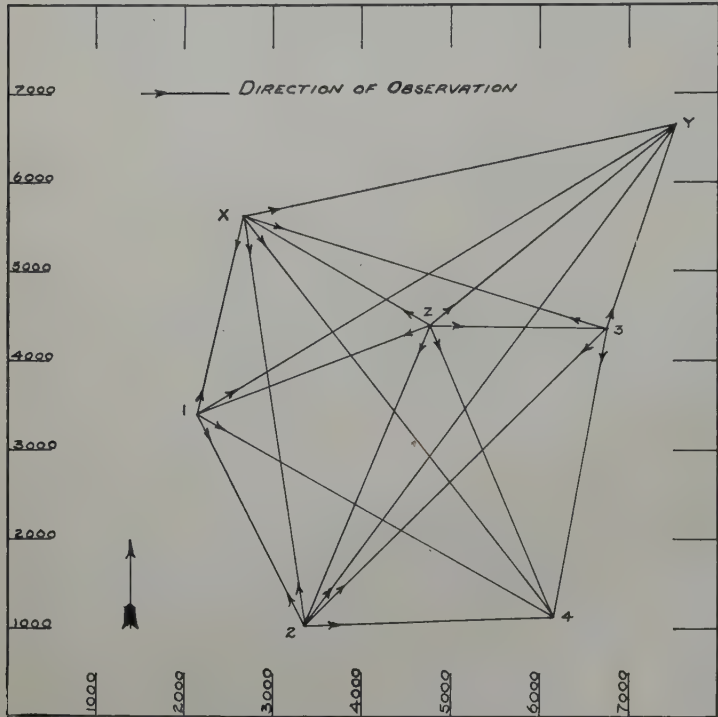


Fig. 1—Triangulation net.

the cotangent method is based on the following formulas whose derivation will be given in Sec. 8. The notation is that of fig. 3.

Given  $E_A$ ,  $N_A$ ,  $E_B$ ,  $N_B$ , and azimuths  $\alpha$ , and  $\beta$ , find  $E_C$ ,  $N_C$ .

$$N_A + (E_B - E_A) \cot \alpha = N_D; E_B = E_D \tag{1}$$

$$E_D + \left[ \frac{(N_B - N_D)}{(\cot \alpha - \cot \beta)} \right] = E_B = E_C \tag{2}$$

$$N_D + (E_C - E_D) \cot \alpha = N_C \tag{3}$$

The following notation<sup>8</sup> is complete and convenient, and embodies all of the writing involved in the solution of the equations.

Sta. A	$E_A$	$N_A$	$\cot \alpha$
Sta. B	$E_B$	$N_B$	$\cot \beta$
			$\cot \alpha - \cot \beta$

Sta. C     $E_C$      $N_C$

The procedure of the calculation is as follows.<sup>4</sup>

1) Set  $N_A$  on the product dials,  $E_A$  on the revolution dials, and  $\cot \alpha$  on the keyboard: The decimal points must be so placed that they are consistent. The rule is that the sum of the decimal places on the revolution dials plus the sum of the decimal places on the keyboard equals the sum of the decimal places on the product dials. For example, if  $E_A$  is set on the revolution dials with the decimal point at three places, and  $\cot \alpha$  is set on the keyboard with the decimal point at six places, then  $N_A$  must be set on the product dials with the decimal point at nine places. These decimal point positions,



it is found, for instance, that 59 (black) is too small and that 69 (black) is too large. Make the numerals on the revolution dials read 60 (black) and proceed with the step. At all times the black (positive) numerals must be used. The computer should guard against going through the red (negative) numerals and ending again with black (positive) numerals. If the calculation is started with the red (negative) numerals, then these must be maintained in a similar manner, remembering that the zero is usually common to both series.

5) Without changing any setting, operate the calculator so that the revolution dials read  $E_B$ .

6) Clear the keyboard only, and set cot  $\alpha$  on the keyboard.

7) Operate the calculator so that the revolution dials read  $E_C$ . Read  $N_C$  on the product dials.

The following rules must be adhered to.

1) The signs of cot  $\alpha$  and of (cot  $\alpha$  — cot  $\beta$ ) must be the same.

2) If the sign of cot  $\alpha$  is positive the original setting of  $E_A$  on the revolution dials uses the positive numerals. If the sign of cot  $\alpha$  is negative the original setting of  $E_A$  on the revolution dials uses the negative numerals. On calculating machines having carryover mechanism on the revolution dials the sign of the numerals on this set of dials is controlled by a key. On calculating machines which do not have the carryover mechanism on the revolution dials there are two series of numbers, one positive and one negative, on the revolution dials. These two series are usually of different colors, such as black for the positive series and red for the negative series. The positive series is the one normally used in multiplication and the negative series the one normally used in division. As noted in step 4 of the procedure the use of either the positive or negative setting of the revolution dials must be maintained throughout the calculation.

3) The signs of  $E_A$  and  $E_B$ , and the signs of  $N_A$  and  $N_B$  must be the same. As it is convenient to have all of the calculations in the northeast quad-

Table II. Comparison of Methods in Multiple Solution

To Solve Triangles and Check	Standard Solution	Cotangent and Semigraphical Solution
Number of functions looked up .....	21	3
Operations of machine .....	33	9
Number of series of operations .....	15	3
Sum = Chance of error and speed of operation ..	69	15

rant, a suitable constant may be added to the coordinates when convenient.

Cotangents near  $0^\circ$ , and those near  $180^\circ$  are such large numbers that their use makes the results unreliable. In such cases the tangents of the azimuths are used. Use the same notation and the same steps, but read tangent for cotangent, easting for northing, and northing for easting.

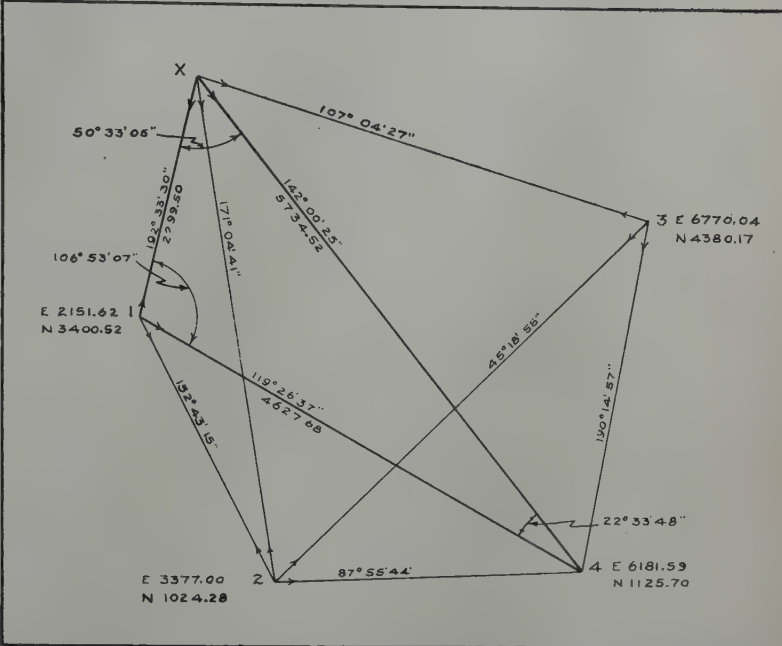


Fig. 2—Information used in solving for Station X by the standard method, and by the cotangent method.

(4) **Semigraphic Computation:** Semigraphic computations,<sup>5, 6, 8</sup> because of their simplicity, are one of the greatest time savers in the computation

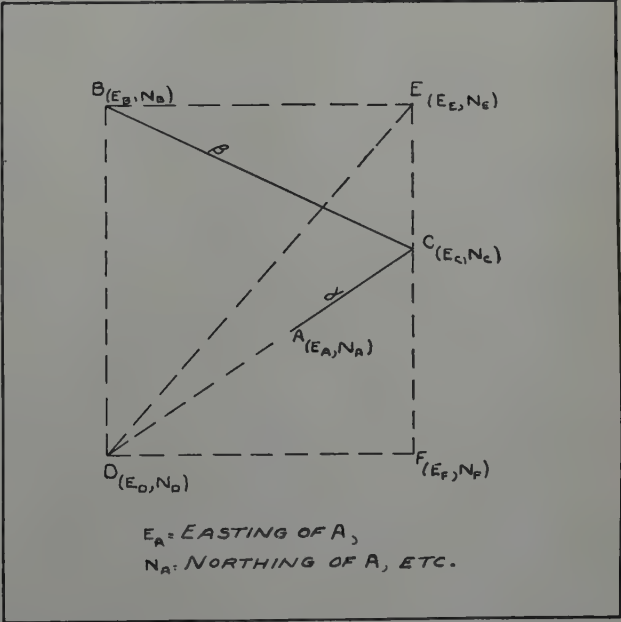


Fig. 3—General diagram for the cotangent solution.

Given the positions of A, B, and C, and the azimuths  $\alpha$  and  $\beta$  from A, and B to C. Construction: (1) Extend  $\alpha$  to intersect  $E_B$ . Call this position D. (2) Call the intersection of  $E_C$  and  $N_B$ , E. (3) Complete the rectangle. (4) Connect D to E.

of plane triangulation. The method consists in calculating the intersection of azimuths, from known stations, with coordinate axes of a position which is approximately correct for the unknown point. Having computed these intersections the information is plotted on graph paper and the intersection of the azimuths (if only two) is scaled. If there are more than two azimuths plotted, an error figure will result and a point must then be selected, which will best fit the given information.

The intersection of the azimuths at Station X from Stations 1 and 4 has been computed in the example showing the computation by the cotangent method. Referring to fig. 1 it will be seen that

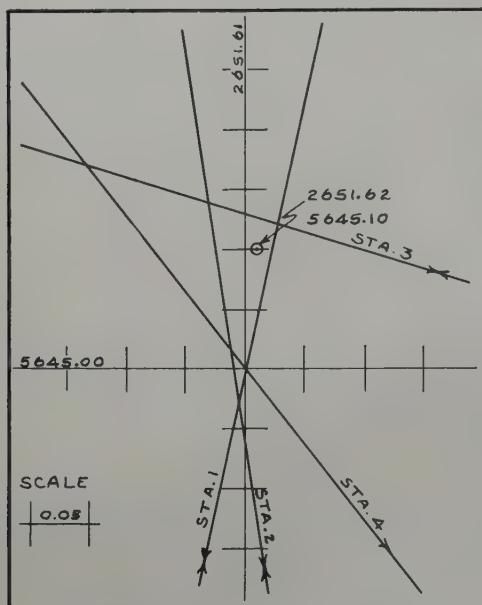


Fig. 4—Error figure for Station X.

there are also observations including Stations 2 and 3. Using the position previously obtained as an approximate position for Station X the following notation illustrates the remaining calculation; the azimuths from Stations 2 and 3 are given.

Sta. X	2,651.61	5,645.00	
Sta. 2	3,377.00	1,024.28	171° 04' 41" —0.156988
Diff. =	—0.01	2,651.60	4,620.72
Sta. 3	6,770.04	4,380.17	107° 04' 27" +0.307147
	4,118.43	5,645.13	+0.13 = Diff.

The results are then plotted on graph paper to a suitable scale, fig. 4. It is evident from fig. 4 that the above computation and plotting is equivalent to the solving of five additional triangles. The results are as accurate as any numerical solution, and require far less labor.

The advantages of the semigraphical solution are: (a) rapidity of computation without loss of accuracy, (b) the ease with which errors may be located (either errors in computation, or errors in field work). To illustrate point (b) an error will be purposely introduced into two of the illustrative problems.

The semigraphic solution is based on the following formulas<sup>7, 8</sup> whose derivation will be given in Sec. 8. The notation is that of fig. 5. Given  $E_A$ ,  $N_A$  and azimuth  $\alpha$  find the intersection of  $\alpha$  with coordinates  $E_B$ ,  $N_B$ .

$$E_B = E_A + (N_B - N_A) \tan \alpha \quad [4]$$

$$N_B = N_A + (E_B - E_A) \cot \alpha \quad [5]$$

Only one of these need be calculated since the line may then be plotted with a protractor. When  $\alpha$  is close to  $0^\circ$ ,  $90^\circ$ ,  $180^\circ$ ,  $270^\circ$ , and  $360^\circ$ , one of the axis intersections may lie quite far from the approximate position chosen. It is therefore convenient to work with the equation which will give the closest axis intersection.

The following<sup>9</sup> rule will produce this result.\*

\* This rule can be used only with natural function tables arranged so that reading from the top forwards are values from  $0^\circ$  to  $44^\circ$ , and from the bottom backwards values from  $45^\circ$  to  $89^\circ$ . Most function tables are arranged in this manner.

1) For the function of  $\alpha$  always use the value of the column headed tangent at the top of the page. This value will always be 1.000000 or less.

2) Find the greatest difference in coordinates between A and B; either  $(E_B - E_A)$  or  $(N_B - N_A)$ .

3) Multiply this difference by the function of  $\alpha$  and add algebraically to that part of the coordinate of A which was not used in step 2. The result will be the closest intersection of  $\alpha$  with one of the coordinate axes of B. The multiplication and addition should be combined on the calculator so that only the final answer need be recorded.

The above rules, like all rules, should at times be violated. Thus this calculation is a convenient check for errors introduced into the computation by the cotangent method, but it would be dangerous to re-use a function.

(5) **The Error Figure:** When more than two azimuths are observed to a point there will in all likelihood be an error figure (see fig. 4). This error figure consists of the area inclosed by the azimuths, their points of intersection, and the area immedi-

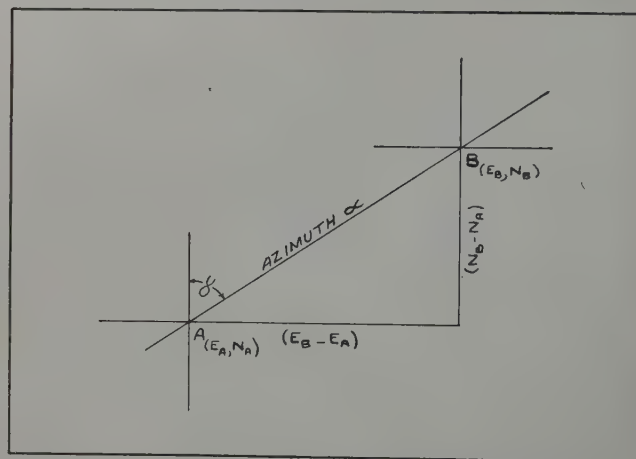


Fig. 5—General diagram of semigraphical solution.

ately adjacent thereto. The error figure will be a triangle in the case of three observed azimuths, and some more complicated figure when there are more than three observations.

The problem of the error figure consists in finding that point which most nearly satisfies all of the observations. This point, when chosen, will be the final coordinates of the station. The following rules are used in selecting the point:<sup>10, 11</sup>

1) The point lies closest (perpendicularly) to the azimuth from the nearest station, and furthest from the azimuth from the furthest station.

2) The point lies in such a position that, re-



gardless of sign, the change in azimuth caused by this selection is roughly the same for all observations.

3) Azimuths observed in both directions carry more weight than those observed in only one direction. This is obviously not true if only one azimuth has been observed in both directions.

Graphs and tables<sup>12, 13</sup> may be constructed showing the relationships in rule 2, and judgment must be used in applying rule 3. It will be found that, after slight practice, the position may be selected by eye with nearly the same accuracy.

The error figure permits the detection of mistakes. An azimuth which plots out of sympathy should be checked for mistakes in computation and field work. If no mistakes are found in either of these, and the position of the fixed station is not in doubt, then this azimuth should exert its proportional strength in the error figure.

**(6) Direction and Azimuth Adjustments:** In a triangulation net which consists of figures more complicated than chains of simple triangles, some form of adjustment must be made so that the results will be the most probable positions for the observations made. For precise work an adjustment by the method of least squares<sup>14</sup> would be justified. For most work a system which approximates this, and which is easy to handle, will suffice. Such a system is found in the direction and azimuth adjustment,<sup>15, 16</sup> (see table III), used in connection with the error figure. This notation consolidates all of the angular information from the beginning of the computation to the final (adjusted) azimuths. This notation permits the rapid extraction of data from separate observation stations pertinent to one problem. The great advantage of this method of handling the observations in conjunction with the cotangent solution is that it permits easy adjustment of the azimuth while eliminating the necessity of the intervisible base line. Thus the observation stations may be chosen for their convenience. As long as all stations used in determining azimuths are in sympathy this introduces no additional error, and is, in many instances, a great help to the observer to whom it allows a greater flexibility.

The form consists of eight vertical columns, and a separate block for each station occupied. Column 1 shows the Station Occupied, followed by the Stations Observed from it. Column 2 shows the Observed Directions. Column 3 gives the Approximate Azimuths from the station occupied to the stations observed. It is convenient to make these values less than the correct values so that the signs in columns 5 and 7 may be positive. Column 4 shows the "Forward Azimuths." These are the "Back Azimuths" values transferred where pertinent. They only appear when the station occupied is unknown, and one

Table III. Directions and Azimuths

(1)	(2) Observed Direction			(3) Approximate Azimuth			(4) Forward Azimuth	(5) Prov. Corr.	(6) Back Azimuth	(7) Final Corr.	(8) Final Azimuth			
	°	'	"	°	'	"	°	"	°	"	"	°	'	"
Sta. 1														
X	00	00	00	12	33	17			33	29	+12	12	33	29
Y	46	17	11	58	50	28			50	40	+12	58	50	40
4	106	53	10	119	26	27					+10	119	26	37
2	140	09	44	152	43	01					+14	152	43	15
											+12			
Sta 2														
1	00	00	00	332	42	50					+25	332	43	15
X	18	21	23	351	04	13			04	40	+29	351	04	42
Y	63	50	06	36	32	56			33	24	-9'38"	36	23	18
3	72	35	39	45	18	29					+26	45	18	55
4	115	12	24	87	55	14					+30	87	55	44
											-27*			
											+28			
Sta. 3														
4	00	00	00	190	14	17					+40	190	14	57
2	35	04	18	225	18	35					+20	225	18	55
X	96	49	41	287	03	58			04	28	+28	287	04	26
Y	188	03	56	18	18	13			18	42	+28	18	18	41
											-30*			
											+29			
Sta. X														
Y	00	00	00	78	22	46			22	58	+12	78	22	57
3	28	41	28	107	04	14	04 28	+14	04	26	+12	107	04	26
4	63	37	27	142	00	13			00	25	+15	142	00	28
2	92	41	44	171	04	30	04 40	+10	04	42	+13	171	04	42
1	114	10	32	192	33	18	33 29	+11	33	30	+11	192	33	29
								+12			+12			
Sta. Z														
1	00	00	00	249	05	15					+03	249	05	18
X	51	16	26	300	21	41					-04	300	21	37
Y	161	40	01	50	45	16					+00	50	45	16
3	201	43	00	90	48	15					-10'12"	90	38	03
4	267	39	21	156	44	36					-04	156	44	32
2	313	22	22	202	27	37					-07	202	27	30

\* Crossed out.

or more directions have been observed in both directions. Column 5 contains the Provisional Corrections. This is the difference between column 4 and column 3. Column 6 gives the "Back Azimuths." This value is the sum of the value in column 3 and the mean of the Final Corrections in column 7, or the sum of the value in column 3 and the mean of the Provisional Corrections in column 5, depending on the case. Column 7 contains the Final Correction. This value is the difference between column 8 and column 3. The consistency, not the absolute values, of the figures in column 7 is a measure of the accuracy of the survey. Column 8 shows the Final Azimuths. These are calculated from the final co-ordinates.

Table IV. List of Coordinates

Sta. No.	E	N
1	2,151.62	3,400.52
2	3,377.00	1,024.28
3	6,770.04	4,380.17
4	6,181.59	1,125.70
X*	2,651.62	5,645.10
Y*	7,519.88	6,645.95
Z*	4,773.33	4,402.27

\* These coordinates calculated, the others are given.

The following is an example of the use of the direction and azimuth adjustment. To be determined: the azimuths to be used in fixing the position of Station X in the net. The coordinates of Stations 1, 2, 3, and 4 are given. The Observed Di-

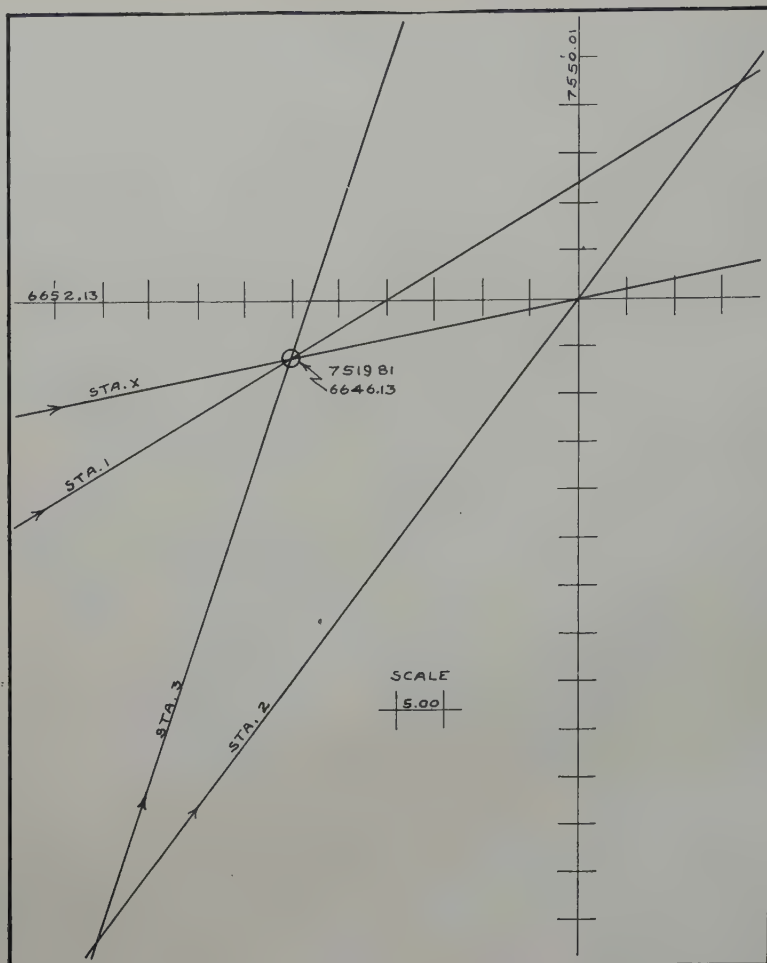


Fig. 6—First error figure for Station Y.

Note the obviousness of the errors in the answers depending on the azimuth from Station 2.

rections in column 2 are given. The coordinates of Stations X, Y, and Z are unknown.

Step 1: Compute the azimuths between Stations 1, 2, 3, and 4 for those directions which have been observed, and enter these values in column 8.

Step 2: Orient in column 3, to a value near those in column 8, the Observed Directions of column 2. Thus under the observations from Station 1, the direction to Station 4 has been oriented 10" lower than the value in column 8, and the other directions have been oriented accordingly.

Step 3: Subtract the values in column 3 from those in column 8 and enter the results in column 7. Mean the values in column 7 and enter the results at the bottom of the blocks. Thus under Station 2 the Final Corrections to Stations 1, 3, and 4 are +25", +26", and +30", respectively, and the mean is +27".

Step 4: Add the means from column 7 to the Approximate Azimuths of Station X as found in column 3 in the blocks under Stations 1, 2, and 3, and enter these values in column 6. The degrees of column 3 are understood to apply to columns 4 and 6. The values in column 6 now give the most probable values for the azimuths, from Stations 1, 2, and 3 to Station X, without consideration of the Observed Directions at Station X. The next steps combine these results with the Observed Directions at Station X.

Step 5: Abstract those values from column 6

which appear opposite Station X under Stations 1, 2, and 3, and enter them in column 4 under Station X opposite the appropriate station. Thus from column 6 under Station 3, opposite Station X, abstract 04' 28" and enter in column 4 under Station X and opposite Station 3.

Step 6: Orient the Observed Directions under Station X to be near the values in column 4 and enter in column 3. The degrees will be 180° from those in the previous blocks.

Step 7: Subtract the values in column 3 from those in column 4 and enter in column 5.

Step 8: Mean column 5 and add the mean to the values in column 3, entering the results in column 6. This is done for the Approximate Azimuths of all stations having coordinates known at this moment. Thus Station 4 is brought into the picture, at this point, as an azimuth observed in one direction only. In computing the position of Station X the mean values between columns 4 and 6 are used. When the coordinates of Station X are fixed, the Final Azimuths to Stations 1, 2, 3, and 4 are computed and entered; the Final Correction is calculated and meaned, or remeaned as in the blocks under Stations 1, 2, and 3, and the sheet is ready for the abstraction of the information needed to compute Station Y.

(7) **Applications:** There are in general three problems which arise in the computation of triangulation. They are: (1) the unknown station is



occupied and two or more of the azimuths are observed in both directions; (2) the unknown station is not occupied but is observed from two or more stations; (3) the unknown station is occupied but is not observed both ways along a common azimuth. The first problem is, of course, the one for which to strive. The closed triangle and the closed quadrilateral are two special versions of this problem. The second problem is that of the point determined by intersection. The third problem is the three point problem. A point may be determined by any of these problems or by a combination of them.

Problem No. 1, the closed figure: the solution for Station X illustrates this case. The mean of the Forward and Backward Azimuths from the Direction and Azimuth sheet (table III) are used. There are four azimuths to Station X and therefore six possible intersections. The first intersection, that of Station 1 and Station 4, is computed by the cotangent method. Since the signs of the azimuths are opposite it makes no difference which is taken as  $\alpha$ , since the sign of  $(\cot \alpha - \cot \beta)$  will be the same. Computing the intersections of the azimuths to Stations 2 and 3 (with the coordinate axes determined by this first intersection) and plotting the results determines the five other intersections (fig. 4). The error figure is then evaluated.

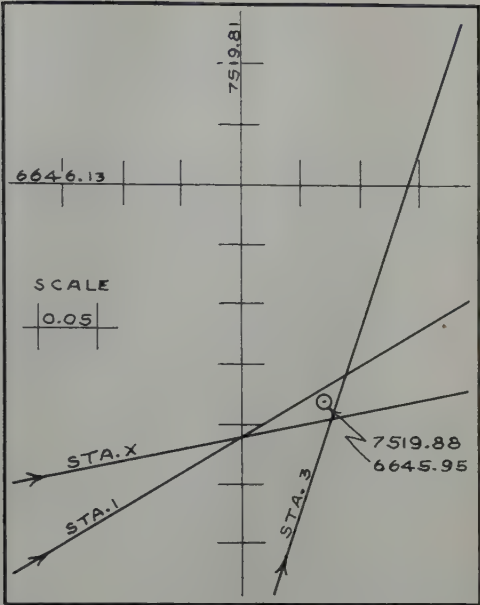
Sta. 1	2,151.62	3,400.52	192° 33' 30"	+4.489074
Sta. 4	6,181.59	1,125.70	142° 00' 25"	-1.280262
				+5.769336
Sta. X	2,651.608	5,645.00		
Sta. 2	3,377.00	1,024.28	171° 04' 41"	-0.156988
	-0.01	2,651.60	4,620.72	
Sta. 3	6,770.04	4,380.17	107° 04' 27"	+0.307147
	4,118.43	5,645.13	+0.13	

Problem No. 2, the intersection: the solution for Station Y is an illustration of this case. In this illustration the azimuth from Station 2 has an error of nine minutes. A similar result would occur if the coordinates of Station 2 were in error, or if a like error were made in computation. There are six possible answers to this problem; three of which are wrong. An examination of the numerical results of the first set of calculations will show that there is an error. By plotting the information (fig. 6) the wrong answers are immediately apparent. An approximately correct coordinate is scaled, and the intersections are recomputed without the need of looking up any new functions. The results are plotted (fig. 7). The calculations involving Station 2 are of course checked for errors in computation.

In calculating the first intersection the azimuth from Station 2 must be taken as  $\alpha$  so that its sign and that of  $(\cot \alpha - \cot \beta)$  will be the same.

Sta. 2	3,377.00	1,024.28	36° 33' 24"	+1.348631
Sta. X	2,651.62	5,645.10	78° 22' 58"	+0.205584
				+1.143047
Sta. Y	7,550.010	6,652.13		
Sta. 1	2,151.62	3,400.52	58° 50' 40"	+0.604562
	5,398.39	6,664.18	+12.05	
Sta. 3	6,770.04	4,380.17	18° 18' 42"	+0.330944
	-28.08	7,521.93	2,271.96	

Fig. 7—  
Final error  
figure for  
Station Y.



Check Computation:				
Sta. 2	3,377.00	1,024.28	36° 33' 24"	+0.741493
	±0.00	7,550.01	5,627.85	
Sta. X	2,651.62	5,645.10	78° 22' 58"	+4.864197
	±0.00	7,550.01	1,007.03	
Recompute:				
Sta. Y	7,519.81	6,646.13	(Scaled from fig. 6)	
Sta. X	2,651.62	5,645.10	78° 22' 58"	+0.205584
	4,868.19	6,645.92	-0.21	
Sta. 1	2,151.62	3,400.52	58° 50' 40"	+0.604562
	5,368.19	6,645.92	-0.21	
Sta. 3	6,770.04	4,380.17	18° 18' 42"	+0.330944
	+0.14	7,519.95	2,265.96	

Problem No. 3, the three point problem: the solution for Station Z is an example of this case. There are numerous solutions of the three point problem but regardless of the solution used the essential part of the solution is the determination of the proper orientation of the observed directions. One solution is the auxiliary point method or Collin's solution;<sup>17</sup> the usual solution is the auxiliary angle method or Chauvenet's solution.<sup>18</sup> Of the two methods Breed and Hosmer state that the second "... is shorter and has a better arrangement for logarithmic computation".<sup>19</sup> While it is correct that the arrangement of the auxiliary angle solution is better for logarithmic computation, it is not the shorter solution if the cotangent method is used in solving the auxiliary point method. In the auxiliary angle solution there are six logarithms and two antilogarithms to look up to find the angles used in orienting the directions. In the auxiliary point solution there are two functions and one antifunction to look up. While in the auxiliary angle method the distances of two sides must be computed as a part of the preparation of the problem, in the auxiliary point solution no distances need be computed.

Having determined the proper orientation, solution of the triangle by the cotangent method gives the first approximate coordinate for Station Z, and solution for the intersection of the third azimuth (it should be 0.00 units) gives a check of the computation. It will be found that for certain azimuths

the tabular differences of the tangent and the co-tangent will introduce a variation up to 0.01 units in the intercept. Since this is well within the limits of the error of observation the calculation may be considered checked.

Any three angles and any three coordinates will give a self consistent answer in a three point problem. A fourth observation is thus needed as a check. This fourth observation will give three more possible answers since neither the positions nor the angles are perfect. All four answers are equally valid, and the most nearly correct position must be the one that most nearly satisfies them all. To solve for all of these answers, numerically, by either method discussed runs into a lot of work. A semigraphical solution is therefore preferable.

Lehmann's solution,<sup>20, 21, 22, 23</sup> so commonly used in plane table work, is very satisfactory. The intercepts of the additional azimuths are computed and plotted (fig. 9). The azimuths are then increased (or decreased) by an arbitrary amount (five seconds in the example) and these results are plotted (the dashed lines). A line (V-line) connecting similar vertices of these two orientations is then the line of all possible intersections of these two lines in all possible orientations. The V-line is actually a segment of a circle but for the short interval considered a straight line is used. Draw all of the V-lines and the points of intersection of three V-lines are possible solutions of the three point problem. In each case the three intersecting V-lines must have the same sense.

In the calculation of the auxiliary point Q, in problem Z, it will be found that the easting value is a negative sum. An addition of 20,000 units to the easting and northing of Stations 1, 4, and Y will place the computation completely in the first quadrant. In the solution of the triangle for point Q the azimuth from Station 1 must be taken as  $\alpha$  since both azimuths have the same sign. Since this sign is negative the revolution dials must be operated with a negative setting.\*

\* Calculating machines not having the carry over mechanisms on the revolution dials are usually equipped with a set of red numerals for negative operations.  $E_4$  is then set in the red numerals and these are maintained on the revolution dials during the calculation.

It is seen from the calculation that the azimuth from Station 3 does not check the computation. A quick recomputation of this azimuth shows that

there is no error in the calculation. Therefore the error must be either in the coordinates of Station 3 or in the observation. Since we know from the previous work that the coordinates of Station 3 are consistent with the net, the observation must be in error. It is therefore disregarded. This leaves five observations, giving a total of ten possible answers. It will be seen that the V-lines giving the solution using Stations 1, 2, and 4 approach parallelism, an indication that Stations 1, 2, and 4 determine a circle passing close to Station Z. This approaches the indeterminate case, so in evaluating the error figure this point is disregarded. (See table below.)

From Station Z six observations were made. There are then, to start with, twenty possible answers. Since the observation to Station 3 is in

Sta. 1	00° 00' 00"	(Abstracted from table III)			
Sta. Y	161° 40' 01"				
Sta. 4	267° 39' 21"				
Z4—ZY	= 105° 59' 20"	= $\gamma$	(See fig. 8)		
Z1—Z4	= 92° 20' 39"	= $\delta$			
Azimuth 1Y	= 58° 50' 40"				
Azimuth YQ	= 58° 50' 40" + 180° + (180° — 92° 20' 39")	= 326° 30' 01"			
Azimuth IQ	= 58° 50' 40" — (180° — 105° 59' 20")	= 344° 50' 00"			
Sta. 1	22,151.62	23,400.52	344° 50' 00"	—3.689093	
Sta. Y	27,519.88	26,645.95	326° 30' 01"	—1.510851	
				—2.178242	
Q	16,938.209	42,633.28			
Sta. 4	26,181.59	21,125.70			
Q	16,938.21	42,633.28	9,243.38/—21,507.58	= —0.429773	
	+9,243.38	—21,507.58	tan (—0.429773)	= 156° 44' 36"	
Sta. Y	7,519.88	6,645.95	50° 45' 16"	+0.816905	
Sta. 4	6,181.59	1,125.70	156° 44' 36"	—2.326817	
				+3.143722	
Sta. Z	4,773.388	4,402.33			
			(20")	+(0.382085)	
Sta. 1	2,151.62	3,400.52	249° 05' 15"	+0.382113	
	2,651.77	4,402.33	$\pm 0.00$		
		(4,402.26)(—0.07)			
			(46")	—(0.585824)	
Sta. X	2,651.62	5,645.10	300° 21' 41"	—0.585791	
	2,121.77	4,402.19	—0.14		
		(4,402.12)(—0.21)			
Sta. 3	6,770.04	4,380.17	90° 48' 15"	+0.014036	
	1,996.65	4,408.19+5.86			
			(42")	+(0.413430)	
Sta. 2	3,377.00	1,024.28	202° 27' 37"	+0.413402	
+0.10	4,773.49	3,378.05			
(+0.20)	(4,773.59)				
Sta. Y	7,519.88	6,645.95	50° 45'(21")	—(0.816865)	
	2,746.49	(4,402.44)(+0.11)			
Sta. 4	6,181.59	1,125.70	156° 44'(41")	—(0.429743)	
(+0.09)	(4,773.48)	3,276.63			

error, there are ten wrong answers out of twenty. There is a one to one chance of picking a wrong combination for the first try; there is a nine to ten chance of picking a wrong answer on the second try if the problem were solved entirely by numerical calculation. If the second try is a right answer the third try must be a correct one if there is to be a check, and the chances of this occurring are again one to one if the first try were wrong, and the second try were right. Lehmann's semigraphical



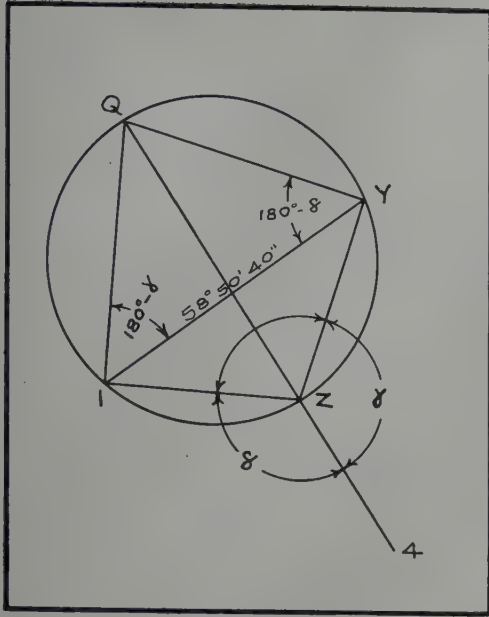


Fig. 8—  
Collins'  
solution of  
the three  
point  
problem.

solution avoids all of this calculation. If the first calculation had used the observations to Stations 3, 4, and Y, none of the extra intercepts would check (fig. 10). The rotated azimuths are calculated and plotted, and the V-lines are drawn. The vast majority of the intersections fall in the heavily circled area. A position is scaled here, and a recomputation will give the answer in fig. 9. An inspection of the V-lines passing through the other intersections show that the observation to Station 3 is involved in all of them. The error, whether of computation, of coordinate, or of observation is thus quickly localized.

#### (8) Derivations: Cotangent Formulas<sup>24</sup> (fig. 3):

Given Station A ( $E_A, N_A$ ), Station B ( $E_B, N_B$ ), and azimuths  $\alpha$  and  $\beta$  from Stations A and B to Station C.

Find Station C ( $E_C, N_C$ ).

- (1)  $\frac{N_C - N_A}{E_C - E_A} = \cot \alpha, \quad \frac{N_C - N_B}{E_C - E_B} = \cot \beta$
- (2)  $N_C = N_A + (E_C - E_A) \cot \alpha = N_B + (E_C - E_B) \cot \beta$
- (3)  $N_A + E_C \cot \alpha - E_A \cot \alpha = N_B + E_C \cot \beta - E_B \cot \beta$
- (4)  $E_C (\cot \alpha - \cot \beta) = (N_B - N_A) - E_B \cot \beta + E_A \cot \alpha$

Add and subtract  $E_B \cot \alpha$  to the right hand member of Eq 4.

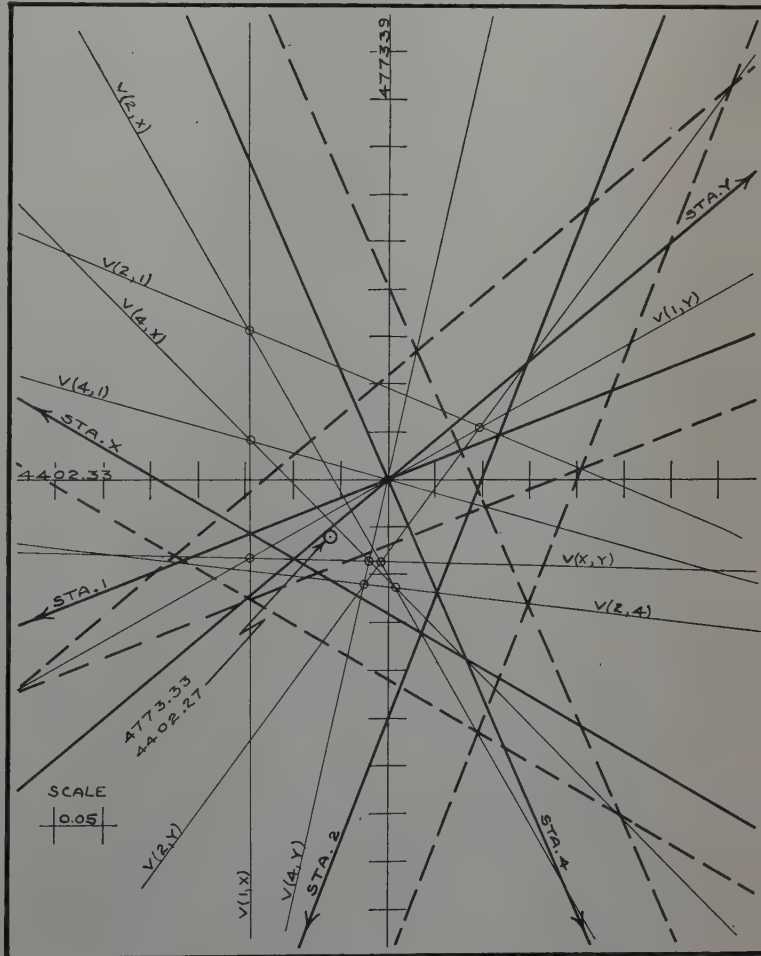
- (5)  $E_C (\cot \alpha - \cot \beta) = (N_B - N_A) - E_B \cot \beta + E_A \cot \alpha + E_B \cot \alpha - E_B \cot \alpha$
- (6)  $E_C (\cot \alpha - \cot \beta) = E_B (\cot \alpha - \cot \beta) + (N_B - N_A) - (E_B - E_A) \cot \alpha$

Fig. 9—Error figure for  
Station Z.

- (7)  $E_C = E_B + \left[ \frac{(N_B - N_A) - (E_B - E_A) \cot \alpha}{(\cot \alpha - \cot \beta)} \right]$
  - (8)  $N_D = N_A + (E_B - E_A) \cot \alpha$
  - (9)  $N_A = N_D - (E_B - E_A) \cot \alpha$
  - (10)  $E_C = E_B + \left[ \frac{(N_B - [N_D - (E_B - E_A) \cot \alpha]) - (E_B - E_A) \cot \alpha}{(\cot \alpha - \cot \beta)} \right]$
  - (11)  $E_C = E_B + \left[ \frac{N_B - N_D + E_B \cot \alpha - E_A \cot \alpha - E_B \cot \alpha + E_A \cot \alpha}{(\cot \alpha - \cot \beta)} \right]$
  - (12)  $E_C = E_B + \left[ \frac{N_B - N_D}{(\cot \alpha - \cot \beta)} \right]; E_C = E_B; E_B = E_D$
- Add and subtract  $E_B \cot \alpha$  to the right hand member of Eq. 2.
- (13)  $N_C = N_A + (E_C - E_A) \cot \alpha + E_B \cot \alpha - E_B \cot \alpha$
  - (14)  $N_C = N_A + (E_B - E_A) \cot \alpha + (E_C - E_B) \cot \alpha$
- Substitute Eq 8 in Eq 14.
- (15)  $N_C = N_D + (E_C - E_B) \cot \alpha; E_B = E_D$

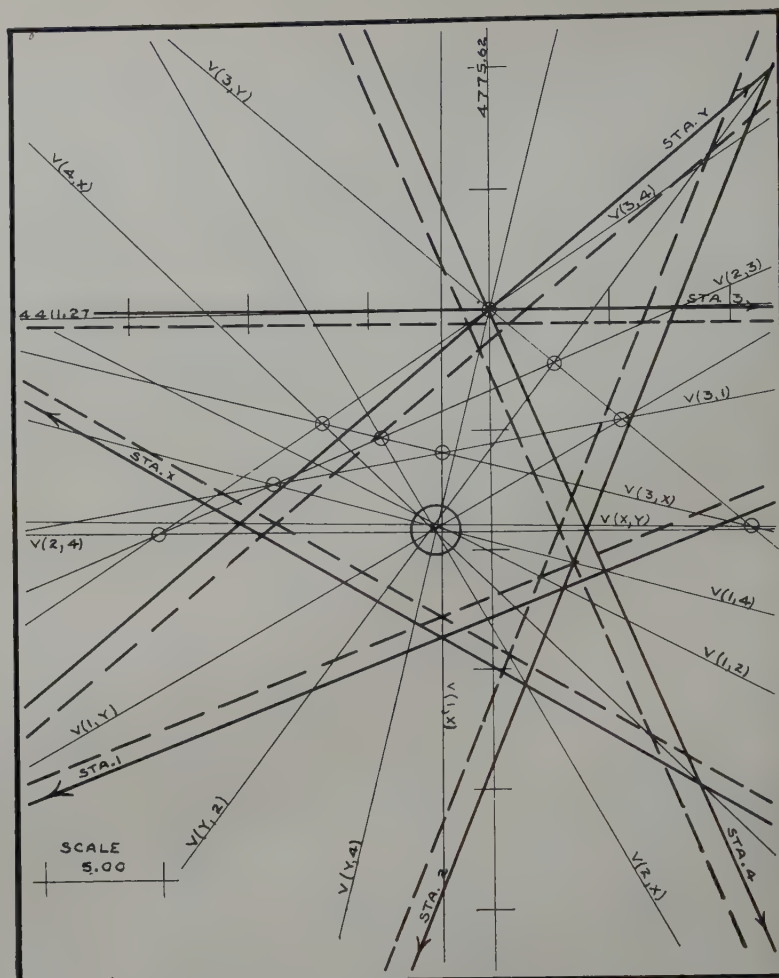
Semigraphic Intersection Formulas<sup>25</sup> (fig. 5).

Given Station A ( $E_A, N_A$ ) and Station B ( $E_B, N_B$ ) and the azimuth from A to B.



**Fig. 10—Error figure for Station Z, using Stations 3, 4, and Y for the first solution.**

Note that Station 3 is involved in all of the solutions falling outside of the heavily circled area.



Then (1)  $E_B - E_A = (N_B - N_A) \tan \alpha$

(2)  $N_B - N_A = (E_B - E_A) \cot \alpha$

The intersection of  $\alpha$  with  $E_B$  is then

(3)  $E_B = E_A + (N_B - N_A) \tan \alpha$

and the intersection of  $\alpha$  with  $N_B$  is

(4)  $N_B = N_A + (E_B - E_A) \cot \alpha$

(9) **Acknowledgments:** Mr. G. B. Lauf, Senior Lecturer, Department of Surveying, University of Witwatersrand, Milner Park, Johannesburg, Union of South Africa, provided the author with reference lists<sup>20</sup> and a very helpful correspondence. The Librarian of the University of the Witwatersrand provided photostatic copies of some of the material apparently unavailable in the United States. The Director of the Trigonometrical Survey of the Union of South Africa provided a reference list<sup>20</sup> and mimeographed notes. The above assistance enabled the author to gather the material for this article. Mrs. E. A. Cheves, Librarian, Miami Public Library, Miami, Ariz., undertook the painstaking chore of locating those references which could be found in North America. The author is indebted to Mr. C. D. Huffine, Chief Engineer, Inspiration Consolidated Copper Co., for his critical reading of the manuscript.

#### References

<sup>1</sup> O'Farrel: An Account of the Graphical Method in Use for Determining the Coordinates of the Secondary Trigonometrical Stations. (1866).

<sup>2</sup> Jules Baillaud: Manuel de Topometrie. Dunod, Paris. (1920).

<sup>3</sup> Mimeographed notes provided by the Director of the Trigonometrical Survey, Union of South Africa. Triangulation, Traversing and Leveling, Chap. 8, p. 7.

<sup>4</sup> Op. cit., Ref. 3, Chap. 8, pp. 7-8.

<sup>5</sup> B. W. C. Van Der Steer: The Determination of Coordinates by the Graphical Method. *Jnl. of the Institute of Land Surveyors of the Transvaal*. 1, 36, 91, 117; 111, 241.

<sup>6</sup> Surveying: Technical Manual No. 5-235, pp. 282-287, pp. 294-301, War Department. (1940).

<sup>7</sup> Op. cit., Ref. 6, p. 283.

<sup>8</sup> Survey Computations. His Majesty Stationery Office, London. (1932) p. 58.

<sup>9</sup> Explanatory Notes on the Methods and Forms of Survey Calculations. Appendix to the Land Survey Act No. 9 of 1927, Union of South Africa. pp. vi, viii.

<sup>10</sup> Op. cit. Ref. 9, p. viii.

<sup>11</sup> Op. cit. Ref. 2, p. 79.

<sup>12</sup> Op. cit. Ref. 2, p. 79.

<sup>13</sup> Op. cit. Ref. 3, Chap. 7, p. 15.

<sup>14</sup> Oscar S. Adams: Application of the Theory of Least Squares to the Adjustment of Triangulation. U. S. C. & G. S. Special Publication No. 28. United States Government Printing Office. (1915).

<sup>15</sup> Op. cit. Ref. 3, Chap. 7, pp. 9-12.

<sup>16</sup> Op. cit. Ref. 9, p. vi.

<sup>17</sup> Op. cit. Ref. 8, p. 29.

<sup>18</sup> William Chauvenet: A Treatise on Plane and Spherical Trigonometry, J. B. Lippincott Co. (1896) p. 82.

<sup>19</sup> Charles B. Breed and George L. Hosmer: The Principles and Practice of Surveying. John Wiley and Sons, Inc. (1938). Vol. 11, p. 68.

<sup>20</sup> Op. cit. Ref. 19, pp. 260-262.

<sup>21</sup> Op. cit. Ref. 2, pp. 99-101.

<sup>22</sup> Op. cit. Ref. 6, p. 297.

<sup>23</sup> Op. cit. Ref. 3, Chap. 7, p. 18.

<sup>24</sup> W. Jordan: Handbuch der Vermessungskunde, 9. Aufl. O. Eggert, Stuttgart (1931). Vol. 2 (1) pp. 425-7.

<sup>25</sup> Op. cit. Ref. 8, p. 58.

<sup>26</sup> The following references were unavailable to the author. J. J. Bosman: Tertiary Triangulation and Traverse Surveys. Juta, Cape Town. (1914).

C. F. Gauss: Werke, Vol. 4, pp. 396-7.



Developments in the

## Application of Activated Carbon to Cyanidation

Including the Desorption of Gold and Silver from Carbons

by E. H. Crabtree, Jr.,

V. W. Winters, and

T. G. Chapman

This paper traces the experimental and pilot plant work completed by the authors since 1939 including the various methods which have been developed in applying coarse activated carbon to cyanidation. The methods and results developed by the authors in desorbing gold and silver from carbons are given and the possibilities for the re-use of the desorbed carbons are discussed.

IN 1939 one of the authors\* described advances in carbon-cyanidation for the period 1932 to 1939

\* T. G. Chapman: A Cyanide Process Based on the Simultaneous Dissolution and Adsorption of Gold. *Trans. AIME* (1939) 134, 207.

and included: (1) the dissolution of gold in an ore pulp by cyanide and its simultaneous adsorption by carbon, (2) the stage addition of adsorptive carbon and its movement countercurrent to the flow of the ore pulp, and (3) the recycling of adsorptive carbon. Considerable additional experimental and pilot plant work have been done by the authors since 1939 and the results of such work

E. H. CRABTREE, JR., Member AIME, is Director of Milling, Eagle-Picher Mining and Smelting Co., Miami, Okla., V. W. WINTERS is Superintendent, Sahuarita Concentrator, Eagle-Picher Mining and Smelting Co., Tucson, Ariz., and T. G. CHAPMAN, Member AIME, is Dean, College of Mines, University of Arizona, Tucson.

AIME San Francisco Meeting, February 1949.

TP 2744 B. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received March 3, 1949.

have been sufficiently encouraging to justify a second paper.

**Developments from 1939 to 1948:** In 1941, Verne W. Winters did the first experimental work in eliminating agitation during dissolution and adsorption and thereby demonstrated that the dissolution of gold by cyanide and its simultaneous adsorption by carbon take place essentially to completion in a stagnant pulp. Shortly thereafter, and as a result of the stagnant pulp experimental work, Winters and Chapman demonstrated that soluble gold, in a cyanide ore pulp, would migrate from the ore pulp through a perforated container and be adsorbed by carbon within the container.

During the same year, the Eagle-Picher Mining and Smelting Co. erected a 25-ton capacity pilot plant at the Harquahala mine about 9 miles from Salome, Ariz. Experimental work at this plant comprised chiefly the stagnant pulp modification of carbon-cyanidation although considerable work was done in the grinding and activation of carbon, and testing various methods of extracting adsorbed gold from carbon.

Soon after the completion of the Harquahala

experimental work, the Eagle-Picher Mining and Smelting Co. erected a continuous, small capacity, pilot plant at its Sahuarita concentrator, about 20 miles from Tucson, Ariz., to continue experimental work with various modifications of carbon-cyanidation. This plant has operated almost continuously since this time.

At the conclusion of the war, the authors experimented with the Government war surplus, coarse gas-mask carbon. Several flowsheets employing such carbon were tried for the adsorption of dissolved gold from ore pulps including the stage addition and countercurrent movement of the carbon in pulps. Among these methods may be men-

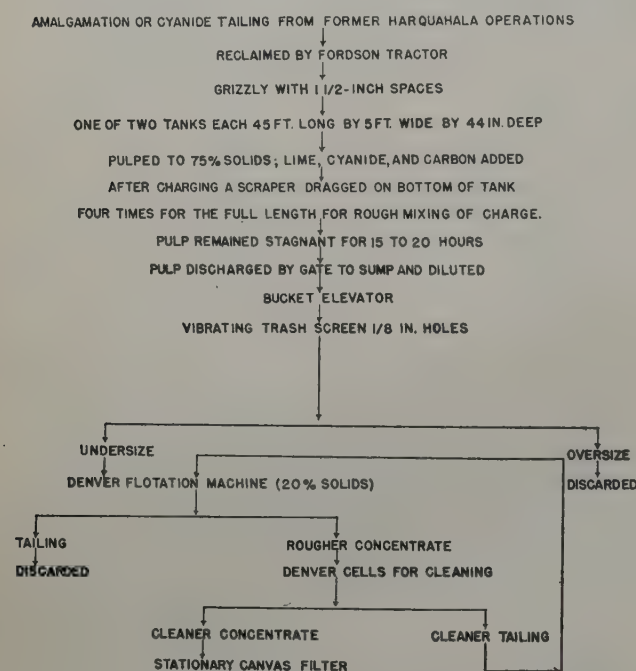


Fig. 1—Stagnant pulp modification, Harquahala pilot plant.

tioned: (1) coarse carbon, ground to approximately 100 mesh was added to the ore pulp and separated from the pulp by flotation; (2) coarse carbon\* was

\* The coarse carbon used for all methods described varied in size but usually was minus 10 plus 35 mesh.

added to a stagnant ore pulp and separated from the pulp by screening; (3) coarse carbon was added to an agitated ore pulp and separated from the pulp by screening; and (4) perforated cylinders, containing coarse carbon, were placed in agitators and partially submerged into the ore pulps. The containers were slowly revolved for the adsorption of soluble gold and in a series of agitators the carbon, with or without the containers, was moved from agitator to agitator countercurrent to the flow of the pulp.

During the progress of the work involving the use of coarse adsorptive carbon it was realized by the authors that if such adsorptive material were to be employed on a commercial basis either a lower cost carbon than that available during the testing period must be provided or the problem of desorbing the gold from the carbon and the re-use of the carbon must be solved. Referring to the availability

of a lower-cost coarse carbon, although considerable progress has been made in reducing the cost, insofar as the authors are informed, the cost of such carbon is still too high to warrant employing it once and destroying it by burning for the recovery of its adsorbed gold. Recent work has demonstrated that carbon derived from ore pulps can be almost completely desorbed of its gold and silver and the carbon re-used for further adsorption.

It is believed, therefore, that a complete carbon-cyanide process, with several modifications, is now available which offers considerable promise from the standpoints of: (1) low capital cost of plant; (2) extractions, as compared to standard cyanide methods, are at least as high in the treatment of fast settling ores and higher in the treatment of clayey ores and some refractory ores; (3) the production of bullion without burning the carbon and smelting the resultant ash is feasible; (4) pregnant solutions resulting from desorption of carbon amount to only 10 to 15 tons per 100 tons of ore treated, thus greatly reducing the size of the gold precipitate unit; (5) a process which can treat slow-settling as readily as fast-settling ores; (6) the flotation modification recovers some forms of insoluble gold while floating the carbon and its soluble gold content; (7) during the last stages of adsorbing soluble gold it is not necessary to maintain free cyanide in the pulp whereas it is necessary to maintain free cyanide in solutions during conventional precipitation with zinc; (8) loss of efficiency in the adsorption stage due to foul solutions has not been encountered in carbon-cyanidation; (9) in the treatment of ores containing graphite or pyrrhotite, loss of soluble gold adsorbed on graphite and loss by premature precipitation by the decomposition products of pyrrhotite does not occur, at least not to the extent it does with standard cyanide methods; and (10) in the presence of certain cyanicides such as copper sulphide the stage addition of cyanide to maintain low concentrations of cyanide shows promise in reducing the consumption of cyanide.

The brief descriptions and data which follow have been taken from plant work completed since 1939 to indicate the progress which has been made with the various modifications of carbon-cyanidation.

**Stagnant Pulp Modification:** The Eagle-Picher Mining and Smelting Co., as previously mentioned, operated a 25-ton capacity pilot plant to confirm results which had been obtained in the laboratory in eliminating agitation during the dissolution of gold from an ore pulp by cyanide and its adsorption by carbon. The plant was erected at the Harquahala mine and operated on very slow settling clayey amalgamation or cyanide tailing from former operations. Fig. 1 gives a flowsheet of the Harquahala test plant. The reclaimed tailing was charged to one of two concrete tanks provided for the dissolving and adsorbing stages. Each tank was 45 ft long by 5 ft wide by 44 in. deep and held 25 tons of dry tailing as a pulp containing 75 pct of solids. The method of operating was as follows: The tailing was reclaimed and conveyed to a grizzly having 1½ in. spaces by a Fordson tractor equipped with Ferguson system hydraulic scraper control.



To fill a tank, 32 scraper loads, conveyed a distance of 150 to 200 yards, were required. Water, lime, and cyanide were first added to the tank after which one-third of the tailing was charged and activated carbon added to the surface. Tailing was again charged until the tank was two-thirds full and more carbon added to the surface. The final charge of tailing was then added with the remaining carbon added to the surface. The grizzly, shown in fig. 1, with scraper attached was then pulled back and forth twice for mixing the charge. The charge then stood, without further agitation, for a period of 15 to 20 hr after which the pulp was discharged to a sump and reclaimed by a bucket elevator. The bucket elevator discharged onto a 4x5 ft vibrating trash screen with 1/8 in. holes and the undersize was treated in Denver flotation cells for the separation of the carbon and its adsorbed values from the pulp.

Lime, cyanide, and carbon were added to the charge at the rates of 0.4, 0.5, and 3.6 lb per ton of tailing, respectively. Soft or hard wood charcoal, ground to from 48 to 150 mesh and activated at the plant by heating to 1400°F and cooling without quenching, was used.

Approximately 4000 tons of this material were treated on an experimental basis. The heads assayed from 0.085 to 0.11 oz gold per ton. The extraction varied from 66 to 79 pct and the carbon flotation concentrate assayed from 16 to 23 oz gold per ton. Barren solutions varied from 0.0042 to 0.0005 oz gold per ton depending upon the fineness of grinding the carbon and its adsorptive efficiency.

Table I gives typical results obtained when minus 150 mesh carbon was used and indicates results obtained with proper fineness of grinding but with rather crudely activated carbon.

**Table I. Harquahala Test Plant Results, Tanks 129 to 137 Inclusive.**

	Tons	Assay, Oz Gold per Ton	Gold, Oz	Gold, Pct
Heads	189.0	0.089	16.739	100.0
Carbon concentrate	0.5744	21.76	12.499	74.7
Barren solution	756.0	0.0009	0.680	4.1
Tailing	188.4	0.019	3.560	21.2

The following observations are offered with respect to the stagnant pool method of carbon-cyanidation and its potential usefulness.

1. The speed of dissolving and adsorbing gold without agitation is much faster than might be expected and is explained by the lower concentration of soluble gold surrounding the carbon particles as compared to the concentration of gold in other parts of the pulp. Similar to the flow of water in a pipe due to a difference in pressure, ions will migrate in an ore pulp if a difference in concentration is maintained between two parts of the pulp. The continuous adsorption of gold by carbon maintains this difference in concentration.

2. The results given in table I were obtained when a proper size but a rather inferior, cheap activated carbon was used. Experience obtained at the Harquahala test plant indicated that it was difficult to float carbon coarser than 100 mesh especially in the more dilute pulp of the cleaner circuit.

3. Due to the imperfect mixing of the charge, the assay of barren solutions varied to a considerable degree. It is believed that a very short agitation period, say 10 to 15 min, after the reclamation of the pulp from the stagnant pool and before flotation, would equalize irregularities due to imperfect mixing and yield not only more uniform barren solutions but also lower value barren solutions.

4. Steam activated carbon would, no doubt, have yielded lower barren solutions than the crudely activated carbon employed at Harquahala. It is questionable, however, that the higher cost steam-activated carbon would have been better from the economic viewpoint at the time the tests were made.

5. There is a potential field of use for such a method of carbon-cyanidation in the leaching of gold from tailing piles in place. If mechanical methods can be devised to pulp the surface of such tailing piles to a consistency of, say, 75 to 80 pct solids the reactions of dissolving and adsorbing gold will, it is believed, take place without agitation.

**Flotation Modification:** Flowsheets for the modification of carbon-cyanidation employing finely ground carbon for adsorption and flotation for the separation of the carbon from the ore pulp have been published.\* This method may be used with

\* T. G. Chapman: A Cyanide Process Based on the Simultaneous Dissolution and Adsorption of Gold. *Trans. AIME* (1939) 134, 207.

one or more stages of adsorption. With multiple stage adsorption the carbon is moved countercurrent to the flow of the ore pulp.

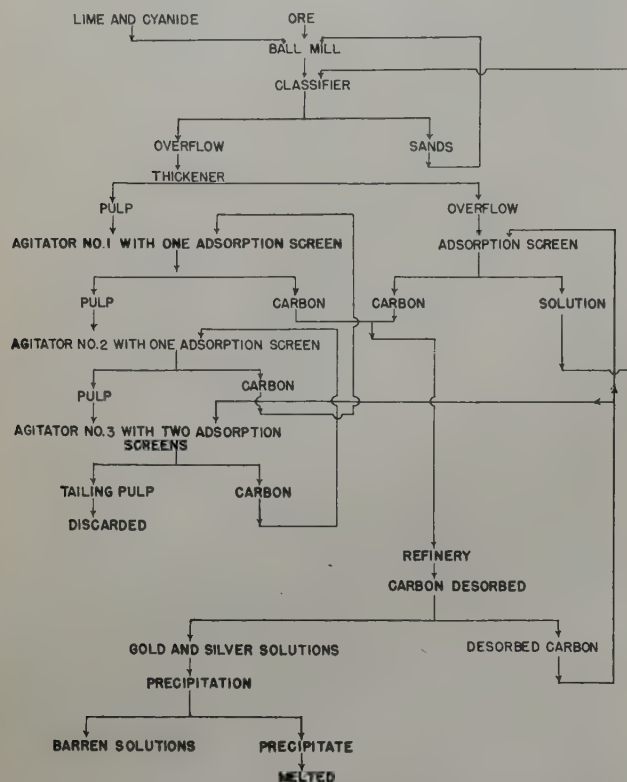
There are at least two advantages to this method as compared to other modifications. First, a soft, rather crudely activated wood carbon may be used as the increased surface of the finer carbon apparently nearly compensates for the crude activation and, second, some minerals which contain gold, insoluble in cyanide, may be recovered in the flotation carbon concentrate.

**Adsorption With Revolving Screens Containing Carbon:** A general flowsheet of this modification is given in fig. 2. It should be noted, however, that the flowsheet, as given in fig. 2, may be changed to meet the requirements of the specific ore to be treated.

G. H. Roseveare, of the Arizona Bureau of Mines, working with Chapman in the laboratories of the University of Arizona was the first, to the author's knowledge, to construct a horizontal, revolving-screen test unit attached to a laboratory mechanical-type agitator. A few months later Chapman and others did the experimental work involving the permissible volume of carbon in a screen and the size of screen required in mill operation. A 5x5 ft Wallace agitator and a 12x15 in. revolving screen were used for this work.

This modification differs from other modifications in that the steps of dissolving the gold, adsorbing the dissolved gold, and separating the carbon from the ore pulp are combined and take place as a continuous process in a single step. It should be noted that the combining of the three steps was described for a stagnant pulp and that the revolving screen only served to make the batch process of the stagnant pulp modification a continuous process. The

Referring to fig. 2, the data presented in this paper for this modification were obtained without the classifier in the grinding circuit and without the thickener following the grinding circuit. The reason for these changes in the flowsheet was the use of Harquahala tailing for testing which did not require grinding or thickening. The Sahuarita test plant flowsheet included a 16 x 32 in. Denver ball mill operated with a light ball load for pulping the tail-



ing. The mill was fed at the rate of 60 lb of dry tailing per hour pulped to 37 pct solids. The ball mill discharged to a 20-mesh vibrating trash screen and the screen undersize was pumped to the first of three 36 x 36 in. Wallace agitators placed in series. Agitators 1 and 2 were each equipped with one 28-mesh stainless steel screen and agitator 3 had two screens. The screens, 9¾ in. in diameter and 5 in. long, were revolved at 35 rpm and were submerged approximately 2½ in. into the ore pulp. Each agitator provided a detention time of 8 hr. The countercurrent movement of the carbon was accomplished by moving the screens at 3-hr intervals from agitator 3 to 2 and from agitator 2 to 1. The carbon on leaving agitator 1 was desorbed and the desorbed carbon was recycled to the screens of agitator 3.

Table II gives the results of a 63-hr test with this flowsheet. The test was operated as described for 15 hr and for the remaining 48 hr a scavenging screen was added to the tailing launder. The tailing pulp

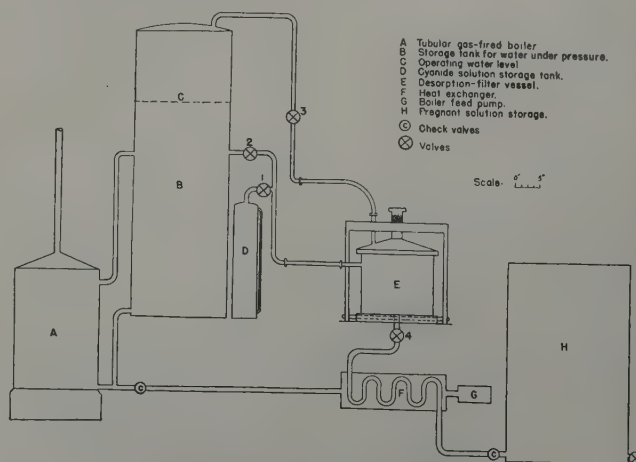
**Table II. Container Modification, Results of 63 hr Test  
at Sahuarita Plant**

	Weight, Lb.	Tons per 100 Tons of Heads	Assay, Oz Gold per Ton	Per Cent
Heads	3,780	100.0	0.096	100.0
Carbon	8.06	0.213	41.86 <sup>a</sup>	92.7
Barren solution	6,426	170.0	0.0012 <sup>b</sup>	2.1
Tailing	3,780	100.0	0.005	5.2

<sup>b</sup> Barren solution from agitator 3 averaged 0.0025 oz gold per ton.

The barren solution given in the tabulation, namely, 0.0012 oz gold per ton, was the solution after treatment in the scavenging screen placed in the tailing launder. The barren solution from agitator 3 is given in footnote b as assaying 0.0025 oz gold per ton. The scavenging screen, with a very short time of contact with the tailing pulp, therefore reduced the assay of the barren solution from 0.0025 to 0.0012 oz gold per ton.

**Desorption of Gold and Silver from Carbon:** The desorption of gold and silver from a carbon that has been in contact with an ore pulp apparently depends upon the shifting of the equilibrium encountered between adsorption and desorption. Four methods of shifting the equilibrium succeeded in desorbing the gold and silver. The four methods comprised: (1) the use of a solvent in conjunction with a large excess of precipitant in order to remove the gold from the receiving solution as the desorption progressed; (2) the use of a solvent in conjunction with electrolysis to accomplish the same purpose; (3) the use of a solvent and large volumes of solutions, added in stages, to keep the concentrations of the gold and silver in the receiving solu-



tions sufficiently low to accomplish desorption; and (4) the use of higher temperatures by employing hot solvent solutions in a pressure chamber. The authors believe that the last method given is the most feasible as this method is very rapid, has the lowest reagent cost, and accomplishes the desorp-



tion with the minimum volume of desorbing solution.

Fig. 3 gives the equipment, designed and constructed by Winters at the Sahuarita plant, for this method of desorption. The equipment consisted of a 1 hp, gas-fired, vertical fire-tube boiler *A*, and a hot water pressure storage tank *B*. The desorption was accomplished in the quick opening pressure filter *E*. Heat exchanger *F* and pregnant solution storage tank *H* completed the essential equipment.

The operating cycle comprised placing the carbon in the filter *E* and adding cyanide solution to the filter from tank *D* through valve 1. Hot water was then added from *B* through valve 2 after which valve 2 was closed and valve 3 opened, which subjected the charge to the full boiler steam pressure of 90 to 95 psi. The cyanide used for each desorbing stage varied from 2 to 20 lb of 90 pct sodium cyanide per ton of carbon treated and the hot water for each desorbing stage amounted to 10 tons per ton of carbon. The soaking period was 20 min after which the closing of valve 3 and the opening of valve 4 forced the pregnant solution through the filter in *E*, through the heat exchanger *F* and into the pregnant solution storage tank *H*. The pipe

tank to prevent "flashing," amounted to 12.32 tons per 100 tons of ore treated which is equivalent to 57.8 tons of pregnant solution per ton of carbon desorbed. The pregnant solution assayed 0.68 oz of gold and 0.48 oz of silver per ton. If the pregnant solution storage tank was operated on a continuous basis, as would be the case in practice, dilution to avoid flashing would not be necessary and the volume of the pregnant solution would be 10.65 tons per 100 tons of ore treated and 50 tons per ton of carbon desorbed.

Assuming an average settling ore with average size of grinding treated by countercurrent decantation, the volume of pregnant solution for precipitation would amount to approximately 300 tons per 100 tons of mill heads. The solution precipitated by standard cyanide methods would therefore be approximately 28 tons as compared to one ton of solution produced by the desorption of carbon.

The desorption of the activated petroleum coke used in this test was not as complete as that obtained with shell carbon as given in table IV. In table III the percentage of gold desorbed was 98.4 for the petroleum coke as compared to an average desorption of 99.5 pct for shell carbon.

Cyanide was added for the test described at the rate of 2 lb per ton of carbon per cycle or a total of 10 lb of cyanide per ton of carbon. With other carbons it was necessary to add greater amounts of cyanide. Tests to determine minimum cyanide requirement for a given carbon have not been made to date.

The maximum capacity of the 10 x 12 in. desorption chamber shown in fig. 3 has not been determined. As operated in tests described in this paper one charge

which discharged the hot pregnant solution into *H* was kept submerged by solution constantly maintained in *H* to prevent flashing. Five cycles as described were used for each charge of carbon.

Table III gives the data for the desorption of the carbon used in the 63 hr test of Table II.

Referring to Table III, the desorption of the gold was 98.4 pct and the desorption of the silver was essentially complete. The volume of the pregnant gold and silver solution, including the diluting water added to the pregnant gold solution storage

of carbon amounted to 2.22 lb and five 20-min soaking periods were used per charge. Assuming a total time of 2 hr for desorption the capacity would be 26.6 lb of carbon per 24 hr. This amount of carbon would be obtained from the treatment of approximately 7.5 to 9 tons of low grade ore. The soaking time per cycle can undoubtedly be decreased so that the capacity given is probably low.

**Comparison of Desorbing Petroleum Coke and Shell Carbon:** Table IV gives the comparative results of desorbing activated petroleum coke and

Table III. Desorption of the Carbon of Table II

	Carbon, Lb	Tons per 100 Tons of Mill Heads	Assay, Oz per Ton		Per Cent	
			Gold	Silver	Gold	Silver
Heads, carbon, screens 11 to 31	8.06	0.213	41.86	29.7	100.0	100.0
Pregnant solution	466.	12.32	0.68	0.48	98.4	100.0
Desorbed carbon	8.06	0.213	0.63	Trace	1.6	0

Table IV. Desorptions of Petroleum Coke and Shell Carbon Compared

Nos. of Screens <sup>a</sup> for Adsorption	Kind of Carbon	No. of Previous Desorptions	Wt. of Carbon Desorbed, Gram	Assays of Carbon Heads to Desorption, Oz per Ton		Assays of Desorbed Carbon, Oz per Ton		Per Cent Desorbed	
				Gold	Silver	Gold	Silver	Gold	Silver
16 to 22 incl.	Petroleum coke	None	1,008	42.76	33.6	0.32	None	99.2	100.0
23 to 29 incl.	Petroleum coke	57 pct fresh, 43 pct desorbed once	1,008	39.76	19.2	0.46	None	98.8	100.0
11 to 15, 30, 31	Petroleum coke	70 pct fresh, 30 pct desorbed once	1,008	43.07	37.2	1.12	None	97.4	100.0
40 to 44 incl.	Shell-Sahuarita activated	None	720	55.44	53.8	0.24	None	99.6	100.0
45 to 49 incl.	Shell-Sahuarita activated	None	720	41.01	49.4	0.136	0.55	99.6	98.9
50 to 56 incl.	Shell-factory activated	None	1,008	40.71	70.6	0.238	2.6	99.4	96.3
65 to 67 incl.	Shell-factory activated	1	432	46.33	52.6	0.27	0.68	99.4	98.7

<sup>a</sup> The screen numbers refer to the screens used for adsorption in the container modification (see fig. 2 and table II), and are given only to identify the carbons used for desorption.

Table V. Recycling of Desorbed Carbons

Nos. of Screens <sup>a</sup>	Time of Test, Hr	Kind of Carbon	New, Desorbed, or Reactivated Carbon Used	Assay of Carbon, After Adsorption, Oz per Ton		Barren Solution Assays, Oz per Ton			
						After Third Agitator		After Scavenging Screen	
				Gold	Silver	Gold	Silver	Gold	Silver
11 to 26	48	Petroleum coke	New, factory activated	43.72	30.5	0.0022	0.007	0.0010	0.004
27 to 33	21	Petroleum coke	Screen 11 to 26 desorbed	37.15	29.4	0.0036	0.010	0.0018	0.006
34 and 35	6	Petroleum coke	Screens 27 to 33 desorbed	36.44	40.4	0.0042	0.010	0.0020	0.005
36 to 39	12	Petroleum coke	Desorbed twice, Sahuarita reactivated	44.35	65.5	0.0032	0.012	0.0011	0.006
40 to 49	30	Shell carbon	New, Sahuarita activated	48.22	51.6	0.0025	0.011	0.0009	0.006
57 to 64	24	Shell carbon	Screens 40 to 49 desorbed	31.37	38.2	0.0028	0.012	0.0014	0.008
50 to 56	21	Shell carbon	New, factory activated	40.71	70.6	0.0017	0.007	0.0008	0.005
65 to 67	9	Shell carbon	Screens 50 to 56 desorbed	46.33	52.6	0.0035	0.013	0.0020	0.010

<sup>a</sup> The screen numbers refer to the screens used for adsorption in the container modification (see fig. 2 and table II), and are given only to identify the carbons used.

shell carbon. The average desorption of gold and silver from petroleum coke was 98.5 and 100 pct, respectively, as compared to 99.5 and 98.5 pct for shell carbon.

Although the results given in Table IV show some increase in the assays of the desorbed residues for the second desorption of the same lot of carbon, the increase is not serious and the repeated desorptions of other carbons, not reported in this paper, did not indicate any build up of gold and silver in the residues after repeated desorptions.

Table V gives a comparison of barren solutions and assays of carbon obtained when freshly activated, desorbed, and desorbed reactivated carbons were employed for adsorption in the container modification.

Referring to the barren solutions, two barren solutions are included in the tabulation. The first is the solution discharged from the third agitator, and the second, the solution leaving the scavenger screen of the tailing launder. The figures in the tabulation show an increase in gold content of the barren solution with petroleum coke and shell carbon in the recycling of the desorbed carbon without reactivation. With petroleum coke, the fresh activated carbon yielded a barren solution of 0.0010 oz gold per ton after the scavenging screen, a barren solution of 0.0018 oz with the recycling of the carbon after one desorption, and a barren solution of 0.0020 oz with the recycling of the carbon after two desorptions. The ratio of solution to solids leaving the scavenger screen was 1.7 to 1. With the reactivation of the carbon, however, after two desorptions the barren solution was reduced to 0.0011 oz. The data in the tabulation is not as complete for the shell carbon but does show an increase in the assay of the barren solution with recycling after one desorption.

The assay values of barren solutions depend not only upon the adsorptive efficiency of the carbon but also upon the kind and length of contact of the carbon with the ore pulp. For example, comparing the barren solutions of the third agitator with those of the scavenging screen it is noted that improved contact between the carbon within the screen and the ore pulp drastically reduced the barren solution loss.

Although the data show that desorbed carbon, which has not been reactivated, has decreased adsorptive speeds it is entirely possible that more effective contact with such carbon will make up to a considerable degree for the loss of adsorptive speed for such carbons. The data indicate that the reactivation of desorbed carbon restores its adsorptive speed to that of fresh carbon.

The authors believe that the loss of adsorptive speed of a desorbed carbon as compared to a fresh carbon is due to the partial filling of the interstitial spaces of the carbon by slime. The reactivation of desorbed carbon, which apparently restores its adsorptive speed, is believed to increase the interstitial spaces by oxidation with steam during reactivation.

**Summary:** This paper describes the use of the carbon-cyanidation process in a stagnant ore pulp in a pilot plant having a capacity of 25 tons per day. Minus 150-mesh activated carbon, with lime and cyanide, were pulped with gold-bearing tailings residues at a consistency of 75 pct solids and allowed to stand for a period of 15 to 20 hr. The carbon was then recovered from the diluted pulp by flotation. About 75 pct of the gold was recovered in the carbon flotation concentrate.

Another method using coarse activated carbon in revolving screen containers partially immersed in a series of three agitators recovered 92.7 pct of the gold in a coarse carbon concentrate assaying 41.86 oz of gold per ton of carbon. This concentrate was then treated by a desorption method described in which 98.4 pct of the gold was extracted in a pregnant solution assaying 0.68 oz of gold per ton of solution. The volume of this solution amounted to only 12.32 tons per 100 tons of ore treated. The desorbed carbon was re-used in succeeding tests.

Comparative desorption tests on activated petroleum coke carbons and shell activated carbons indicated that gold desorption of the shell was slightly better, reaching as high as 99.6 pct of the gold. Tests indicated that the adsorptive efficiencies of the carbons after desorption were not quite as good as originally, but that after two desorptions followed by reactivation, the adsorptive speed was practically restored to that of fresh carbon.

Further work is required to determine just how many times the carbon may be re-used.



# Continuous Countercurrent

## Decantation Calculations

by T. B. Counselman

Continuous countercurrent decantation calculations have always been a headache to the cyanide man (and the chemical engineer) because of the simultaneous equations involved. These are tedious to solve, and there is considerable opportunity for mathematical error. This paper presents a shortcut method for making these calculations, which has been in successful use for twenty years by the author, and which is particularly valuable in selecting the best of several possible flowsheets, or evaluating the effect of various assumptions.

EVERYONE who has to calculate cyanide circuits, using either thickeners, filters or both, realizes the headaches involved in solving a set of simultaneous equations. When you calculate a large number of these, based on experimental data, trying to arrive at the best flowsheet, you can spend days at the task.

T. B. COUNSELMAN, Member AIME, is Manager, FluoSolids Division, The Dorr Co., Stamford, Conn. AIME New York Meeting, Feb. 1950.

TP 2783 B. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received Aug. 23, 1949.

Herewith is presented a short method, equally accurate with the standard simultaneous equation method, and much quicker.

Here for instance is the standard simultaneous equation method, for 100 tons per day of \$10 ore, with the conditions assumed as below (fig. 1).

### Conditions Assumed

- a. 100 tons of ore per day crushed in cyanide solution.
- b. Discharge from all thickeners with 50 pct moisture.
- c. \$10 value dissolved per ton of ore.
- d. 50 pct in mill and 50 pct in agitators.

- e. 400 tons of solution from thickener V precipitated to \$0.02.
- f. Agitation with a dilution of 2 of solution to 1 of solids.
- g. Let V, W, X, Y and Z represent value in dollars per ton of solution discharged from the respective thickeners.

### Equating Out of and Into Each Thickener

1.  $100V \text{ plus } 400V = 500W$   
plus  $(0.50 \times \$10 \times 100)$
2.  $100W \text{ plus } 600W = 500X \text{ plus } 100W$   
plus  $(0.50 \times \$10 \times 100) \text{ plus } 100V$
3.  $100X \text{ plus } 500X = 100W \text{ plus } 500Y$
4.  $100Y \text{ plus } 500Y = 100Z \text{ plus } 100X$   
plus  $(400 \times 0.02)$
5.  $100Z \text{ plus } 100Z = 100Y$   
plus 100 tons of water (value \$0.00)

### Simplifying:

1.  $V = W \text{ plus } 1.00$
2.  $W = X \text{ plus } 1.20$
3.  $X = Y \text{ plus } 0.24$
4.  $Y = 0.2Z \text{ plus } 0.064$
5.  $2Z = Y$





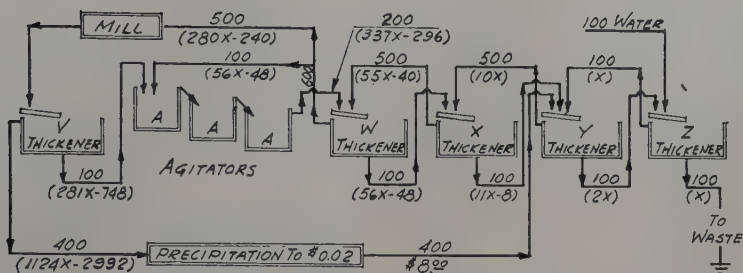
Checking this:

Total value dissolved in plant, 100	
tons at \$10 (50 pct in mill)	
(50 pct in agitators) = \$1000.00	
400 tons precipitated:	
(1124x — 2992) — 400 times \$0.02	
= 1004.94 — 8.00	996.944
Loss or residue (x)	3.556
Total	\$1,000.500

back to thickener Z, (displacing an equal amount of wash water previously added to that thickener), and the stronger filtrate back to thickener Y, the tonnages would divide as follows (fig. 3):

The (x) values balance out, by the method described above, as shown for the last part of the flowsheet.

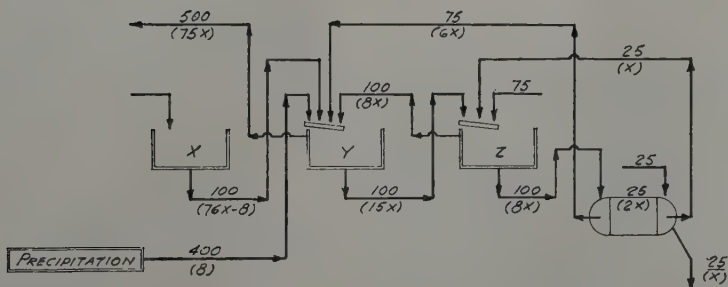
It can, of course, be assumed that there will be a volumetric displacement of, say, 75 pct of the solution in the filter cake, followed by dilution of



FIGURES ABOVE FLOW LINES REPRESENT SOLUTION TONNAGES

FIGURES ( ) BELOW FLOW LINES ARE TOTAL VALUES PER 24 HRS.

Fig. 2—Shortcut method.



FIGURES ABOVE FLOW LINES REPRESENT SOLUTION TONNAGES

Fig. 3—Shortcut method with filter.

The above flowsheet can be made and the loss calculated in a few minutes, as against 20 or 30 min for simultaneous equations, and gives satisfactorily accurate results.

The same simplified method can be used to include filtration, at any desired point in the flowsheet (fig. 3). For instance in the example given above, if the final thickener underflow were filtered to a final cake moisture of, say, 25 pct, you would assume that the filtration took place in two steps. First the straight filtration to final moisture, and second the addition of an amount of wash water equal to the moisture left in the cake, with a second filtration to final cake moisture. Conventionalized for calculations, and taking the wash filtrate

the remaining solution with water. The same principle of computation would apply to whatever assumption was selected.

This same method of calculation may be used for calculating countercurrent decantation circuits in the chemical industry, for example causticizing flowsheets, the manufacture of alum from bauxite, or the washing, by countercurrent methods, of any solid, free from solutions.

#### Acknowledgment

Thanks are due to Mr. Dycus, of Golden Cycle Corp., and Nathaniel Herz, of Homestake Mining Co., for helping to work out the application of this method of calculation to cyaniding.

# Effect of Conditioning on Flotation of Chalcocite

Chalcocite flotation is affected by agitation intensity during conditioning. Sodium sulphide in minute quantity may activate or depress, depending on conditions. Oxygen is a depressant while oxygen and sulphide neutralize each other by reaction at the mineral surface. Diffusion across a stagnant film at the surface governs reaction rates, while film thickness is reduced upon increase of agitation intensity.

by S. B. Tuwiner and S. Korman

## Introduction

THE purpose of this study is to investigate the effect of agitation upon the flotation of chalcocite. It is believed that the concentration of collector which is required to produce a water-repellent surface upon the mineral is determined among other things by the rate of collector diffusion through an impeding film of more or less stagnant fluid. Its thickness depends upon the intensity of agitation.

S. B. TUWINER, Member AIME, is Assistant Director, and S. KORMAN, Member AIME, is Metallurgical Chemist, Research and Development Department, Phelps Dodge Corp., Laurel Hill, N. Y.

AIME Columbus Meeting, September 1949.

TP 2771 B. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received May 4, 1949, revision received Oct. 10, 1949.

In the first part of this study a method is developed for determining the thickness of the fluid film upon a polished surface of synthetic chalcocite. The film thickness is then determined as precisely as possible under three readily reproducible conditions of agitation intensity. These determinations serve as a quantitative evaluation of agitation.

In Part 2 it is demonstrated that in solutions of lime at pH 10.5 and in the absence of air or dissolved oxygen the concentration of collector required to develop water-repellency upon the surface in any given time is a function of the fluid film thickness, and that after about 10 min the relation becomes linear. This points to the possibility that the collector ions may be involved in a diffusional process through the fluid film.

In Part 3 it is shown that where air is present more collector is required to produce water-repellency than for the case in which air is excluded. Next, it is shown that for the latter case the most minute amount of sodium sulphide suffices to preclude development of water-repellency. Both oxygen and sulphide are in a sense, therefore, depres-

sants for chalcocite in a lime solution at pH 10.5. When oxygen and sulphide are present simultaneously there is a region of minimum critical collector concentration for water-repellency which is very markedly lower than the concentration where oxygen alone or where sulphide alone is present. Moreover, this minimum appears to be identical with the collector concentration which is required where sulphide and oxygen are both excluded. This would appear to indicate that in certain ranges of concentration of oxygen and sulphide, each cancels the effect of the other. The range of concentration of solution for this effect in solutions saturated with oxygen is a function of the agitation intensity. Thus we are led to believe that a chemical reaction involving sulphide and oxygen occurs at the surface of the chalcocite.

Part 4 of this study is a further development of this idea and some of its consequences from an electrochemical standpoint, while in Part 5 the disappearance of sulphide from pulps of several minerals is noted with an indication that both dissolved oxygen and the mineral are necessary, the reaction being catalytic or pseudocatalytic.

## Part 1. Determination of Fluid Film Thickness

**The Fluid Film Concept:** The hindrance to the diffusion of collector to the surface of the chalcocite is a barrier of stagnant fluid in the vicinity of the surface. It is the same barrier that limits the rate of transfer of heat, the rate of solution of dissolving crystals, or any of the other rates pertaining to chemical engineering unit operations in which the film resistance to diffusion plays a principal role.<sup>1</sup> Fluid films are present upon the surface to an extent which varies with the geometry and intensity of agitation. They result from the fact that in the region within the fluid near a surface, motion is entirely parallel to it. From the standpoint of transfer to or away from the interface the effect is as though the fluid were stagnant. Outside of the film, movement of the fluid may be streamline or turbu-



lent. Transfer is here principally by convection and usually very much more rapid than across the fluid film, in which diffusional processes are controlling. If a soluble substance such as a flotation collector is present in low concentration in a flotation pulp, and if this substance is consumed continually at the interface, then it is conceivable that its concentration near this interface may approach zero. The solute will continuously diffuse through the film

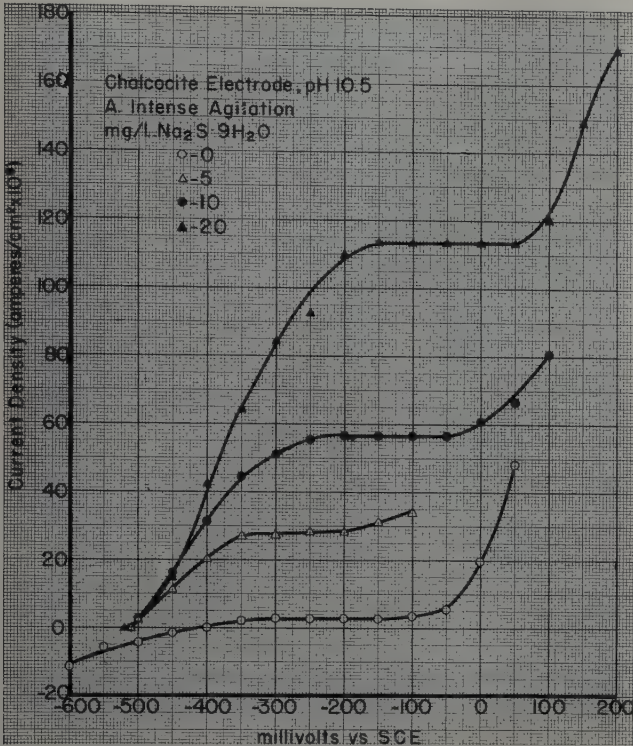


Fig. 1—Sulphide diffusion in air-free solution.

from the bulk of the solution, in which rapid transfer by convection predominates. The rate of its transfer through the film will be proportional to the concentration difference, and inversely, to the length of path traversed, the effective thickness of the fluid film.

Under conditions of quiescence this film will be comparatively thick and the diffusion relatively slow. On the other hand, in an intensely agitated pulp the stagnant film must be much thinner, and the diffusional rate consequently greater. In numerous chemical engineering studies these very elementary and basic considerations have resulted in an effective correlation of such physical factors as velocity, viscosity, and density with the rates of fluid flow, heat transfer, and material transfer in a rational and consistent manner. The same kind of analysis can be made as fruitful for the design of processes and equipment in the art of flotation as it has been for boilers, evaporators, condensers, distillation columns, and other equipment.

**Method of Study:** From this investigation it is shown that the collector concentration which is needed to give a water-repellent surface in any specified time is very greatly affected by the intensity of agitation. From this it is inferred that surface phenomena involving collector as well as

other soluble substances which affect the flotation process are hindered by a diffusional resistance. It is assumed that there exists upon the solid surface a layer of fluid through which material must pass by a relatively slow process of diffusion, moving always from a region of higher to one of lower concentration either to or away from the interface. This film is never observed directly but is manifest only through the properties which it possesses, its effective thickness being determined in any case by a study of some rate process of material transfer.

In this study the effect of agitation intensity upon the conditioning of chalcocite is observed in an alkaline solution containing known amounts of potassium n-butyl xanthate and of Reagent 208. A specimen of synthetic chalcocite is mounted in lucite and polished in the usual way. Three degrees of agitation intensity are selected for the conditioning of this specimen. For the first degree, designated as "intense agitation," the specimen is held face down against the stream produced in a Waring Blender. For the second, termed "moderate agitation," the chalcocite surface is left face up in a glass

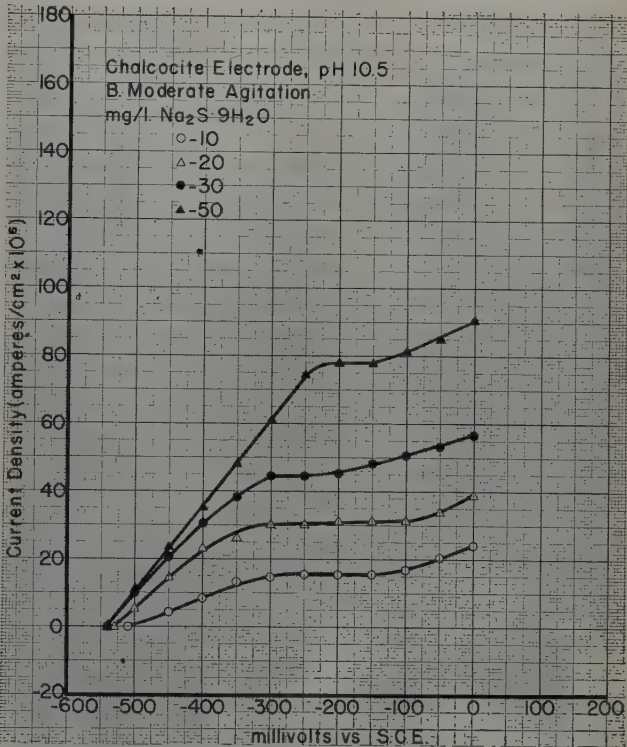


Fig. 2—Sulphide diffusion in air-free solution.

cell such as is used for contact angle measurement, agitation being produced by means of a stream of gas or air passing upward through the solution from a porous stone diffuser. The third degree of agitation is obtained in the same cell but under the "stagnant" condition which has become customary for contact angle study.

The terms "intense," "moderate," and "stagnant" have only qualitative significance; a quantitative measure of intensity is preferable. For this the effective fluid film thickness obtained by measuring the rate at which sulphide ions from solutions of known concentration are oxidized at a chalcocite



electrode by means of positive electric current. In these experiments, air is excluded. Under each condition of agitation and for several concentrations of sodium sulphide, increments of positive current were passed through the mineral, the cathode being a platinum wire electrode. The electric potential was noted for the chalcocite anode referred to a saturated calomel electrode.

**Experimental Details: Measurement of Sulphide Diffusion Currents:** For measurement of the diffusion current during intense agitation in the Waring Blendor the polished mineral specimen is mounted face down and held in position by means of an insulated  $\frac{1}{8}$  in. diam brass rod which is screwed into the back of the lucite mounting to make electrical contact with the chalcocite. The assembly is sealed into the cover of the Blendor. The cover is also provided with a sealed entry tube for pure hydrogen gas and a sealed platinum cathode wire. The saturated calomel electrode is clamped in fixed position through a loose fitting hole in the cover, which also provides for exit of gas.

The specimen, after preliminary polishing with metallographic paper, is polished with levigated

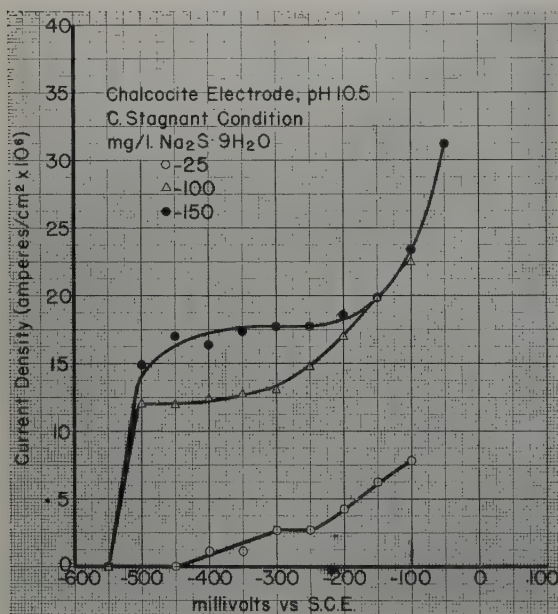


Fig. 3—Sulphide diffusion in air-free solution.

alumina and rinsed thoroughly with demineralized water. It is then placed in lime water, pH 10.5, containing 0.004 mol sodium sulphide, where it is held for a period of 1 hr. The Blendor is filled with 800 cc of lime water at pH 10.5 containing a specified concentration of sodium sulphide. Commercial tank hydrogen passed through copper turnings at 600°C is flushed through the solution for 10 min. The chalcocite-cover assembly is installed upon the Blendor and secured in position with rubber tape. Hydrogen gas is allowed to flow continuously at a rate of 1 liter per min during the course of the current potential measurements as described below.

For moderate agitation or stagnant condition, the chalcocite specimen is polished and conditioned in sodium sulphide solution as described above. Electrical contact is obtained by means of an insulated threaded shaft at right angles through the lucite to

the side of the specimen. The chalcocite specimen is then placed face up in a solution of 300 cc of lime water at pH 10.5 containing specified concentration of sodium sulphide at the bottom of a 400 cc capacity, rectangular, glass cell similar to that commonly used in contact angle measurements. The same platinum cathode and saturated calomel electrode are installed and held rigidly by clamps. A cylindrical stone diffuser  $1 \times \frac{7}{8}$  in. diam for hydrogen is

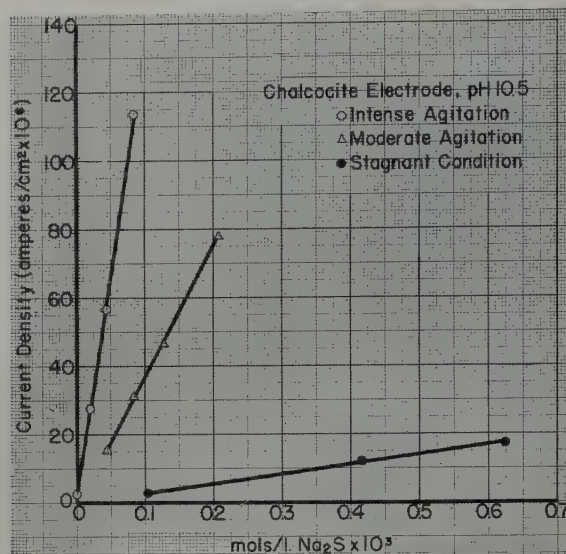


Fig. 4—Sulphide diffusion in air-free solution.

introduced to the bottom of the cell. A flow rate of 1 liter per min, as measured with a laboratory flowmeter, produces a reproducible degree of agitation and at the same time removes all oxygen from the solution and prevents its further access. For the stagnant condition, following preliminary flushing of the solution with hydrogen for 15 min, the diffuser is raised until it is completely out of the solution but is maintained at the top of the cell above the solution during the course of the measurements. Great care must be taken to insure that the diffuser for this type of conditioning does not disturb the surface of the solution, since it was found that the slightest turbulence at the surface is sufficient to provide a very appreciable increment in the diffusion currents obtained.

**Current Potential Measurements:** Current is supplied from a dry cell battery source across which is a 180 ohm slide wire potentiometer. That portion of the current which is supplied to the chalcocite surface is controlled by the position of the tap. The circuit is completed through a Weston microammeter with the positive lead to the chalcocite and the negative lead to the platinum cathode. Potential measurements are obtained by means of a Beckman industrial type potentiometer in circuit with the chalcocite and the saturated calomel electrode.

Current values are obtained by adjustment of the position of the slide wire tap and read on the microammeter. Corresponding potentials are read directly on the Beckman potentiometer.

**Sulphide Diffusion and Film Thickness:** A number of curves expressing the current-potential relations are shown in figs. 1, 2, and 3. Significantly,



there is a potential in all cases corresponding with the plateau region of the curves in which primary reaction involving the oxidation of sulphide ions appears to have reached its upper limit, the amount of electric current in each case being determined by the rate at which the sulphide ions are able to diffuse through the fluid film resistance. As is shown in fig. 4, these so-called diffusion currents are precisely proportional to the sodium sulphide concentration, indicating that the driving force of this

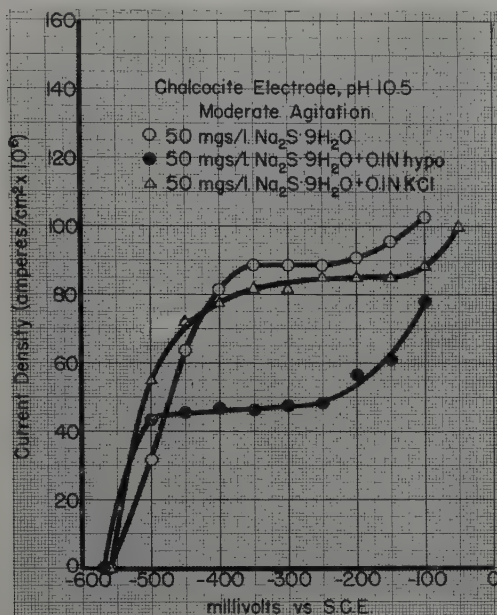
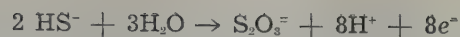


Fig. 5—Sulphide diffusion in air-free solution.

material transfer process is the concentration gradient across the fluid film which has its highest value at the potential corresponding with the plateau regions of the current-potential curves at which presumably the concentration of sulphide at the mineral interface has become zero. To compute the effective fluid film thickness in each of the three conditions of agitation the following formula<sup>2</sup> is used:

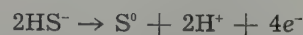
$$l = \frac{D}{i/A} n F C$$

Where  $l$  is the mean fluid film thickness in cm,  $i/A$  is the current density in amperes per cm<sup>2</sup>.  $D$  is the diffusion coefficient for sulphide ions in cm<sup>2</sup> per sec, and  $n$  is the number of electrons transferred in the oxidation reaction from each sulphide ion.  $F$  is the faraday, equal to 96,500 amp-sec, and  $C$  is the concentration of sulphide in the solution in gram molecules per cm<sup>3</sup>. All of these quantities are known from our measurements, except the diffusion coefficient for sulphide or hydrosulphide,  $D$ . This is not given in the literature, although  $1.4 \times 10^{-5}$  cm<sup>2</sup> per sec has been given as the approximate diffusion coefficient for H<sub>2</sub>S at 15°C.<sup>8</sup> Assuming this value for the diffusion coefficient the following values of the mean fluid film thicknesses are computed: 41, 147, 1940 microns for the three conditions of agitation studied. In calculating these it is assumed that  $n$  in the equation above is equal to 4, corresponding to the chemical reaction at the electrode surface of



The following facts support this assumption:

As shown in fig. 5 the addition of potassium chloride equivalent to 0.1 mol does not appreciably alter the diffusion current in these measurements. The effect of KCl is to reduce the voltage (IR) drop between electrodes. This would indicate that the currents actually are composed entirely of diffusion currents and that the so-called migration current is not included. Adding to these solutions sodium thiosulphate in the amount of 0.1 normal has the effect of reducing these diffusion currents by approximately one-half as shown in fig. 5. This effect of thiosulphate is the product of the sulphide oxidation and that by an excess of thiosulphate the reaction is altered from that shown above to the following:



The following considerations are taken into account in determining the most likely electrode process for the oxidation of sulphide: In the first place, the fact that linear relations are obtained between maximum diffusion current density and concentration of sulphide, shown in fig. 4, is a strong indication that the electrode reaction leads to a single sulphide oxidation product. Furthermore, it is unlikely that the diffusion currents measure the formation of oxidation products such as sulphite, pentathionate, and tetrathionate, all of which react directly with sulphide ion. Again, in view of the fact that thiosulphate ion does not produce a diffusion current in the range of potential studied, indicates that more extensive oxidation to di- and trithionate ion does not occur. Thus, oxidation to thiosulphate represents the greatest extent of oxidation possible as a single product which is not present in excess. When thiosulphate is present in excess, the diffusion current is reduced to half value, indicating a shift to a process forming a still lower oxidation product, namely, sulphur.

A flow of positive current to the mineral surface is used to oxidize dissolved sodium sulphide. This current (assuming no side reactions) would be proportional to the rate of diffusion, and discharge in accordance with Faraday's law. Rate of diffusion would in turn depend upon the potential imposed upon the mineral for it is this potential which determines the concentration of sulphide in the solution at the interface. At a potential at which this concentration becomes zero any further change can have but slight effect inasmuch as it does not further augment the concentration gradient of sulphide across the stagnant film. It is the concentration drop, the driving force, which is responsible for the diffusion, and thereby for the current.

**Meaning of Diffusion Currents:** The phenomenon illustrated in figs. 1, 2, and 3 is a result of an extreme state of concentration polarization which results from the depletion of sulphide at the electrode surface by the electrode reaction. The limit to the diffusion current which is marked by the plateau region of the graph signifies that sulphide ions are oxidized as rapidly as they reach the chalcocite electrode surface and their concentration at this surface remains constant at a value which is negligi-

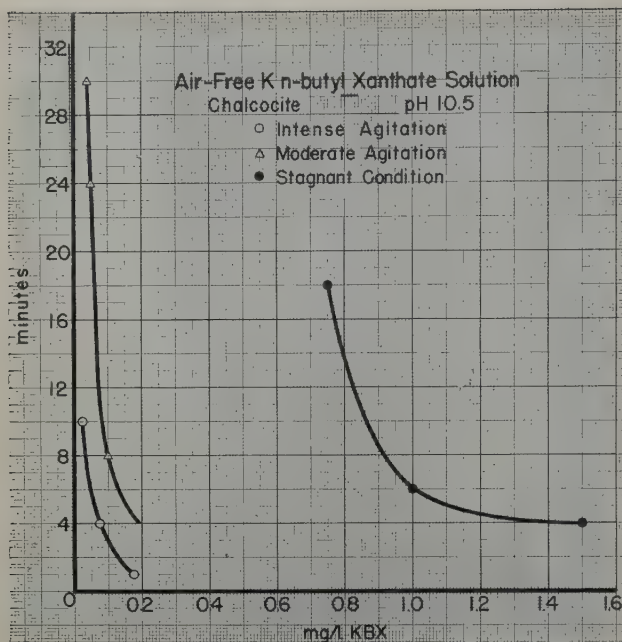


Fig. 6—Critical collector concentration vs. conditioning time.

bly small compared to the concentration in the body of the solution.

Under these conditions the current resulting from the electrode reaction is independent within certain limits of the applied e.m.f. and is governed solely by the rate of supply of the oxidizable substance to the electrode surface from the surrounding solution. In the most general case, reducible or oxidizable ions are supplied to the depleted region at the electrode surface by two forces: (1) a diffusive force proportional to the concentration gradient at the electrode surface, and (2) an electrical force proportional to the electrical potential difference between the surface and the solution. Reducible or oxidizable ions are therefore supplied to the electrode surface partly by diffusion and partly by electrical migration so that the limiting current can be regarded as the sum of a diffusion current and a migration current.<sup>4</sup>

It is not necessary to consider a migration current under these experimental conditions. Hence, the concept may for the present be simplified by regarding the currents measured, as a result solely of the tendency of sulphide ions to move from a region of relatively high to one of low concentration, the rate of this movement being proportional to the concentration gradients. These gradients are then the driving forces.

## Part 2. Minimum Collector Concentration vs. Agitation Intensity

Quantitative knowledge of the agitation intensities are thus available for the three conditions of agitation to be employed in this study. Next, with air absent and using a stream of purified hydrogen gas the minimum concentration of collector is determined which suffices to cause adherence of a captive bubble upon the mineral surface using the standard technique.

For intense agitation in the Waring Blendor, the chalcocite specimen is polished and tested in demineralized water to zero angle in the usual manner. It is then overturned and screwed to a ½ in. diam bakelite rod, without removing the specimen from the solution.

The specimen is then transferred to the solution in the Blendor, clamped into the cover assembly previously described, and conditioned for a specified time in 800 cc of collector solution at pH 10.5 with the agitator turned on, and hydrogen flowing at 1 liter per min.

At the end of the conditioning time, the specimen is taken off the rod and immediately placed in the cell used for mild agitation with H<sub>2</sub> gas flowing continuously through the stone diffuser. The solution in this cell has a composition identical with the one in the Blendor. The transfer of the specimen requires less than 30 sec. Between the time when Blendor conditioning ends, and the first bubble contact measurement is made, 1 min has elapsed, allowing 30 sec for bubble contact before measurement.

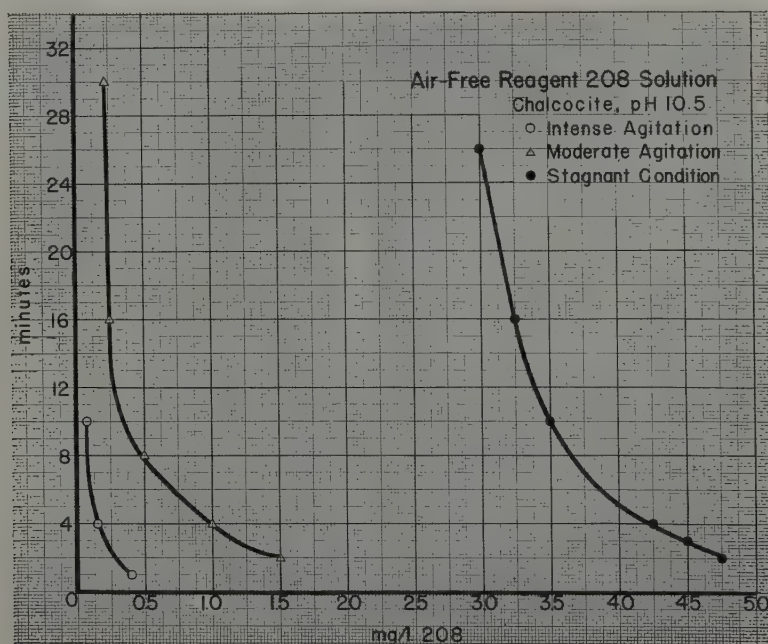
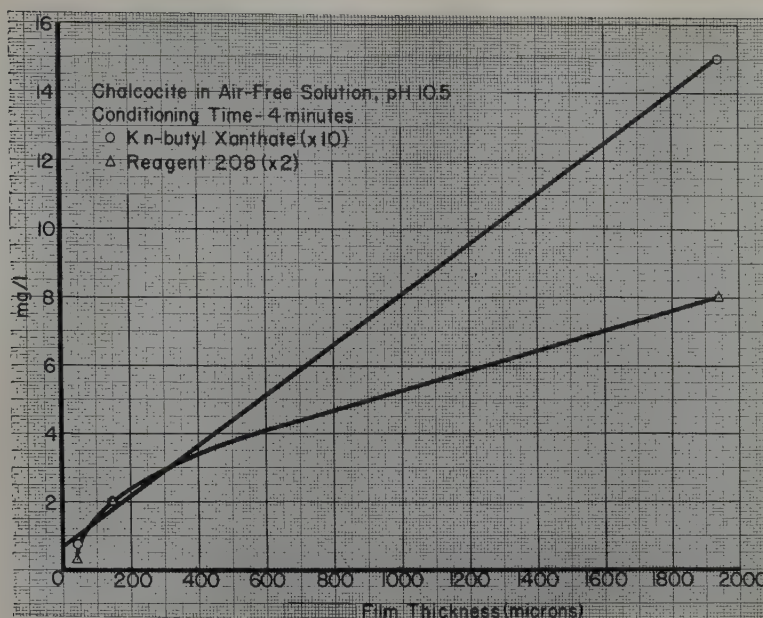


Fig. 7—Critical collector concentration vs. conditioning time.



**Fig. 8—Critical collector concentration vs. film thickness.**



Measurements on three other areas of the specimen may be made in 2 to 3 min.

Minimum measurable angle with the equipment used was  $25^\circ$ . The curves for this data represent the locus of all points for which any lower concentration produced no measurable angle.

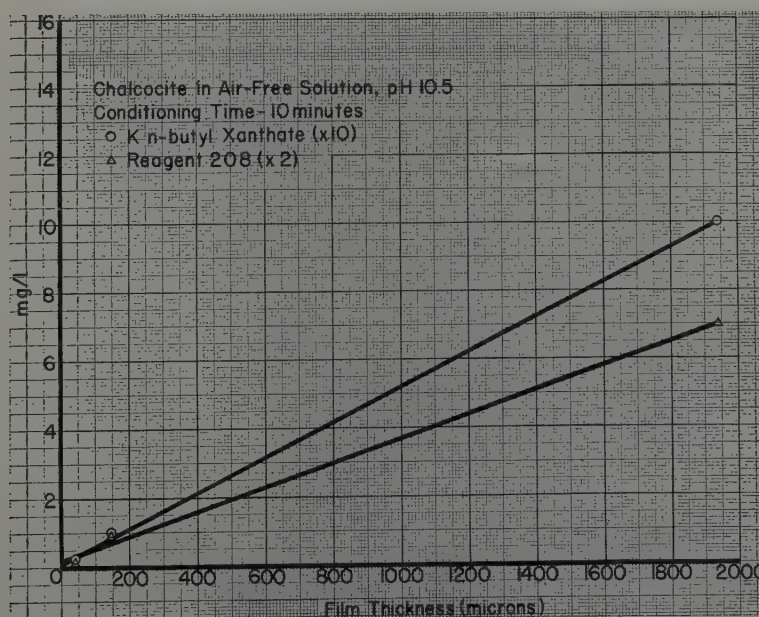
For mild agitation, the specimen, after test for cleanliness as described above with demineralized water, is placed in the cell containing 300 cc lime water at pH 10.5 containing specified collector concentration with  $H_2$  gas through the diffuser and solution at 1 liter per min. This instant represents zero time. Angles are tested minutewise for 6 min, and then every 2 min for a total of 30 min, with  $H_2$  gas flowing continuously at all times to maintain the agitation and oxygen-free solution. The time at which a measurable angle first appears is plotted as the point in time at which the tested collector concentration is a critical minimum value for this degree of agitation.

For stagnant conditioning, this procedure is repeated, except that, after a preliminary purging of the test solution with  $H_2$  gas, the diffuser is held directly above and out of the solution as the specimen is lowered into it.

It is found that the minimum concentration with air absent is a function of the conditioning time as well as the agitation intensity, the relationships being shown in fig. 6 for potassium n-butyl xanthate and in fig. 7 for Reagent 208. Figs. 8 and 9 show how, for a conditioning time of 4 and 10 min, collector concentration is related to the effective fluid film thickness. The relations become linear in the latter indicating degeneracy to a relatively simple process.

The conclusion appears to be inescapable that the process of conditioning involves a heterogeneous surface reaction in which the diffusion of collector across a fluid film barrier is the rate-determining process.

**Fig. 9—Critical collector concentration vs. film thickness.**



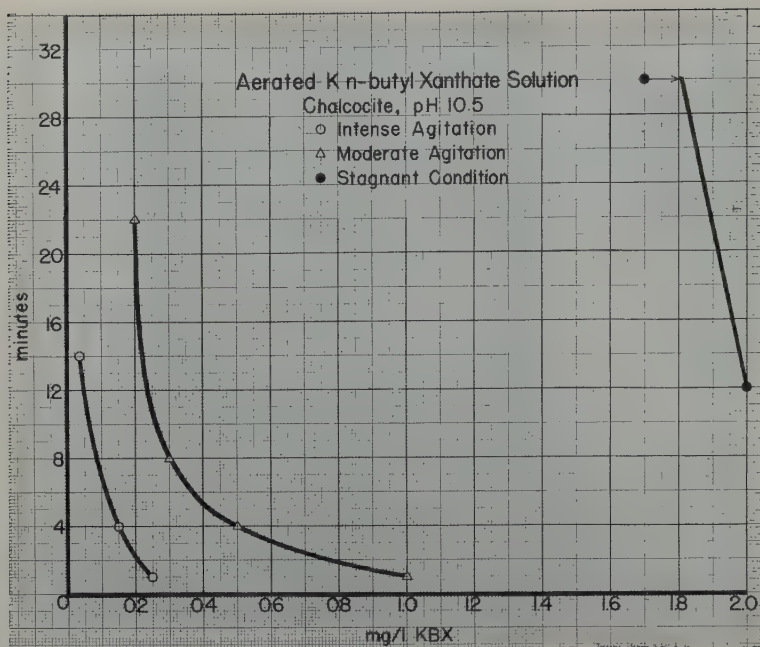


Fig. 10—Critical collector concentration vs. conditioning time.

### Part 3. Dissolved Oxygen and Sodium Sulphide

**Effect of Oxygen:** When the tests described in Part 2 are repeated with air used in place of hydrogen, the figs. 10 and 11 are obtained in place of figs. 6 and 7. The relation of minimum collector concentration and fluid film thickness obtained for 10 min of conditioning, in fig. 9, is changed to that shown in fig. 12.

The comparison shows that a higher collector concentration is required when oxygen is present during conditioning with collector. In other words, oxygen acts as a depressant in this system.

**Effect of Sulphide:** The most minute amount of dissolved sodium sulphide will effectively depress chalcocite if air is excluded. This may be shown by grinding 30 g of chalcocite in a ball mill with 0.5 g CaO and 200 ml of water, transferring to a 4 liter Fagergren flotation cell, introducing purified

nitrogen for 5 min and then adding 150 mg of potassium n-butyl xanthate to produce a strong float, and then 1 ml of a 1.6 pct solution of  $\text{Na}_2\text{S} \cdot 9\text{H}_2\text{O}$  which depressed the float. It was found that a sample of the pulp solution contained an amount of sulphide which is not detectable with lead nitrate reagent. The same result was obtained in another test using 300 mg of Reagent 208.

**Effect of Simultaneous Presence of Sulphide and Oxygen:** Figs. 13 and 14 show how sodium sulphide affects the critical concentration of collector when air is present. The method of measurement is identical with that of Part 2. It will be observed that small additions of sodium sulphide reduce the collector requirement while increasing amounts beyond an optimum result in aggravating the requirement. As figs. 13 and 14 illustrate, the optimum sulphide concentration (depending upon the degree

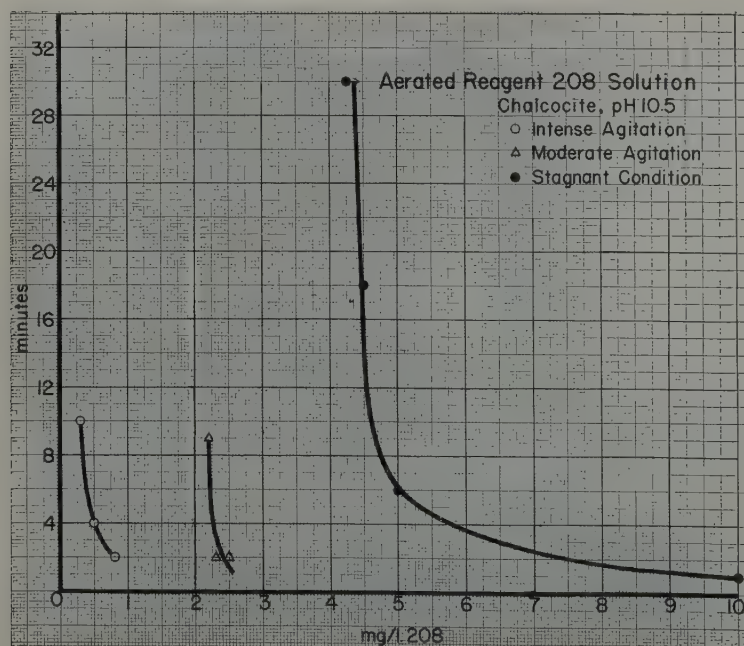


Fig. 11—Critical collector concentration vs. conditioning time.



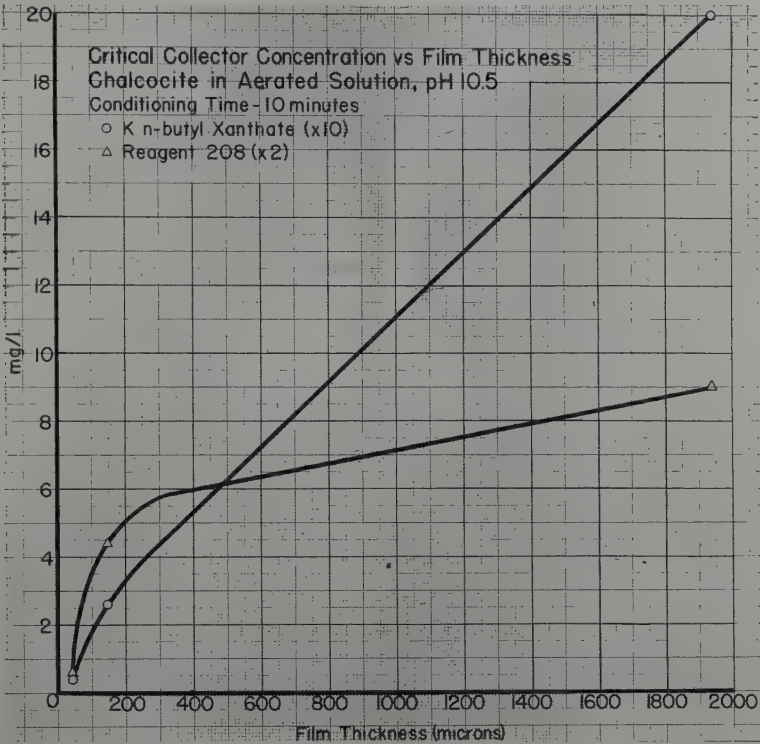
of agitation) corresponds to a minimum in the initial collector requirement. Comparison in table I shows that these minimum values correspond almost exactly with the minimum values obtained in Part 1, figs. 6 and 7, where sulphide and oxygen are both absent.

**Table I. Minimum Critical Collector Concentration**

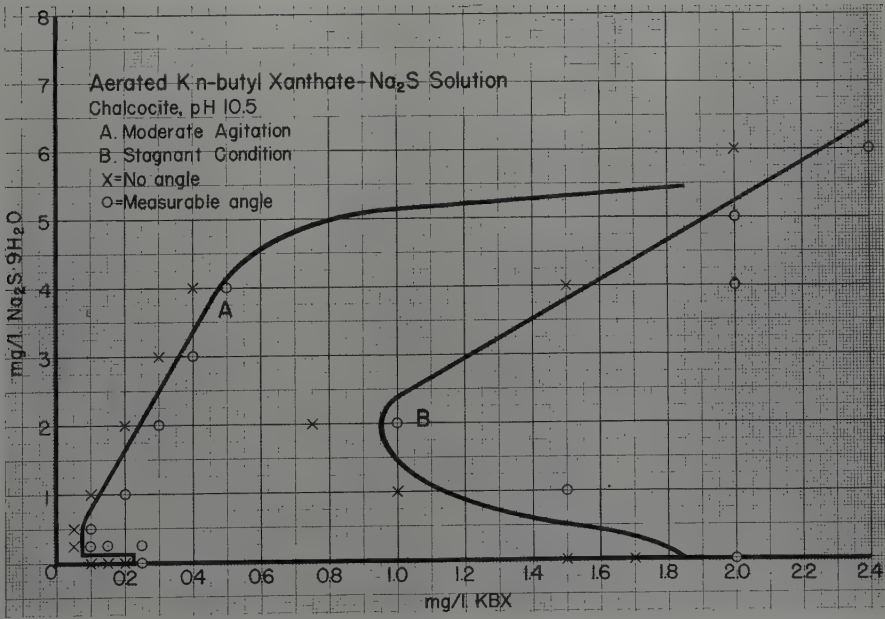
Collector	Agitation	Sulphide and Air Present, Mg per L	Sulphide and Air Absent, Mg per L
K n-butyl xanthate	Stagnant	0.95	0.86
Reagent 208	Stagnant	3.25	3.5
K n-bX	Moderate	0.087	0.08
Reagent 208	Moderate	0.25	0.22

In the region of low concentrations the sulphide appears to act as an activator while at higher concentrations it becomes a depressant. Not only must it be recognized that one reagent may fulfill both roles, depending upon its concentration, but also the question of whether it serves as one or the other depends upon the intensity of agitation. No doubt the system here involving at least three substances in a diffusional process is very complex. As demonstrated later, sulphide serves to react with, and presumably destroys, dissolved oxygen at the interface. It also tends to render the chalcocite extremely reactive electrochemically. The first property may relate to its role as an activator, the latter serve to explain its depressing action. The explanation of

**Fig. 12—Critical collector concentration vs. film thickness.**



**Fig. 13—Effect of sulphide on critical collector concentration.**



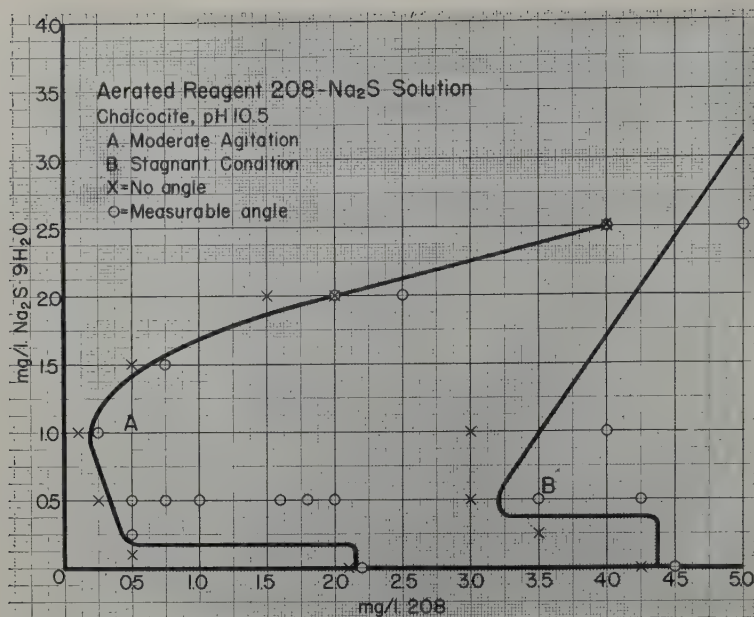


Fig. 14—Effect of sulphide on critical collector concentration.

depression as a surface displacement of all other ions by sulphide is valid at higher concentrations, as a later group of experiments will demonstrate, using sulphide concentrations of higher order.

In connection with the action of sodium sulphide on chalcocite, it is noted that there is a marked increase in reactivity of chalcocite specimens which have been polished and immersed in 0.004 mol sodium sulphide solution pH 10.5 for 1 hr. This is demonstrated by a great increase in sensitivity to the introduction of oxygen to a lime solution containing the specimen and a calomel electrode. The potential in the absence of oxygen is approximately the same as that of a freshly ground surface during and immediately after abrasion on a carborundum wheel (about  $-400$  mv). On the other hand, a chalcocite surface which has been exposed for some length of time to the air, becomes quite insensitive; its potential is about  $-80$  mv. The sensitized chalcocite is also capable of displaying great reactivity to sulphide; this is demonstrated by the current-potential curves for sulphide electrochemical oxidation given above, which are curves for a chalcocite specimen conditioned in sulphide for 1 hr.

It is important to recognize that amounts of sodium sulphide much smaller than determinable by ordinary analytical methods nevertheless have a profound effect upon flotation of chalcocite. This is of interest in view of the observations that sulphide ion is reported to be present in some pulps.<sup>5</sup>

#### Part 4. Electrochemistry of Sulphide and Oxygen

For purposes of analysis, it may be assumed that oxidation of sulphide at the mineral-liquid interface is electrochemical. If it is assumed to be electrochemical it does not matter whether or not the anode and cathode reactions occur separately upon discrete areas. It will help to postulate that the function of oxygen may be assumed by a positive electric current, that of sulphide by a negative electric current. Thus, the effects produced by the two reactants may be studied individually and quantitatively by means of electrical measurements.

If the sulphide solution is saturated with air, the

cathodic reaction is not limited by diffusion inasmuch as oxygen is relatively plentiful. Nevertheless, in concentrations as high as 52.8 mg per liter of  $\text{Na}_2\text{S} \cdot 9\text{H}_2\text{O}$  the potential in volts of a chalcocite surface under "moderate" agitation is almost the same in an air-saturated solution as it is in one purged with nitrogen. This means that in this range it is the sulphide which exerts the dominating influence in determining the surface properties.

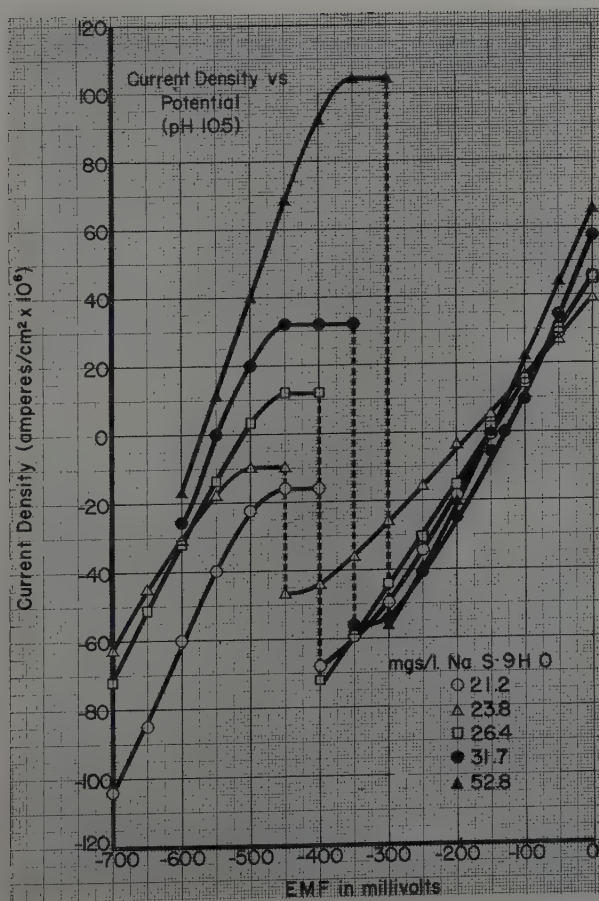
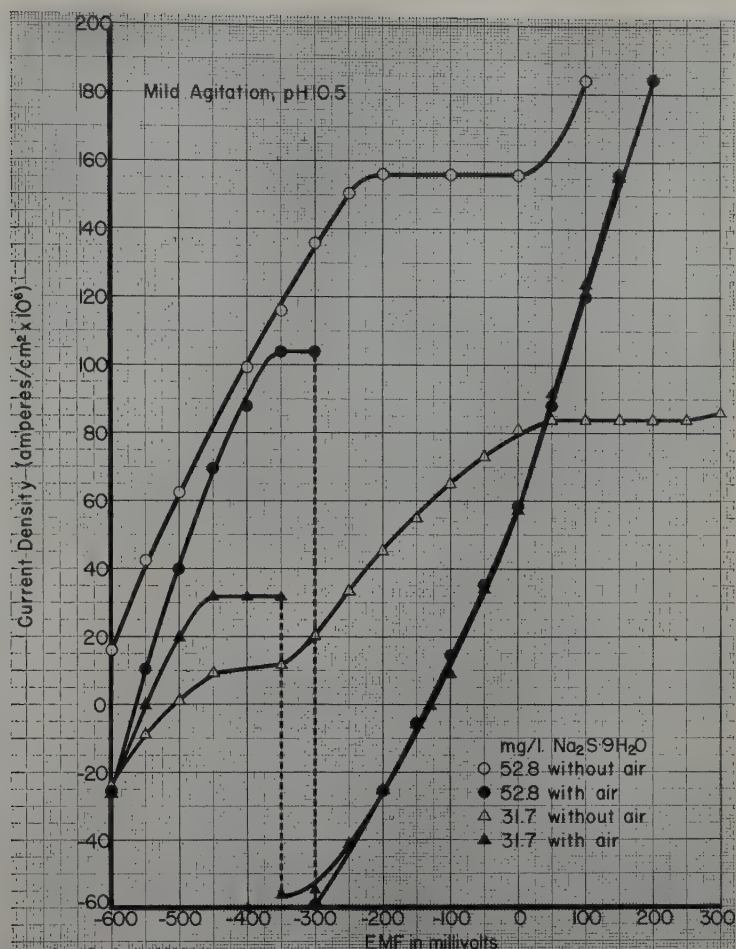
Imposing a positive current upon such a surface moves the potential in the positive direction in the air-saturated solution containing sulphide. As shown in fig. 15a for the same agitation condition, namely, moderate, mean fluid film thickness 147 microns, it will be seen that at first the current for any given high negative potential rises as the potential is increased whether the oxygen is present or not. Moving along in the direction of increased potential, the differences between the two curves begin to augment. Finally, a point is reached at which the cathodic activity builds up to a point at which the current density to maintain the potential shifts suddenly from a positive to a strongly negative value. Whereas heretofore it was required to supply positive current to the chalcocite it is now necessary to furnish negative current. The curve then continues as shown in fig. 15, crossing the  $x$ -axis again at  $-0.155$  v in the case of 52.8 mg per liter  $\text{Na}_2\text{S} \cdot 9\text{H}_2\text{O}$ .

In the absence of an imposed current the mineral surface assumes the potential for zero current, that is, the intersection of the graph with the  $x$ -axis. However, it is evident that under these conditions of study, two such potentials are possible. On the one hand, a stable potential for zero current represented by the left-hand intersection in the graph, corresponds to the presence of unoxidized sulphide ion at the surface. On the other hand, the right-hand intersection is interpreted as a stable potential corresponding to excess oxygen at the surface. A transition from excess sulphide to excess oxygen at the surface, in this high concentration of sodium sulphide, requires the addition of energy to the system. In this instance, this is accomplished by im-



Fig. 15a (right)—Chalcocite in sulphide solution.

Fig. 15b (below)—Chalcocite in sulphide-air solution.



posing a positive current, which has the effect of depleting the sulphide ion concentration at the surface. The analogous transition from excess oxygen to excess sulphide at the surface is accomplished by depleting the oxygen concentration at the surface; namely, by purging the solution with nitrogen, or, by allowing the surface to be conditioned for a considerable time in a stronger sodium sulphide solution.

A characteristic curve is obtained for each concentration of sodium sulphide used. Those for the lower concentrations reveal a transition from excess sulphide to excess oxygen lying wholly below the  $x$ -axis. This is taken to mean that the sulphide depletion required to effect the transition is brought about by means of the added oxygen alone. From the assumptions made herein, this is accomplished by virtue of the cathodic activity of oxygen when it reaches the mineral surface. On this basis, the range of potentials covered by the transitions comprise a region in which oxygen exhibits strong cathodic activity, presumably to form hydroxyl ions. In this system, the extent to which this activity occurs is dictated by the requirement of sulphide depletion at the surface; when the sulphide has reduced very nearly to zero concentration at the surface, the oxygen activity would then have reached a maximum rate. This view is supported by comparison of curves in fig. 15a. The transitions occur at potentials somewhat lower than those corresponding to the upper limit of the sulphide diffusion currents under hydrogen. This suggests that the

condition for a transition from excess sulphide to excess oxygen is a concentration of sulphide ion approaching zero at the interface. Admittedly, a more satisfactory substantiation of this inference would be preferred.

The sulphide concentration corresponding to the maximum which oxygen alone can deplete spontaneously in this system of agitation intensity lies, as shown in fig. 15, between 23.8 and 26.4 mg per liter of  $\text{Na}_2\text{S} \cdot 9\text{H}_2\text{O}$ . Fig. 16 shows contact angles measured upon a specimen of chalcocite conditioned prior to each test in stronger sodium sulphide and then transferred to the solution containing the specified concentration of sodium sulphide with an amount of xanthate or thiophosphate collector as indicated.

The contact angles of fig. 16 are those measured 1 min after the specimen enters the solution. This is the shortest time experimentally convenient. Over longer periods of time the contact angle grows less and ultimately disappears.

A critical sulphide concentration of 24.5 to 26.5 mg per liter from fig. 16 compares well with 23.8 to 26.4 mg per liter from fig. 14. This concentration is one below which a contact angle is obtained; it does not necessarily persist.

All of the above refers to a system saturated with air and under "moderate" agitation. On the other hand, if nitrogen or hydrogen is used to purge the solution, the merest trace of sulphide precludes a contact angle.

**Experimental Procedure: Preparation of Specimen:** A piece of synthetic chalcocite, made as previously described, was mounted in lucite and polished with levigated alumina in the conventional manner. The exposed polished area was 1.25 sq cm. It was then treated with 0.004 molar sodium sulphide in lime solution at pH 10.5 for 1 hr.

The specimen was then transferred with care to remove all liquid, but without washing, to a glass contact-angle cell containing 300 cc of a solution of

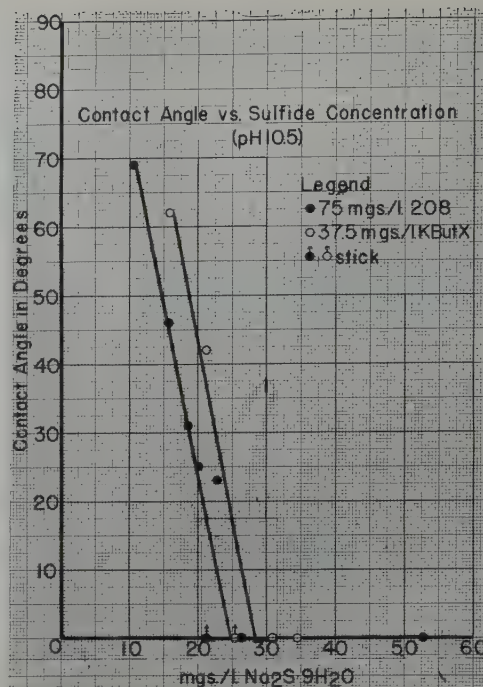


Fig. 16—Chalcocite in sulphide-air collector solution.

lime to bring it to pH 10.5 after addition of collector and sodium sulphide.

**Contact Angle Measurements:** When the prepared specimen is placed in the cell a stone diffuser is introduced through which air passes at the rate of 1 liter per min. The contact angle measurement is made within the first minute after the specimen is placed in position. We have observed that these contact angle values can be obtained only when the sulphide and collector concentrations are not varied during the preparation for measurement, namely, by washing into distilled water for angle measurement. Angle values obtained in the specified media

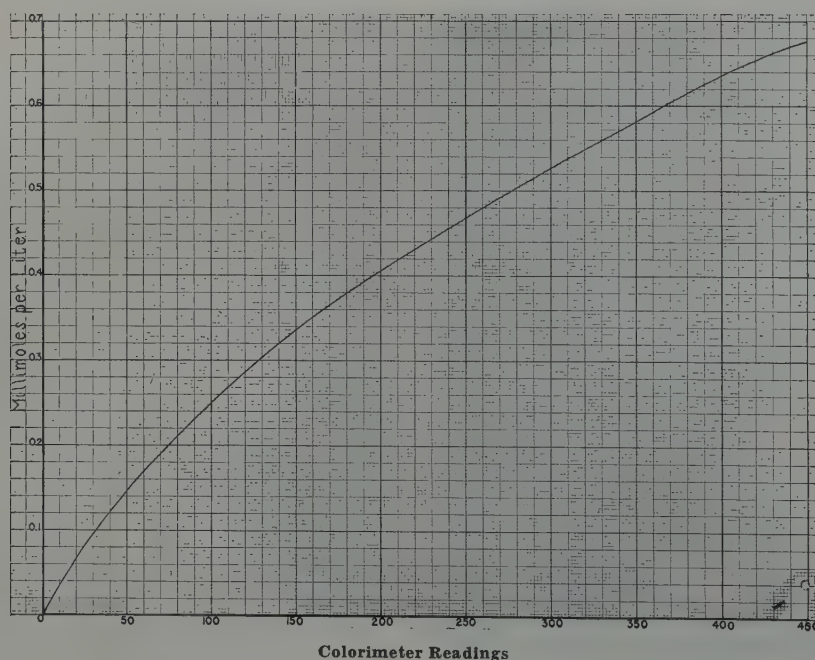
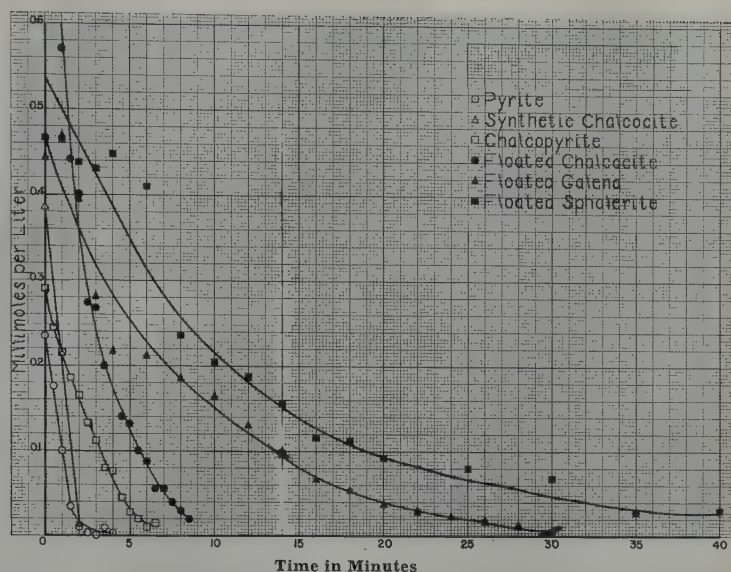


Fig. 17—Light absorption vs. concentration of  $\text{Na}_2\text{S}$ .



Fig. 18— $\text{Na}_2\text{S}$  disappearance, minerals in air.



remain during the first minute, but tend to reduce to zero in the next few minutes at a rate which depends upon the sulphide concentration.

**Current-potential Measurements:** The equipment and procedure has been described in Part I. When the concentration of sodium sulphide is less than the critical value, the solution is purged with nitrogen for a few minutes, during which period the potential is observed to fall rapidly. When a minimal negative current necessary to maintain say  $-600$  mv is attained, the air is admitted in place of nitrogen. The current changes only slightly, and a position of stability is soon reached for each succeeding potential setting on the curve. With sulphide in excess of the critical value no preliminary nitrogen treatment is necessary to cause the potential to fall readily to a stable negative value.

#### Part 5. Oxidation of Sulphide in Pulps

An intensely active reaction between dissolved sulphide and oxygen upon a suitably conditioned chalcocite surface has been made evident from the preceding electrochemical study. As a consequence one would expect that sulphide would be rapidly consumed in flotation pulps containing sulphide minerals such as chalcocite. It will now be shown that this is indeed the case.

**Experimental Procedure:** A Denver 8 x 8 in. laboratory ball mill was used to grind 30 g samples of various minerals with 0.5 g of c.p. lime and 200 cc of water. Grinding time was 5 min. The charge was transferred along with all washings to a Fagergren laboratory flotation cell of 4 liter capacity. Sulphide was added in an amount needed to satisfy any absorption by the mineral and leave a measurable excess. The machine agitator was run in the usual manner for flotation testing and pulp samples were withdrawn at intervals for determination of their soluble sulphide content. In those tests for sulphide disappearance anaerobically the flotation cell was fitted with a tightly drawn pliofilm cover. Nitrogen (water pump) purified with alkaline pyrogallol was passed into the air inlet cock of the flotation cell. In these tests samples were withdrawn through the bottom opening of the cell.

Temperature of pulps was 14 to 18°C (57.2 to 64.4°F).

Synthetic chalcocite was made by precipitation of cupric sulphide which was then melted. A solution of 9 lb of copper sulphate (commercial blue vitriol) was dissolved in 9 liter of water. Hydrogen sulphide was passed into this solution until precipitation of cupric sulphide was complete. After decanting several times with water, the material was filtered, oven-dried, and melted in a graphite crucible heated by induction. The melt was allowed to cool under a cover of sulphur, after which the crucible was broken, leaving a sound casting, the analysis of which is given in table II.

It was not possible to obtain chalcocite sufficiently free of gangue and pyrite inclusions, and therefore synthetic as well as the natural mineral was used.

**Analytical Procedure:** A sample of pulp was drawn and at once filtered through a small Buchner funnel until at least 50 cc of filtrate was obtained. Exactly 50 cc of this pulp liquor was mixed with 50 cc of a solution containing 0.2 pct glacial acetic acid and 0.02 pct lead nitrate. A brown color developed immediately. After standing for 2 min the light absorption of this colloidal solution was measured with a Klett-Summerson photoelectric colorimeter. From the previously obtained calibration curve for this instrument obtained with known additions of sodium sulphide, see fig. 17, the sulphide concentration in the pulp liquor was obtained. The sulphide solution used for calibration purposes was itself standardized by titration with standard iodine and thiosulphate, having been prepared from the best obtainable grade of sodium sulphide ( $\text{Na}_2\text{S} \cdot 9\text{H}_2\text{O}$ ) crystals.

**Results:** The time rate of disappearance of the sulphide from the pulps is presented in the curves of figs. 18 and 19 and in table IV. In a blank run made, using all of the reagents and conditions including ball mill grinding, but omitting the addition of any mineral, the sulphide concentration dropped at a very slow rate which was, however, appreciably greater than when lime and plain tap water were used without grinding. In the latter case the sulphide concentration appeared unchanged after more than 30 min.

Pyrite, chalcopyrite and chalcocite, both natural and synthetic, caused a disappearance of sulphide at a rate which would be considered rapid when related to time ordinarily allowed for conditioning and flotation in mill circuits.

Galena and sphalerite, though contributing appreciably to sulphide disappearance, do so at a relatively low rate.

Not only the mineral, but the air is essential to the rapid disappearance of sulphide with pyrite and chalcocite. This is shown in fig. 19 comparing the rate in nitrogen with that in air, other conditions remaining the same.

It is also apparent from the results of these tests that the minerals chalcocite and pyrite, which are most effective in the disappearance of sulphide, can repeat their performance substantially unimpaired when a fresh addition of sulphide is made following the complete disappearance of a previous addition. In this sense the process of sulphide disappearance may be said to be catalytic. This is not intended to imply that the role of the mineral, or of any product thereof, is that of a true catalyst inasmuch as this study does not afford any evidence to substantiate such a hypothesis. It is merely that from a practical standpoint the effect insofar as flotation

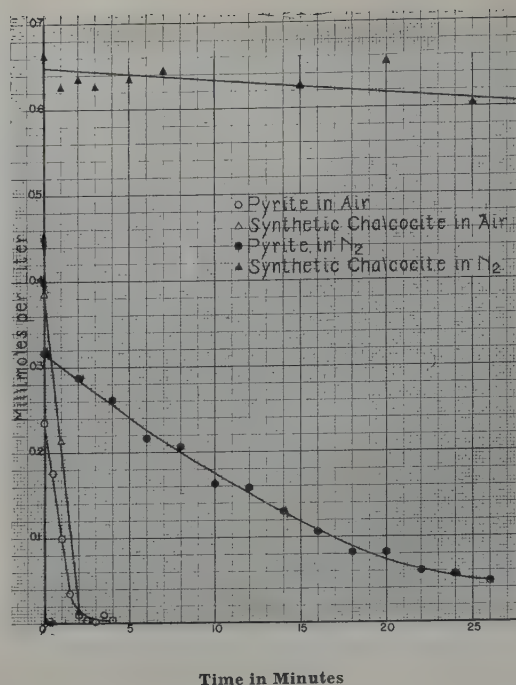


Fig. 19—Na<sub>2</sub>S disappearance in air and in N<sub>2</sub>.

Table II. Composition of Minerals

Mineral	Cu	Fe	Pb	Ni	Zn	S	SiO <sub>2</sub>	CaO	MgO	Al <sub>2</sub> O <sub>3</sub>
Pyrite	0.09	47.30	Trace		Nil	51.61	0.78			
Chalcopyrite	28.70	30.63	0.22	Trace	Trace	34.48	2.75	0.68	0.26	1.05
Chalcocite floated	35.05	4.22	0.46	Nil	1.07	11.62	45.31	0.80	0.19	0.64
Galena	Nil	0.16	84.81	Nil	0.11	13.35	0.70	Nil	0.21	Nil
Sphalerite		3.21	0.28		63.35	32.76	0.47			
Synthetic chalcocite	78.40	0.32		0.06		20.04	0.21			

Table III. Size Analysis of Minerals

Mineral	Galena			Pyrite			Sphalerite			Synthetic Chalcocite		
Mesh	Wt.	Pct	Cum. Pct	Wt.	Pct	Cum. Pct	Wt.	Pct	Cum. Pct	Wt.	Pct	Cum. Pct
On 65	0.02	0.02	0.02	0.4	0.4	0.4	0.3	0.3	0.3	1.8	1.8	1.8
100	0.03	0.03	0.05	1.8	2.0	2.4	1.4	1.4	1.7	5.5	5.6	7.4
200	0.20	0.21	0.26	12.8	14.0	16.4	8.9	8.7	10.4	25.6	26.1	33.5
270	0.33	0.34	0.60	9.5	10.4	26.8	7.6	7.5	17.9	13.9	14.2	47.7
325	0.19	0.20	0.80	5.3	5.8	32.6	4.5	4.4	22.3	6.7	6.8	54.5
400	17.3	18.0	18.8	22.9	25.0	57.6	16.8	16.5	38.8	26.7	27.2	81.7
560	16.9	17.6	36.4	6.4	7.0	64.6	18.1	17.8	56.6	11.6	11.8	93.5
800	14.3	14.9	51.3	8.5	9.3	73.9	11.0	10.8	67.4	2.5	2.5	96.0
1120	10.2	10.6	61.9	5.0	5.5	79.4	7.4	7.3	74.7	0.4	0.4	96.4
1600	7.0	7.3	69.2	2.7	3.0	82.4	4.2	4.1	78.8	0.1	0.1	96.5
Through 1600	26.5	30.7	99.9	16.2	17.7	100.1	21.7	21.3	100.1	3.4	3.5	100.0

Table IV. Sulphide Disappearance Rate for Successive Additions of 5 Pct Na<sub>2</sub>S·9H<sub>2</sub>O<sup>a</sup>

Time, Min		Na <sub>2</sub> S Millimols per Liter					Time, Min		Na <sub>2</sub> S Millimols per Liter			
0		0.08	0.10	0.115	0.136	0.145	0		0.57	0.385	0.445	0.608 <sup>b</sup>
1		0.065	0.045	0.064	0.024	0.050	1			0.194	0.174	0.604
2		0.02	0.003	0.009	0.005	0.024	2		0.010	0.012 <sup>a</sup>	0.010	0.600
3		0.01	0.005	0.008	0.020	0.030	3					
4				0.002			4					0.324
5					0.000	0.005	5					
6							6					0.012

<sup>a</sup> 10 cc 5 pct Na<sub>2</sub>S · 9H<sub>2</sub>O = 2.08 millimols Na<sub>2</sub>S <sup>b</sup> 40 cc added at one time

is concerned is equivalent to a catalytic oxidation.

It is to be understood that all reference herein made to sulphides does not necessarily apply to complex soluble sulphides such as polysulphides.

#### Acknowledgment

George Bobadilla and Mrs. Dolores Casella performed the experimental work upon which this paper is based.

Thanks are due to Phelps Dodge Corp. and to W. H. Osborn, Director of its Department of Development and Research, for permission to publish this work.

#### References

- <sup>1</sup> Badger and McCabe: Elements of Chemical Engineering. 2nd ed. Chap. 6. McGraw-Hill.
- <sup>2</sup> I. M. Kolthoff and J. J. Lingane: Polarography. 436. 1941. Interscience.
- <sup>3</sup> International Critical Tables, V., 64.
- <sup>4</sup> I. M. Kolthoff and J. J. Lingane: Polarography. 15. 1941. Interscience.
- <sup>5</sup> O. C. Ralston, et al.: Trans. AIME (1930) 87, 369.



# Canadian Deposits of Uranium and Thorium

by W. F. James,

A. H. Lang,

Richard Murphy

and

S. N. Kesten

The paper is a summary of information on Canadian deposits of uranium and thorium up to the end of 1948, prepared by geologists of Eldorado Mining and Refining Ltd. and by the specialist on uranium deposits for the Geological Survey of Canada. The paper contains a history of Canadian uranium and thorium discoveries, a working classification of the deposits, fairly detailed descriptions of the Eldorado mine and of properties in northern Saskatchewan, and shorter accounts of several other discoveries.

## Introduction—by W. F. James and A. H. Lang

This paper describes the geology and present state of development of uranium and thorium deposits in Canada. It is expected that this information will be added to that now being assembled from available American and British data to provide a background for engineers and others who may be called upon in the future to examine or search for radioactive deposits. Some generaliza-

---

W. F. JAMES is with Eldorado Mining and Refining (1944) Ltd., A. H. LANG is with the Geological Survey of Canada, RICHARD MURPHY, Member AIME, is with the Newmont Mining Corp. of Canada, and S. N. KESTEN is Field Geologist, Eldorado Mining and Refining (1944) Ltd.

AIME San Francisco Meeting, February 1950.

TP 2769 I. Discussion (2 copies) may be sent to Transactions AIME before March 30, 1950. Manuscript received May 9, 1949.

tions regarding the Canadian deposits are included, but many of these are necessarily tentative. The information contained in the paper is up to date for the end of 1948.

The information contained in this paper has been derived from many sources. Much of it is from the records of the Canadian government-owned Eldorado Mining and Refining (1944) Limited and from examinations of government and private properties made by officers of the Geological Survey of Canada. Some information on privately-owned properties has been quoted from other sources mentioned in the text.

The information has been assembled by W. F. James of Eldorado Mining and Refining (1944) Limited and A. H. Lang of the Geological Survey of Canada, who have written the sections for which they are credited. The description of the Eldorado mine is based on reports by Richard Murphy, formerly chief geologist for Eldorado; the report by Mr. Murphy was published previously by the Canadian Institute of Mining and Metallurgy,<sup>1</sup> to whom the writers are indebted for permission to republish. The section dealing with the geology of and development work in the Goldfields area, where Eldorado has important holdings, is by S. N. Kesten, field geologist for Eldorado. The geological diagram of the Goldfields area (fig. 2) was kindly supplied by A. M. Christie of the Geological Survey of Can-

ada, and is a much simplified version of his fairly detailed mapping of the region.

The paper includes a brief history of the most significant discoveries of radioactive minerals in Canada and a statement of the controls exercised by the Federal government. It discusses the known types of Canadian uranium and thorium deposits and provides a working classification. Several deposits are then described briefly, in the order of the classification.

### History

The first Canadian uranium discovery was made at Lake Superior and described in 1847 by J. L. LeConte, a distinguished American geologist. Several attempts to rediscover this occurrence were unsuccessful because the location was described indefinitely. A discovery known as the Camray, made in 1948 by a prospector searching for the old occurrence, has attracted wide attention and aroused considerable interest in the circumstances surrounding the early find. Therefore it may be of interest to describe them fairly completely.

LeConte stated<sup>2</sup> that he had received a specimen from the Lake Superior region, and continued:

This mineral forms part of a collection made by Mr. B. A. Stanard on the north shore of Lake Superior.... This mineral as I am informed by Mr. Stanard, occurs on the north shore of Lake Superior, about 70 miles from the Sault Ste. Marie, at the junction of trap and Sienite; the vein in which it is found is about two inches in width; but on account of its position (on the face of an almost perpendicular cliff) only a few specimens were obtained, and those with great difficulty.

The Mr. Stanard referred to was probably Benjamin Stanard, who was captain of a schooner operated on Lake Superior at that time by the American Fur Company.<sup>3</sup>

LeConte considered that the mineral was closely related to pitchblende, but because of slight physical and chemical differences between it and the typical pitchblende of Joachimsthal in Bohemia, he classed it as a new mineral which he called "coracite." He probably derived that name from the Greek word for "raven," in allusion to the black color of the mineral.<sup>4</sup>

J. D. Whitney published a note on the mineral in 1849.<sup>5</sup> He believed that it contained uranium of a different valence from that of typical pitchblende, and that it was more readily soluble in acids.

In 1857 F. A. Genth published the following description of the Stanard specimen:<sup>6</sup>

Dr. John L. LeConte kindly presented me with a specimen of the mineral from about 90 miles above Sault Ste. Marie on the north shore of Lake Superior, which he had described as coracite. Its great resemblance to pitchblende favoured the opinion that it was really nothing else... though it is interesting that it is so readily soluble in chlorhydric acid, this fact alone is not sufficient to separate it from pitchblende.

The first geological studies of the region were made by the Geological Survey and described by W. E. Logan in "The Geology of Canada," published in 1863. Under the heading "Uranium," the report states "An ore of this rare metal is said to occur at Mamainse," then continues with a condensation of the data published by LeConte and Whitney.<sup>7</sup>

The occurrence was summarized in four later reports of the Geological Survey, and officers of the survey made inquiries at various times in efforts to relocate the discovery, and answered numerous requests from prospectors for information on the occurrence. All efforts were unsuccessful, however, because of the conflicting locations given in the different publications. It now seems likely that LeConte's estimate of 70 miles from Sault Ste. Marie was nearly correct and that Genth's mention of 90 miles was unfortunate. It also appears likely that the use of the name Mamainse in "The Geology of Canada" followed an old custom of referring to the entire region as Mamainse, whereas more recent investigators have assumed that it referred to Mamainse Point which is considerably south of the recent discovery.

Robert Campbell, discoverer of the Camray deposit, stated that during the winter of 1947 to 1948 he studied all the references to the old coracite occurrence. He formed the Camray Prospecting Syndicate with a capital of \$4000, subscribed by Toronto backers, and spent the season of 1948 prospecting by boat and on foot along the shore of Lake Superior in the general region of the old discovery. During the earlier part of the season he had two assistants, but when they left he continued his search alone. He was equipped with a Geiger-Müller counter. On September 8, he obtained a strong reaction on his counter near the south wall of a gorge on Theano Point, close to the water's edge, and on close inspection found that the reaction came from a stringer of black mineral. Other stringers, some of which contained uranium stain, were found at intervals along the wall, over a total distance of about 600 ft. He staked 30 claims and recorded them on October 16.

The discovery agrees so closely with LeConte's description that it is considered a rediscovery. News of the find caused the greatest staking rush that has occurred in Ontario for over 20 years. Over 2000 claims were staked in a few months and other discoveries were reported, but snow has prevented examination of these up to the time of writing.

After 1904, radioactive minerals were identified in numerous specimens submitted from many parts of Canada. Practically all were from pegmatite deposits, but one occurrence in Ontario<sup>8</sup> and one in Quebec<sup>9</sup> consisted of small veins of uranium-bearing hydrocarbon related to anthraxolite. None of these occurrences has been developed commercially, but considerable work has been done on pegmatite deposits in the Wilberforce region of southern Ontario with a view to producing uranium as a by-product of mining for fluorite and other minerals.

In 1914, interest in radium caused the governments of British Columbia and Ontario to offer bonuses for discoveries of radioactive minerals in commercial quantities.<sup>10</sup> The rewards were not claimed and the offers were withdrawn.

The outstanding date connected with Canadian radioactive discoveries is 1930, when Gilbert Labine and E. C. St. Paul found pitchblende, cobalt, and native silver in veins at Great Bear Lake, just south of the Arctic Circle. The Eldorado Gold Mining Co. was organized to operate the mine and a radium refinery was built at Port Hope, Ontario. Great dif-



difficulties of transportation, climate, and treatment processes had to be overcome before production was begun in 1933. In 1940, difficulties of wartime operation caused a suspension of mining. During the years between the discovery of the Eldorado mine and the beginning of the last war several other pitchblende discoveries were made in the Great Bear Lake region, and one of these properties, now called International Uranium, produced a relatively small amount of pitchblende concentrate incidental to mining native silver. In 1935, pitchblende was identified in samples from the Nicholson gold-copper prospect at Lake Athabaska, but further development did not take place until recently.

When atomic fission was perfected during the war the Canadian deposits, previously of interest as sources of radium, assumed outstanding importance as British-American sources of uranium. It was decided, accordingly, that they should be placed under state control. This was done by placing a temporary ban on staking and mining for, and trading in, radioactive substances, and by purchasing the shares of the Eldorado company at an agreed price and transferring them to a "Crown company" operated by the Federal Government. The Eldorado mine was reopened in 1942 with as little publicity

as possible. Equipment was added to the concentrator, and the refining method was changed from a batch process to one of continuous extraction with emphasis on uranium production. The Crown company established its own air service and bought and expanded the barge transportation system that serves the Mackenzie District. In addition to servicing the Eldorado mine which is 1350 miles by rivers and lakes from railhead, it supplies the private freighting requirements of the Mackenzie District, other than those of the Hudson's Bay Co.

The Crown company engaged in prospecting and geological studies, with the aid of the Geological Survey of Canada. Several discoveries were made, staked, and tested. Emphasis has been placed on those near Lake Athabaska, which are only one-quarter as far from railhead as is the Eldorado mine, and where climatic conditions are more favorable.

A few years after the war, the Atomic Energy Control Board decided to permit private staking and mining for radioactive substances in order to stimulate uranium discoveries and production, but to retain the Eldorado company as a government-owned producer and as the only refining and marketing agency. Revised regulations were issued,

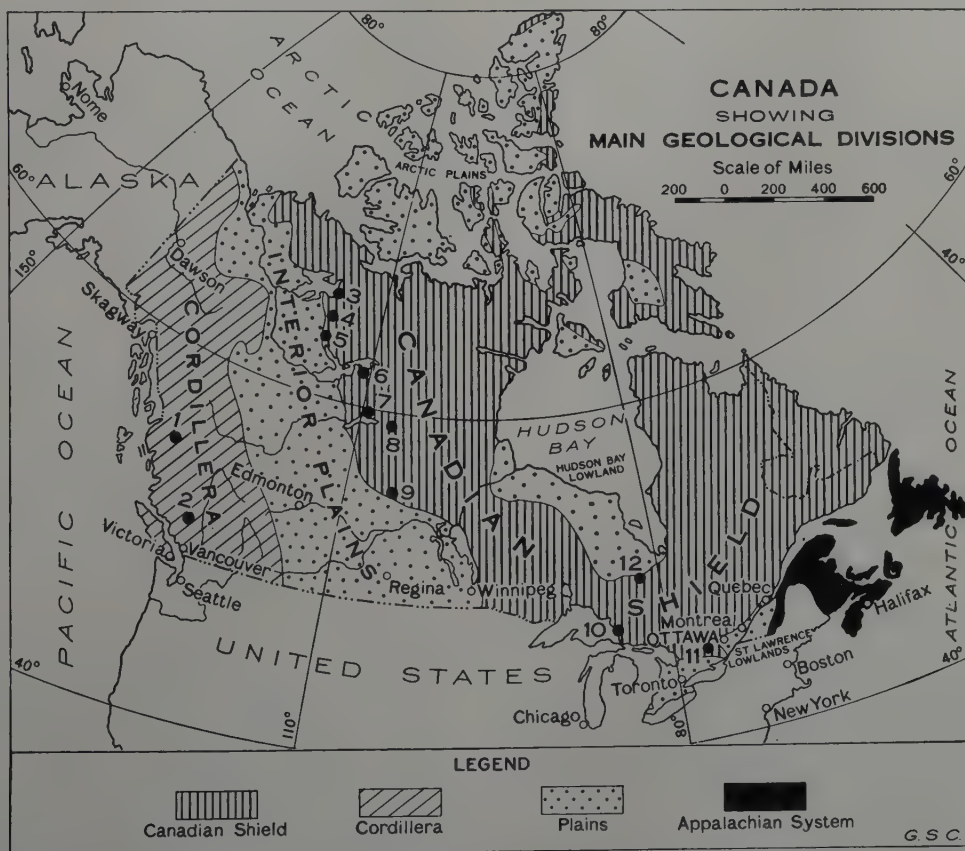


Fig. 1—Main geological features of Canada, and location of properties and districts mentioned in text.

- 1—Hazelton View. 2—Gem. 3—Eldorado Mine. 4—Hottah Lake. 5—Giauque.
- 6—McLean Bay. 7—Goldfields. 8—Black Lake. 9—Lac la Ronge. 10—Camray.
- 11—Wilberforce. 12—Otter Rapids (thorium).

therefore, in March 1948, and subsequent amendments have been made. Regulations pertaining to prospecting and development work are now essentially as follows:

1. Discoverers must notify the Atomic Energy Control Board or the Geological Survey of Canada giving the location and other details as soon as possible after there has been reasonable opportunity to stake, and before making any public announcement. Such information is treated confidentially.

2. Permits must be obtained from the Board before proceeding with any development work other than that required for making a discovery. This regulation is not made as a deterrent to development work, but so that the Board will always be aware of what is being done.

3. Permits must be obtained from the Board before removing any bulk samples or other shipments of radioactive material, except that specimens for display or ordinary samples for assay may be removed without an order.

The government guaranteed to buy, for five years, all acceptable ores and concentrates with a minimum uranium content equivalent to 10 pct by weight of uranium oxide, at a floor price of \$2.75 per pound of contained uranium oxide. The price is f.o.b. rail at any point in Canada, the government absorbing the losses and refining cost. This offer was later extended for another two years, to March 31, 1955.

The attitude of the Canadian Government now is to encourage in every way prospecting for uranium minerals by mining companies and individuals. It is recognized that, initially at least, considerable technical help will be needed, and this is given principally by the Geological Survey, the Bureau of Mines, and several Provincial Mining Departments. It is felt that a prospector might hope ultimately to acquire the same compensation from a discovery of a uranium deposit as he would from the discovery of a gold prospect of comparable size and grade. As a result of the government action, interest among prospectors was aroused and throughout the past season, prospecting has been active in many of the Provinces of Canada, principally in Ontario, Saskatchewan, and British Columbia, and in the Federally administered Northwest Territories. Although to date none of the recently discovered properties has reached the production stage, it is expected that eventually a number of properties will produce significant amounts of uranium concentrates. In the meantime, prospecting by Eldorado has continued. Underground operations have been undertaken on one new Eldorado property and are planned on several others. Publicity regarding discoveries and properties under development is permitted, but production and ore-reserve figures are not allowed to be published.

### Types of Canadian Deposits

Uranium and thorium occur in Canada in deposits of several different types. Fig. 1 shows the main geological divisions of Canada and the location of properties and districts. All of these types are listed below, for the sake of completeness, but only pitchblende deposits believed to be of hydro-

thermal origin are of known economic importance, at least at present.

#### Uranium deposits:

1. Original constituents of granitic rocks.
2. Pegmatite deposits.
3. Hydrothermal deposits
  - A. Veins, stringer systems, and disseminations
  - B. Giant quartz veins.
4. Bedded deposits in sedimentary strata.
5. Deposits of secondary uranium minerals in gossans, etc.

#### Thorium deposits:

1. Pegmatite deposits.
2. Hydrothermal deposits.
3. Bedded deposits in sedimentary strata.
4. Placer deposits.

Most of the discoveries are in the Canadian pre-Cambrian Shield. A few mineral occurrences have been reported from the Appalachian Region, and some discoveries that may prove to be important have been made recently in the Cordilleran Region.

The present uranium discoveries in the Shield are strikingly restricted to its western and southern margins. This may be partly due to accessibility, as most of the discoveries have been made near the large lakes that flank the Shield, and more complete prospecting of other regions may alter the pattern of discoveries. On the other hand, the distribution may be explained more theoretically by the facts that many of the age determinations on pitchblende from the Shield indicate Proterozoic and early Paleozoic ages, and that these deposits occur in belts of Proterozoic mountain building. This suggests that other areas of Proterozoic orogeny may be worth prospecting.

Pitchblende-bearing veins and stringers occur in many types of sedimentary, volcanic, and granitic rocks. Therefore no sweeping generalizations regarding favorable host rocks can be made, but some generalizations can be made for individual districts. Many of the deposits in the northwestern part of the Shield occur in or alongside prominent north-easterly-trending fault zones. The individual ore shoots constituting a deposit are generally rather small, and they tend to occur in tension fractures related to the fault zones.

The minerals with which pitchblende is most frequently associated are cobalt-nickel minerals, native silver, and hematite. It is convenient to divide the pitchblende deposits, other than the giant quartz-vein type, into two general classes, one containing complex mineral assemblages and the other containing chiefly pitchblende, hematite, carbonate minerals, and quartz. The writers consider the Camray occurrence a special example of the latter class, in which no hematite has as yet been found.

Most of the other types of deposits are discussed briefly in later sections of this paper.

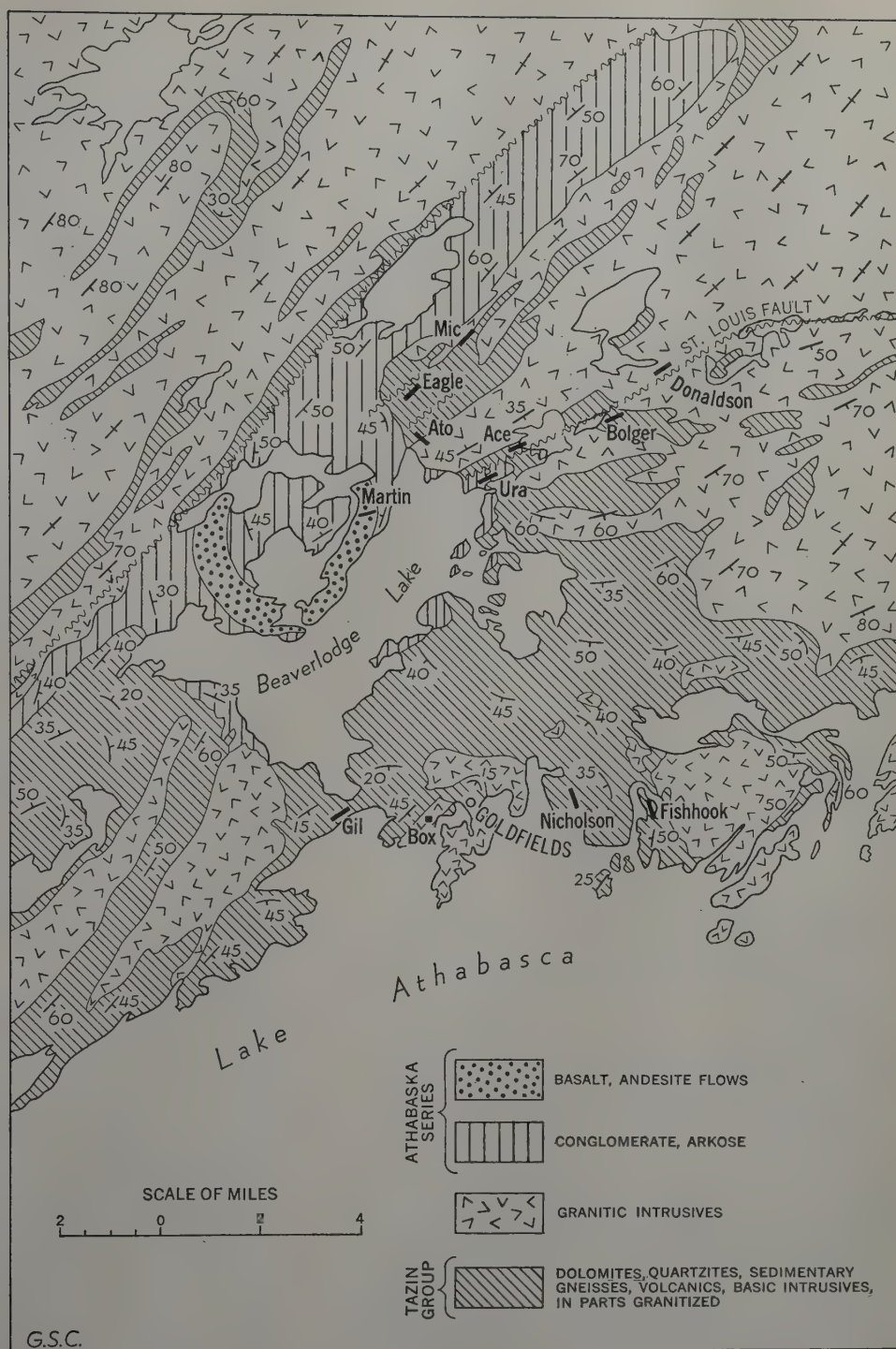
### Granite and Pegmatite Deposits

Many samples of granitic rocks have been shown to contain about 0.003 pct uranium oxide equiva-



**Fig. 2—Generalized geological map of Goldfields region.**

(Map by A. M. Christie, Geological Survey of Canada.)



lent, and contents up to 0.01 pct are not uncommon (H. V. Ellsworth, personal communication). All information available at present indicates that the uranium in these rocks occurs in original rock-forming minerals, and has not been introduced later.

Minor quantities of uraninite and, to a lesser extent, of other uranium minerals, have been found in numerous pegmatite bodies associated with granitic rocks. Such deposits occur in widely separated parts of Canada, but particularly in the southern part of the Canadian Shield, in a belt extending from Lake Huron to the Gulf of St. Lawrence. None of these has yet been proved to be a commercial source of

uranium.<sup>11</sup> The most promising pegmatite deposits seem to be those containing other valuable minerals such as feldspar, fluorite, mica, apatite, beryl, and lithium minerals, from which uraninite might be recovered as a byproduct. Other uranium-bearing pegmatite deposits were found at Lac la Ronge, Saskatchewan, in 1947 and 1948. In view of the large number of uranium-bearing pegmatites already known, prospecting for others cannot be recommended highly until such time as some of the known occurrences are shown to be productive.

Considerable thought has been given to the question of whether areas containing uranium-bearing pegmatite deposits, or granites with higher-than-

average uranium content, would be favorable localities for the discovery of greater concentrations in the form of hydrothermal deposits. It seems quite probable that such granitic rocks would have provided vein deposits but, theoretically, the hydrothermal solutions would be expected to have migrated farther from their source than did the pegmatite-forming material. Consequently, hydrothermal deposits, derived from the same sources as the granite or pegmatite, would not be expected to occur in the immediate vicinity of the pegmatite, but in the border zones of the intrusions, and many such deposits may have been removed by erosion. The pitchblende deposits that do occur in granite seem to be much later than the granite, and are believed to have been derived from deep-seated sources younger than and unrelated to the host rocks of the deposits.

#### The Eldorado Mine—by Richard Murphy

The Eldorado mine (fig. 1) is at Labine Point on Great Bear Lake, 28 miles south of the Arctic Circle. The region is one of sparse vegetation, and ice remains in the lakes for all but a few summer months. The general geology of the area was described by D. F. Kidd of the Geological Survey,<sup>12</sup> and later work was done at and near the mine by other officers of the Survey and by the writer.

**General Geology:** The mine is in the pre-Cambrian Shield and all the rocks nearby are believed to be of Proterozoic age.<sup>13</sup> They may be tabulated as follows, with the oldest rocks at the bottom of the table:

##### Intrusives

9. Late diabase
8. Early diabase
7. Aplite
6. Granite
5. Diorite

##### Echo Bay Group

4. Porphyritic lavas and pyroclastics
3. Feldspar porphyry
2. Sediments
1. Massive tuff

At LaBine Point the principal rocks are the lowest recognized members of the Echo Bay group. At the base of the series is a fine to dense, dark uniform rock of uncertain origin, but presumed to be a tuff. Very little is known as to its relation with the succeeding sedimentary member, which is highly quartzose and varies widely in appearance. It includes fine- and coarse-banded quartzites, dense cherts brecciated by irregular chloritic seams, and vaguely banded metamorphosed silicic rocks. These are buff to pink in color, much stained with hematite. Magnetite is a common constituent as disseminations and veinlets and replaces certain bedded zones to a degree that suggests iron formation. Thinly bedded argillaceous horizons are traceable for a few hundred feet on surface and underground, but for the most part the different phases of the quartzite are distributed in no regular manner. There are very minor amounts of limestone and conglomerate.

The feldspar porphyry is a fine-grained crystal-

line rock with small scattered phenocrysts. Where freshest, the rock is purplish-gray and the phenocrysts are light in color and prominent. With alteration the rock is generally redder, in extreme cases grading into a dense, reddish brown phase. In least altered areas the porphyry is uniform, but through the zones of alteration, it is very irregular in character. Its identity is rarely in doubt, however, because there are usually recognizable remnants.

The porphyry forms bands and masses in the sediments and has not been found in other members of the Echo Bay group. In general, the bodies of porphyry conform roughly to the structure in the sediments and could, therefore, be either sills or flows. Since the contacts are obscure, and are rarely placed with certainty within 5 ft, the details are not helpful. On Cobalt Island, the porphyry sends several stubby apophyses into the sediments though the contact is elsewhere strictly conformable. There are a few instances in which masses of porphyry appear to truncate the sediments at low angles. At one locality, now concealed by tailings, several blocks of sediments were found as inclusions in the porphyry up to a distance of 20 ft from the contact. While not conclusive, the evidence in the mine area is that the porphyry is an intrusive.

A little further afield structures anomalous to an intrusive are found in the porphyry. Exposures show faint brecciation and occasional banding, and in one instance well defined pillows. Breccia commonly appears near the upper part of the porphyry band and might be interpreted as a fragmental zone topping a flow. Since there are other explanations for these relatively obscure structures, they are not considered to be proof that the porphyry is not an intrusive, but merely serve to keep the question open.

In contrast with the sediments, the overlying lavas and pyroclastics are fresh in appearance. The flows are porphyritic andesites with a dense, gray groundmass, and prominent feldspar and hornblende phenocrysts. In certain horizons amygdules are abundant. The interbedded pyroclastics consist of breccia, agglomerate, and thinly-bedded tuff. The volcanics invariably weather brownish as opposed to the decidedly reddish color of the underlying formations. Higher up in the series conglomerate and sandstone are found with the flows, but no important amounts occur near the mine.

The oldest major intrusive is diorite, a medium to coarse grained rock, showing considerable variation. The diorite occurs some two miles northeast of the mine as a large elongate body, cutting across the Echo Bay group. A number of small stocks have been found intruding both porphyry and sediments.

Granite underlies Great Bear Lake offshore from LaBine Point, outcropping on many islands and at several places on the mainland. The rock is coarse-grained, reddish, and contains a fair proportion of hornblende. It clearly intrudes the Echo Bay group. At normal contacts the granite shows a rough parallel banding for about 10 ft, and numerous partially digested inclusions. Granite tongues have been encountered at a few places in the most westerly underground workings.

Pinkish aplite dikes, varying in width from a few



inches to 20 ft, are frequently seen cutting the granite and older rocks. No aplite has been recognized beyond a thousand feet from the granite.

Intrusives of two or three different ages are identified as diabase. The early diabase includes a number of fine-grained, dark dykes, sometimes amygdaloidal. These are steep in attitude and are known to be earlier than the pitchblende veins. The late diabase is a medium grained, grayish rock of variable texture. It occurs as a flat-lying sill or dyke, showing columnar jointing. The outcrop of the diabase throughout the district is marked by prominent cliffs. Remnants of the diabase in the vicinity of the mine suggest that the dyke formed a continuous domed horizon across the country only a short distance above the present surface. Narrow offshoots of this dyke are found in the mine workings cutting the veins. This is the only intrusive known to be younger than the pitchblende mineralization.

As exposed at the mine, the massive tuff may be 500 ft in normal thickness, while the sediments-porphyry complex approximates 1500 ft. Measurement of a section across the volcanics has never been completed, but the thickness of these rocks undoubtedly exceeds 2000 ft.

**Structure:** The structure in the Echo Bay group is monoclinial. The strike of the series is northeasterly and the dip is 40° to the southeast, away from the granite. Steep dips and local folding, as at LaBine Point, are found near the granite contact, but average dips decrease leaving the granite until beds are nearly horizontal.

In the mine area the recognized structure involves only the three lowest members of the Echo Bay group, namely, the massive tuff, sediments, and porphyry. These three members are folded on axes striking north-northeast with folds plunging slightly to the north. The details have been determined principally by tracing the porphyry bands, although some confirming evidence has been found by mapping in the sediments. A geological section through the mine would show on the west a syncline with porphyry enclosing sediments to a depth of 1000 ft, and on the east an open anticline with the gently rolling porphyry sheet outcropping over a wide zone.

From the disposition of the folds in respect to the granite, it would appear that the folding can be coupled to the intrusion of the batholith. No such obvious association can be claimed for the major faults in the area, to which the vein fractures are related. These faults fall roughly into two groups, striking east of north and northeast. They have a strong topographic expression and dislocations measured in miles. Numerous lesser faults and fractures are developed in complex pattern over a widely extended area. These "breaks," deeply eroded, form the many narrow valleys, clefts, and crevices which are so noticeable a feature of the topography. The majority have a northeasterly bearing, but some strike south of east, particularly where the fractures fan out in horsetail arrangement from a main break. The faults are often sealed by mineralization of the quartz-hematite type, in extreme cases forming veins hundreds of feet in width.

The period of faulting has been sufficiently extended to affect all the rocks in this area, though the later diabase has suffered only minor displacement. Where diabase cuts through a fault zone there are always certain mud seams which displace it a few feet.

The complexity of the faulting and the brecciated "open" appearance of the fault zones suggest that the movements have taken place at shallow depths. It seems unlikely that the faulting can be ascribed to the deep-seated forces favoring the emplacement of the granite.

The vein-bearing structures are regularly disposed across LaBine Point at about 600 ft intervals, dipping to the north and with strikes converging to the northeast. No. 1, the most southerly, is a strong shear zone up to 40 ft in width, which has been followed by workings and diamond drill holes for over 5000 ft. The apparent horizontal displacement is 300 ft and the movement is left-handed. No. 2 and 3 are structurally minor breaks. They consist of narrow subparallel shearings and related tension members. Displacements do not exceed 20 ft and movements in opposing directions have been observed in different parts of the zones. The northerly vein, called the Dumpy, or No. 5, vein, is another strong shear roughly parallel to No. 1. There are other lesser fractures but all fit into the pattern, fanning out through the block of ground between the main shear zones. This pattern of shearings and supplementary fractures is frequently repeated in the area, but no other group within a radius of three miles has been recognized as of equal economic importance.

**Mineralogy:** The mineralogy of the pitchblende deposits has been treated in detail by Kidd and Haycock,<sup>14</sup> who have identified over forty metallic minerals and described the paragenesis. The following description is concerned only with observations in the field.

The veins are composed primarily of quartz, carbonate, and hematite in varying proportions with a minor amount of chlorite. Pitchblende, chalcopyrite, nickel-cobalt minerals, bismuth, native silver, argentite, galena, and pyrite are the metallic minerals readily identified. Botryoidal pitchblende was common in the upper parts of the mine but is now rarely encountered. Native silver also was found chiefly in the upper workings. Deposition probably occurred in four distinct stages, and has produced different types of veins depending on the conditions of the fracture zones. In large shear zones stockworks have formed; in narrow fractures there are clean-cut, persistent veins. All veins carry numerous inclusions and, as an extreme instance, No. 3 vein is a breccia cemented by a little vein material.

The No. 1 zone shows the most complex relations. The earliest filling has been quartz with seams and inclusions of chlorite, and a little carbonate. These veins carry some pyrite, but no other metallic minerals. The quartz shows no banding and may, in fact, be hypothermal.

The succeeding quartz-hematite mineralization is the most abundant in the zone. Veins of this type are up to 10 ft in width and form stockworks up to

40 ft wide. The quartz is banded and stained with hematite, forming parallel and concentric structures. Numerous vugs, occasionally a few feet in diameter, are lined with crystals of carbonate, hematite, quartz and chalcopyrite. Carbonate, apparently, constitutes the end phase. Parts of the vein are a dense, solid, rose-red chert. Veins of the hematitic quartz angle across masses of the chloritic quartz. In detail the contacts show intimate penetration into the earlier quartz and it is probable that in the process of refilling the zone the chloritic quartz has been largely replaced.

Narrow bands of sulphides, arsenides, bismuth, silver, and pitchblende are found in the quartz-hematite veins. They show a consistent preference for the cherty phases of the vein, and they are less apt to appear in the prominently banded and vuggy portions. Since these shoots of ore appear in only a small proportion of the quartz-hematite zone, they may represent a separate, third stage of mineralization. On the other hand, the association with certain phases of the hematitic quartz suggests a fairly close relation.

The segregation of the various metallic minerals separately in lenses in the vein is characteristic, although megascopic intergrowths of these minerals on the scale of a hand specimen are not unusual. Thus pitchblende and nickel-cobalt minerals may occur as separate stringers several inches apart, and veinlets of bismuth and calcite may be similarly isolated. This veining with various minerals would appear to offer many opportunities for determining the sequence of the metallic minerals. Such opportunities are rare, however, and the details in observed instances are conflicting.

The metallic minerals are not coextensive. The nickel-cobalt minerals have a wider distribution in the vein than the pitchblende and the latter may occur with only negligible amounts of the other minerals. Silver, though shown by assay to be present throughout the vein in visible amount, is very erratically distributed. It occurs as dendrites in buff carbonate veins, as intergrowths with the nickel-cobalt minerals, and as plates and wires in the pitchblende. On the field evidence some of the silver is surely placed as the youngest of the metallic minerals.

The pitchblende may form solid bands up to a foot and masses up to several feet in width. It is most commonly a persistent, lensing vein a few inches wide, or a lacing network of stringers with some coarse dissemination. The limits of the ore are generally well-defined, and once beyond those limits only an occasional spot of pitchblende is found. Measurements of the radioactivity in the remainder of the vein show that the uranium minerals are absent.

In the last recognized stage in the No. 1 zone, small veins of quartz, carbonate, and chalcopyrite have been introduced, cutting all other veins. Vugs in the older veins have also been filled, or lined with brilliant clusters of crystals. This mineralization is younger than the later diabase. Where the dyke cuts No. 1 zone, it is itself cut by a few veinlets of quartz and carbonate. Carbonate veins in the diabase on Cobalt Island contain metallic bis-

muth and nickel-cobalt minerals. No silver has been found in the association, but at Gunbarrel Inlet, 40 miles south, a calcite vein cutting the younger diabase does carry silver. It is possible that all the silver may be attributed to this last stage. Galena, which is present with the silver in small amount, may also be related. Narrow veins of galena, normal to No. 1 zone and offsetting its contacts slightly, are clearly a late phase of the mineralization.

Extensive and prolonged movement has taken place on the No. 1 zone. The footwall fault is strongly developed, containing up to 3 ft of gouge. Branch faults lace their way through the zone from side to side creating extremely weak ground. Faulting has continued after the intrusion of the later diabase, as indicated by minor displacements of the dyke contacts. A few veinlets of quartz and carbonate, possibly belonging to the youngest stage of mineralization, are found in the gouge. In a few places, lenses of massive chalcopyrite several feet in width, with a dense and "muddy" appearance, appear to have replaced the gouge.

The Dumpy zone resembles the No. 1 although deposition has taken place on a much reduced scale. The zone is marked by an abundance of chalcopyrite, in irregular masses, of the same type noted in the No. 1 zone.

No. 2 and 3 veins each have their own characteristics. There is no evidence here of the extended period of mineralization recognized in the large shear zones. Various sections of these veins show distinct and different parts of the general mineralization. The openings must have been accessible for relatively brief and critical periods.

In part, No. 2 vein is a beautifully banded carbonate including some rhodochrosite. Pitchblende occurs as seams and botryoidal crusts, and a few sections are rich in silver. Hematite, chalcopyrite, and nickel-cobalt minerals are common, and fluorite may be present. There is no faulting and everything suggests quiet and uninterrupted deposition in an opening probably produced by tension. The remainder of the No. 2 lies along narrow chloritic shearings. Strong quartz-hematite veins carry some of the pitchblende. Other shoots are sparsely mineralized zones in which pitchblende is spotted through the schist accompanied by a few stringers of quartz and carbonate.

No. 3 zone is also sparse in vein material. The breccia, up to 5 ft wide, is sharply defined and contains numerous fine and coarse angular fragments in a dark matrix. Disseminated pitchblende and chalcopyrite, occasional masses of pitchblende, and a very few quartz and carbonate veinlets have been introduced. The breccia zone may die out leaving only a thin gouge seam to mark the zone, but the pitchblende may persist as spots in the gouge. The pitchblende in No. 3 forms a high proportion of the total mineralization and this is true of many other minor occurrences in the district.

The characteristics of these veins are those of deposits formed under moderate conditions of pressure and temperature. The veins seem to belong in the upper part of the mesothermal group.

An unusual feature of the mineralization are the coarsely crystalline aggregates which replace sedi-



ments, and porphyry to a minor degree, in highly irregular fashion. These aggregates consist of amphibole, epidote, apatite, carbonate, and magnetite, with lesser amounts of garnet, feldspar, chlorite, and pyrite. The skarn mineralization is not obviously related to any of the veins nor to the altered zones, but the amount of radioactivity within the areas of occurrence implies some connection; perhaps it is a product of reaction between lime-rich phases of the sediments and the invading solutions.

**Alteration:** The alteration in the mine area is prominent, and the "baked" appearance of all formations, except the later diabase, has been frequently mentioned. Both the alteration on the veins and the more general alteration are of the same type, and result in wide-spread discoloration of the rocks by hematite and the obliteration of original textures. This so-called red alteration, undoubtedly related to the quartz-hematite period of mineralization, affects the quartzose rocks most severely, but where alteration was intense there is little selectivity. The rocks are then reduced to a dense reddish "jasperoid." The exact nature of the alteration has not been determined, but quartz, hematite, magnetite, sericite, chlorite, and carbonate are obvious constituents.

Away from the mine the degree of alteration falls off, but the mineralization is so pervasive that in no case can it be said that examination has been carried out of the zone of alteration. The distribution of the alteration points to the mine as being a center of mineralization in the district, and indicates that the veins and alteration have a common hydrothermal source. In a geophysical examination of the area, A. A. Brant has noted an unusually high magnetite content of the rocks on LaBine Point. The percentage of magnetite decreases away from the mine, thus giving further evidence as to the locus of mineralization.

The envelope of reddish "jasperoid" enclosing the veins is the most intense form of alteration. Within 4 or 5 ft of the vein there are seldom even recognizable remnants of the wall rock. Although the width of the altered zone is roughly proportional to the width of the vein, there are examples of unusual penetration, probably on numerous small mineralized fractures. The red alteration may mark areas in which veins are concealed and is therefore a useful guide in exploration.

In the larger shearings, in addition to the red alteration, there are strongly developed talcose and chloritic zones. The chloritic phase is poorly veined and has probably not been long accessible to the altering solutions. The buff and reddish talcose phases are various advanced stages accompanying abundant vein deposition.

**Factors Controlling Ore Deposition:** The ore bodies are, for the most part, narrow lensing streaks of pitchblende in a much larger volume of vein material. Shoots of minimum stoping width range in length from 50 to 700 ft, and have been followed vertically for more than 600 ft. The ore bodies locally widen to as much as 15 ft because of the occurrence of multiple stringer zones and masses of pitchblende.

Two of the important factors in ore deposition

are the wall rock, and the nature of the openings. The best evidence of their influence at Eldorado is offered in the case of veins in narrow fractures. In the large shear zones, where the proportion of total vein material to ore is very high, it is difficult to demonstrate any effective control.

The optimum of control must have existed during the period of formation of No. 2 vein. The zone is a composite of shear and tension members, over 2000 ft in length, each part of which carries some ore. The rocks intersected by the zone are sediments, porphyry, and early diabase. Most of the ore shoots have been found in sediments apparently favoring the finely banded phases. Some ore occurs where the shear has followed the contact of the sediments and early diabase. No ore is found in the porphyry except as the ends of the larger ore shoots in the adjacent sediments. In longitudinal section the termination of the ore shoots at the trace of the porphyry contact on the vein is very well shown. Thus, while different types of sedimentary rocks as well as the early diabase are favorable to ore deposition, the porphyry is decidedly unfavorable.

The noticeable constriction of the fractures on entering the porphyry has not resulted in the exclusion of vein material and does not, therefore, fully explain the absence of ore. The sediments have been affected by vein alteration to a much greater degree than the porphyry. They must have exercised a correspondingly greater influence chemically on the vein-forming reactions. The more favorable chemical environment of the openings in the sediments and the restricted passage of solutions through the porphyry have probably combined to localize the ore shoots.

The greater proportion of ore in No. 2 occurs in the tension member. This fracture is filled with a well-banded carbonate, carrying better than average percentages of pitchblende. The principal carbonate mineralization is a late phase of the quartz-hematite stage, and the tension fracture must have opened up during the invasion of the carbonate solutions. Not only did the fracturing occur at precisely the right time, but also in view of the grade of the vein, conditions must have been ideal for pitchblende deposition. There are many similar pitchblende showings in the area, too small to be minable. In these showings 1 to 3 in. of vein material may fill a well-defined tension fracture for 20 ft or so. Pitchblende may amount to as much as 50 pct of the vein and in every case the proportion of pitchblende to vein material is higher than average.

The tension fracture in No. 2 zone has an interesting relation to the shear members and also to the local rock structure. The zone cuts across a small syncline of sediments lying in porphyry. The tension fracture is at a low angle, both on plan and section, between two parallel shearings. At the same time it is confined to the sediments, extending from the porphyry on one flank to the porphyry on the other. Maintaining the same relative positions on successive levels, the fracture shortens appreciably with depth and will presumably die out when the bottom of the sedimentary trough is reached.

In No. 1 zone, the large veins have been localized by slight changes in attitude of the shear. At least this is suggested by development work to date on

No. 1 zone. It will require much more development, fixing the position of the vein accurately on several levels, before definite conclusions can be drawn. The ore shoots within the vein occur without regard for the wall rock. It is possible that ore deposition has been related to certain phases of the quartz-hematite vein but no systematic arrangement of the ore shoots has resulted. Most of the pitchblende is found on or near the footwall, but other lenses are so disposed across the zone as to require thorough exploration from wall to wall.

**Source of the Ore Solutions:** It is usually accepted that ore deposits are related in time to one or another of the intrusives in an area. Locally a closer relationship is suggested by the distribution of the ore in relation to an intrusive. No evidence was found at the mine to link the pitchblende deposits closely to one particular intrusive. The more general consideration, that the ore solutions have had a common origin with the granite, is open to several objections.

In the first place, the faults and fractures have been formed at shallow depths as shown by their brecciated and open appearance, a fact which contrasts with the deep-seated emplacement of the granite. Second, the faulting is regional in its distribution and cannot be related solely to the mass of granite lying west of the mine. Third, the faults displace several intrusives younger than the granite thus introducing the likelihood of a considerable interval between the granitic intrusion and faulting. Four, some of the faulting cuts the later diabase and, if these final movements are attributable to the same forces, the interval is great enough to make improbable any genetic relationship to the granite.

The major faults have tapped an abundant source of solutions, to which the "giant" quartz veins bear evidence. If granite differentiates have originated at the same time they have not been recognized.

The later diabase can be coupled to the ore deposits more closely than any other intrusive. Some of the silver is known to be later than the diabase and the association is elsewhere accepted as evidence of a common source. The diabase, however, clearly intersects the pitchblende veins. It has not been subjected to the general alteration. There are no general considerations supporting the proposed relation between pitchblende ores and diabase.

The conclusion is that during an extended period of faulting solutions from an undisclosed source have risen to form the relatively shallow-seated pitchblende deposits. This period of mineralization came to a close but activity was renewed during intrusion of the later diabase with the consequent deposition of silver.

**International Uranium Ltd.—by W. F. James  
and A. H. Lang**

Nine miles south of the Eldorado mine a native silver-pitchblende deposit was operated intermittently prior to the war. The deposit consists of three veins containing mineralization of the same general character as that of the Eldorado mine, but occurring in granodiorite near the contact of a younger granitic intrusion.

A 25-ton concentrator is reported<sup>15</sup> to have produced 6933 lb of  $U_3O_8$  in 1938 and 1939. The present company, called International Uranium Mining Co. Ltd., was incorporated in 1942. Operations were suspended for a time, and the mill was destroyed by fire. In 1947 the company shipped high-grade silver ore. Some pitchblende ore shoots were outlined in old workings. Shaft sinking was resumed in 1948, with a view to driving two deeper levels in the hope of outlining additional pitchblende ore.

**Goldfields Area, Saskatchewan  
by S. Norman Kesten**

The presence of radioactive minerals in the Goldfields area was recognized first in 1935 on the present Nicholson property, two miles east of the former mining town of Goldfields on the north shore of Lake Athabaska (fig. 1 and 2). A deposit of pitchblende was found while the property was being explored for copper, and although some work, including the driving of two small adits, was done on the showing, it aroused little interest at that time. However, during World War II, when intensive exploration for uranium minerals was undertaken, the Nicholson again received attention. In September 1944, the showings were examined by A. W. Jolliffe and R. F. Murphy. The former represented the Geological Survey of Canada while the latter acted for Eldorado Mining and Refining (1944) Limited, the Crown company formed to mine the famous pitchblende deposit at Port Radium on Great Bear Lake. As a result of this examination, Eldorado staked forty-seven claims adjoining the Nicholson property on the north and east.

Detailed exploration of these claims was carried out during the field seasons of 1945, 1946, and 1947, including 24,000 ft of diamond drilling. At the same time, the camp established at Fish Hook Bay as a base for this work also served as a base camp for expanding operations within a 20 mile radius of Goldfields. By the end of 1948, the Crown company had staked 193 claims in the area and had carried out detailed exploration on about two-thirds of the ground covered. Except for recent activity on the part of the Nicholson company, Eldorado has done virtually the only work on radioactive deposits in the area, and the following remarks will deal almost entirely with the latter company's operations and findings.

**General Geology: Formations:** The rocks of the Goldfields area are of pre-Cambrian age. The oldest are those of the Tazin group, consisting of dolomite, quartzite, argillite, conglomerate, flows, and fragmental rocks. There is some doubt as to whether these are Archaean or Proterozoic. Their deposition was followed by intrusions of granitic rocks. On these were laid the sediments classed by some writers as the Beaverlodge series, which in turn were intruded by basic rocks. A second granite intrusion was followed, apparently after a long period of erosion, by deposition of the sediments of the Athabaska series, which is Proterozoic. This sedimentation was accompanied by the extrusion of basic lavas and the intrusion of small basic dykes. These are the youngest rocks in the area; erosion has brought the land surface down approximately



to that which existed at the time of their deposition so that there remain only remnants of the Athabaska where these rocks were deposited in basins or subsided in the course of late faulting and folding.

**Structure:** Earth movements appear to have followed each period of sedimentation outlined above. Structurally the area may be divided into four parts, each a separate geographic district (fig. 2).

1. The Cornwall Bay district lies along the north shore of Lake Athabaska, from Elliott Bay on the west to MacIntosh Bay on the east and includes Goldfields, Cornwall Bay, and Fish Hook Bay. Here Tazin and Beaverlodge sediments have been folded in a broad, open syncline, plunging gently to the south, forming a semicircle about the town of Goldfields. The folded formations are mainly thick beds of quartzite and dolomite which have been intruded by gabbro sills and by the younger, pre-Athabaska granite.

2. The Beaverlodge Lake district embraces Beaverlodge and Martin lakes which are to the north and west of Goldfields. The margins of Beaverlodge Lake are underlain by conglomerates and sandstone of the Athabaska series which, in most cases, appear to have been deposited in erosion troughs in the older sediments. The basin in which Martin Lake lies is formed by a syncline of Athabaska sediments interbedded with basalt flows, plunging gently a few degrees east of north.

3. The Eagle-Ace lakes district is north of Beaverlodge Lake and south of Mickey and Fredette lakes. It is underlain by Tazin rocks which include graywackes, basalts, and possibly some tuffs and basic sills. There is some of the older granite in the northern part of the district and most of the quartzite has been highly granitized. The rocks, in addition, have been severely crushed, folded, and squeezed on several occasions, so that they now appear as a series of tight, overturned folds striking northeasterly. The formations are cut by members of several sets of faults or "zones of movement" which also strike northeasterly.

4. The Donaldson Lake district is south of the lake of that name and between Ace Lake on the west and Raggs Lake on the east. The situation here is similar to that in the Eagle-Ace district except that there is more granite, granitization is more intense, and arkoses have been so altered as to be almost indistinguishable from granites and granitized sediments. Faults have been masked by later alteration, but that there was much movement is evidenced by the widespread development of amylonites. One fault, later than most of the alteration, is of great importance in the district. This is the St. Louis fault, which strikes N 65° E from Beaverlodge Lake, through Ace Lake to Raggs Lake, gradually swinging to the north, having a strike of about N 55° E at Christie Lake. It is a gravity fault dipping 45° to 65° to the south. Between Beaverlodge and Ace lakes down-faulted Athabaska sediments have been protected from erosion by the movement.

**Economic Geology:** Deposits of pitchblende of economic interest are both numerous and widely scattered in the Goldfields area. The Eldorado com-

pany has found well over a thousand radioactive anomalies on its claims within ten miles of Goldfields. The deposits studied in detail have many characteristics in common.

Pitchblende has been found in fractures in virtually all the rock types in the area and is younger than any known intrusion. While not confined to any type of rock, the veins are richer and better developed in basic rocks, including limestone and dolomite. The veins were deposited in open fissures at shallow to moderate depths. Characteristics of the fractures indicate that, for the most part, they owe their origin to tension. Carbonate, mainly calcite, is the common gangue mineral, and it usually has associated with it small quantities of hematite. With a few notable exceptions the only other mineral associates are very minor amounts of quartz and chalcopryrite. Except in some of the deposits along the shore of Lake Athabaska, the wall rocks of veins have been altered characteristically to a red cherty material, the result of the addition of feldspar and hematite.

**Cornwall Bay District:** Occurrences on Eldorado's Fish Hook Bay and Gil properties and on the Nicholson ground have been intensively explored. Pitchblende has been found on the Box gold property of the Consolidated Mining and Smelting Co., and anomalies are said to have been found by prospectors during 1948 on ground immediately north of Cornwall Bay.

1. On the Fish Hook Bay property almost 400 separate radioactive anomalies have been found, and pitchblende is visible in about 15 pct of these. The pitchblende-bearing veins exposed on surface are confined to dolomite, dolomitic iron formation and dolomitic quartzite of the Tazin series. They are in tension fractures striking subparallel to the topographically prominent Fish Hook valley. In one vein masses of botryoidal pitchblende are accompanied by massive arsenides of cobalt and nickel. At two other occurrences, masses of an unidentified vanadium mineral were found. None of the veins exposed on surface proved to be of sufficient extent to warrant further investigation at present. However, the larger portion of the 24,000 ft of diamond drilling carried out on the property was concentrated in Fish Hook valley. The bay and the muskeg were found to cover beds of quartzite and siliceous iron formation which have been severely crushed and then cemented by red hematite and calcite. The brecciated hanging-wall and footwall rocks of this westerly-dipping, northerly-striking crushed zone are locally well mineralized with pitchblende. It has been impossible at the present stage of development, to define clearly one or more distinct veins, or zones of mineralization, but further work may prove the pitchblende to be present in minable concentrations.

2. On the Nicholson property work has been done on only a few of the radioactive anomalies found. The geology of the individual occurrences is not well known to the writer, but most of the deposits exposed appear to be related to bodies of crushed and hematitic quartzite similar to but smaller than that at Fish Hook Bay, one half mile to the east. The deposit presently considered to be the most im-

portant lies in or near the contact between a crushed zone on the east and Tazin sediments on the west. The crushed zone underlies a valley parallel to the Fish Hook valley. When last seen by the writer the vein had been explored by trenching for a length of 500 ft and its presence detected by Geiger-Müller counter for a length of 1000 ft. It was later stated to have been trenched at intervals for a total length of about 1000 ft. It is pierced by two adits, vertically 60 ft and laterally 200 ft apart. Mineralogically the vein is unique among those in the Gold-fields area. The following remarks are quoted from a report dated October 6, 1944, written by R. Murphy.

The pitchblende occurrence is similar to that at No. 1 Vein at Eldorado, with a gangue of red-banded quartz, carbonate and platy hematite and the usual related nickel-cobalt minerals concentrated with the pitchblende in narrow seams in the ore. Textures in the ore are similar, with botryoidal and crustified structures exhibited by the sulfarsenides and gangue. The single divergent feature is the presence of thucholite at the Nicholson.

It is worthy of note that samples from this vein gave significant values in gold and platinum.

3. At the Box property fragments of pitchblende have been found in the overburden within about 2000 ft of the main shaft of the mine, but the occurrences have not been investigated as far as the writer knows. In addition, in 1946, members of a party of the Geological Survey of Canada found many anomalies in the outcrop of the ore body. These were associated with narrow fractures, a few feet long, in the granite. The walls of the fractures are stained a deep red. No mineralization was found.

4. On Eldorado's Gil property there are nine pitchblende-bearing veins in flat-lying gabbro sills intruding massive quartzite beds of the Beaverlodge series. The veins are deeply weathered and fresh material has not been exposed, but the veins are believed to conform to the general pattern outlined at the beginning of this section. While the veins are confined to the sills, surface evidence suggests that the fractures in which they were deposited persist from one sill to another, through the intervening quartzite. Surface evidence also suggests that there are two or more sills sufficiently thick to give the veins considerable promise. However, diamond drilling results were disappointing: the uppermost sill appears to have a maximum thickness of 50 ft and the lower sills are much thinner.

*Beaverlodge Lake District:* Occurrences of radioactive anomalies on five of Eldorado's properties near the shores of Martin and Beaverlodge lakes have been explored by surface operations, and one of them by underground workings. The pitchblende-bearing veins on these properties all lie in fractures in rocks of the Athabaska series. They conform closely with the general description of veins in the area. The fractures trend easterly to northeasterly. The properties are the Beaver, Westy, Murphy, Ra, and Ura. Of these the last two deserve separate discussion. On the other three the veins rarely exceed 50 ft in length, show values in pitchblende insufficient to render them individually of economic importance, and are not sufficiently closely grouped to enhance their possible value.

1. On the Martin Lake property (Ra), at the east shore of Martin Lake, a series of veins in a fracture zone several hundred feet long is being investigated from an adit. Exploration of the zone, first found on surface by methods to be described, has shown the possible existence of minable concentrations of pitchblende. The fracture zone is parallel in strike to the St. Louis fault, mentioned in the section on structure, and although it is not asserted that the fault may be traced this far west, it seems likely that the fracturing is related to it. Here, as elsewhere, the mineralization favors the basic rocks and is found almost exclusively in the basalt flows, only rarely in the sandstone beds.

2. On the Ura property, 2½ miles to the east of the Ra, on the shore of Beaverlodge Lake, a series of parallel fractures, some of which are 200 ft long, occur in a zone more than 2000 ft long. Diamond drilling 1400 ft in this zone gave inconclusive results, but it appears that the mineralization found on surface does not persist to a depth of 100 ft. It may not continue from the shallow Athabaska conglomerate, in which it occurs on surface, into the underlying Beaverlodge quartzite. Both the zone of fracturing and the individual fractures within it are parallel to the St. Louis fault. The occurrences are about 1000 ft south of the fault.

*Eagle Lake District:* The block of ground explored in this district is west of a north-south line through Eagle Lake, extending more than a mile both north and south of the lake and about a mile and a half to the west. Approximately 200 anomalies, a large proportion with visible pitchblende, have been found here. Many of the veins have been exposed by trenching and one large group of occurrences has been explored by diamond drilling.

1. The Ato showings are in the southern part of the district. They consist of several pitchblende-bearing veins in fairly flat-lying slaty sediments. They are believed to be in tension fractures related to inferred faults striking northeasterly. The fractures strike at a large angle to the strike of the faults. If the mineralization is confined, or almost confined, to basic rocks, as experience elsewhere suggests, its importance may be greatly restricted by the thickness of the slaty sediments.

2. The Eagle property in the center of the district has been explored by about 22,000 ft of diamond drilling. Here again the veins are found for the most part in the basic lavas and sediments and, where values are found in the granitized quartzite, they are usually within 10 ft of a contact with less acid rock. The formations here are more favorable to mine-finding than they are a mile to the south; the beds of basic rock are up to 100 ft thick and dip fairly steeply. The diamond drilling results suggest, however, that there are many short veins and only a few up to 300 ft in length.

3. The Mic showings are at the north end of the district. Deposits of pitchblende which appear important were found at the base of a scarp which marks a northeasterly-trending fault. The mineral is in a swarm of small irregular fractures within 40 ft of the fault. The strike of the fracture zones is about normal to the strike of the fault. The show-



ings have been stripped only, therefore no unaltered mineralization has been seen, but there does not appear to be much gangue accompanying the pitchblende.

**Ace-Donaldson Lakes District:** This subarea lies between Beaverlodge Lake on the west and Raggs Lake on the east. It is limited on the north by Donaldson Lake and extends a few thousand feet south of the St. Louis fault. Most of the occurrences lie either along the fault or within 2000 ft to the north or south of it. The fractures in which the veins are found are usually parallel or subparallel in strike to the fault. The deposits on the Ura property, discussed above, might be included in this group. Three other properties are discussed below.

1. On the Ace property the St. Louis fault dips  $48^{\circ}$  to the south and separates basic rocks of the Tazin series in the footwall from conglomerate and minor sandstone beds of the Athabaska series on the hanging wall. On surface some anomalies were found in the conglomerate but most promise was given by showings in the footwall rock. A number of short and narrow carbonate veins, rich in pitchblende, occur in tension fractures subparallel to the strike of the fault and within 100 ft of its surface trace. One important vein appears to dip at  $38^{\circ}$  towards the fault but the majority of the veins dip steeply in the same direction. The fault has now been pierced by 28 diamond drill holes, drilled from the hanging wall into the footwall. The core has yielded some high values in the  $U_3O_8$ , extending to a hole depth of about 100 ft below the fault. In some instances the high values are confined to streaks and stringers of pitchblende, whereas in others the radioactive mineral appears to be disseminated thinly throughout a body of intensely-altered, red rock. The pitchblende is in very small and irregular fractures of the tension type and is accompanied by red calcite.

2. The Bolger property is of interest for a reason additional to the probable value of its veins. Over an area of 6500 sq ft, the overburden, which is a few feet deep in most places, is rich in uranium in the form of the bright yellow and orange hydrous oxide. This rich overburden lies in a slight depression in the slope of a hill. It is suggested that rainwater collected in the shallow basin, leached uranium from underlying veins and deposited it throughout the overburden in the form of the oxide. Several trenches were dug at the locality but consolidated rock was reached at only a few places. The presence of pitchblende-bearing carbonate veins in fractures parallel to the strike of the St. Louis fault was indicated by this work.

3. The Donaldson Lake zone is about 10,000 ft long and up to 400 ft wide. The zone is subparallel to the strike of the St. Louis fault and about 1000 ft north of it. It extends southwesterly from a point 2000 ft east of Donaldson Lake. Here anomalous radioactivity is often found to emanate from the surfaces of large outcrops of rock, but the activity is usually localized in fractures trending parallel to the strike of the fault. The fractures occasionally hold coarse white carbonate and minor quantities of pitchblende. The original character of the host rock is masked by replacement by coarse red feldspar

in which there are sheared shreds of green chlorite. The interstices between grains of feldspar are filled by white carbonate. The walls of the fractures are usually a deeper red than the remainder of the rock. It is considered that the rock in a large shear zone has been replaced by the feldspar and minor carbonate and that these minerals were accompanied by pitchblende. The economic possibilities in this zone have yet to be explored, but there is some evidence that it contains one or more of the vein type of deposit as well as large, low-grade bodies.

**Other Occurrences:** In addition to the occurrences discussed above, many widely scattered anomalies have been found within 20 miles of Goldfields. Few of these have been thoroughly explored. The majority of them are in or near bands of basic rock and are associated with pronounced topographical features which trend northeasterly. The veins, some of which are pitchblende-bearing, have characteristics similar to those already described.

**Methods of Exploration: Introductory: the Geiger-Müller Counter:** Prospecting for and the exploration of radioactive mineral deposits differs from the same operations in connection with, say, gold deposits, chiefly in that work with the radioactive minerals is greatly facilitated by the use of the Geiger-Müller counter. The instruments used by the Eldorado company were developed by the National Research Council of Canada and adapted for several different purposes. The use of the counter has served to enlarge the visual sense of those employed in exploration. It will detect the presence of radioactive minerals where they cannot be seen by the unaided eye, even when the mineral is beneath a few feet of overburden or several inches of rock.

**Prospecting:** In the Goldfields area prospecting was carried out by experienced men who employed normal methods of prospecting but who were greatly aided by the use of counters. The model developed for the purpose is light, rugged, simple to operate, and requires little servicing. The tube is with the other parts of the instrument in a small box weighing about 7 lb. There are but two adjustments for the operator to make: he turns it on and switches it off, and increases the voltage as the batteries weaken. The instrument has no other signal but that heard through earphones. The batteries which provide the power have a life of six weeks to two months and are simple to change when worn out. The instrument is slung from a shoulder strap so that the tube is at waist height. For closer inspections the instrument is held a few inches above the ground. It is probable that few, if any, of the prospectors' many discoveries in the area would have been made without aid of the counter.

**Geophysical Surveying:** After a discovery has been made and claims have been staked, the next step in exploration is a detailed radioactivity survey of the staked ground. The survey consists of close examination of the ground with a portable counter especially adapted for the purpose. It is similar to the prospectors' model, but a little larger and 2 lb heavier. It is slightly more sensitive, having a 5 in. Geiger-Müller tube instead of a 4 in. one. A late model contains three synchronized 6 in.



tubes. Both models have microammeters by which roughly comparative readings may be taken.

The survey of a property entails traversing the ground at intervals of 15 ft. Although specimens of pitchblende 4 or 5 lb in weight lying on the ground may be detected by a counter operator at a distance of 10 ft when the instrument is carried at waist height, a vein under a few inches of soil may be undetected at a distance of 7 or 8 ft. Consequently re-surveys generally disclose additional deposits, as many as 50 pct of the number found in the original survey, even when both surveys are made with traverses at 15 ft intervals.

When the operator detects an anomaly by an increase of the rate of clicking in his earphones and confirms it by noting an appreciable increase on his milliammeter scale, he holds the instrument close to the ground surface and attempts to trace out the dimensions of the anomaly. Then he uses a light pick to uncover the source of the radioactivity, if possible. Finally, he locates the discovery relative to chainage points on picket lines cut on the property previous to the survey. The discovery is reported to a geologist who examines it to determine its possible importance. Exploration from this point proceeds according to usual practices.

Uranium minerals deposited in overburden occasionally give misleading results in surveys of this kind. Furthermore, it is very desirable to make such surveys before any blasting is done, as fragments of pitchblende scattered by blasting will "salt" a large area.

*Diamond Drilling: Special Aids:* The drilling of groups of deposits has taken place following strip-ping, trenching, and sampling in some instances. In other instances, however, drilling has followed immediately after the geophysical survey and geological examination, because it has not been anticipated that trenching of the deeply weathered carbonate veins would succeed in exposing fresh material at economic depths. Instead, the evidence given by the counter as to the value of the occurrences, together with the geological relationships, has been taken as sufficiently reliable.

Interpretation of drilling results has been greatly aided by adaptations of the Geiger-Müller counter for logging the core and for logging the holes.

The drill core is logged by two different forms of the counter. In the first method a sensitive, portable, battery-operated counter, normally used in the radioactivity surveys, is shielded by lead blocks 2 to 3 in. thick and arranged in such a way that a box of core may be passed close to the unshielded Geiger-Müller tubes. This method results in the detection of radioactivity in the core equivalent to that emanating from a concentration of as little as 0.05 pct  $U_3O_8$ , but is usually reliable no lower than 0.10 pct  $U_3O_8$ . Recently, more satisfactory results have been obtained from a line-operated instrument with a 6 in. tube in a probe at the end of a short cable. The probe is placed among the lead blocks much as the portable instrument was, and from there scans the core in the same manner. The line counter consistently detects concentrations of less than 0.03 pct  $U_3O_8$ .

The instrument employed to log drill holes is

powered by batteries. The tube is in a probe at the end of a cable several hundred feet long. The probe is dropped, or pushed, down the drill hole, and ammeter readings are taken at 5 ft intervals. The instrument is equipped with a loud speaker which instantly signals an increase in radioactivity. When this happens readings are taken at 1 ft or  $\frac{1}{2}$  ft intervals. After three years of operation sufficient data have been gathered to make it possible to relate scale readings to values obtained from the core. Where high readings have been obtained in the drill hole and no anomalous radioactivity has been found in the core it is possible to interpret the readings in terms of the probable concentrations of  $U_3O_8$  in the walls of the drill hole. The diameter of the drill core is thus enlarged, in effect, from  $1\frac{1}{8}$  in. to approximately 1 ft.

#### **Tobey-Albrecht Deposit—by W. F. James and A. H. Lang**

The Tobey-Albrecht property at Black Lake (fig. 1), 110 miles east of Goldfields, was discovered in 1948 by two prospectors working under the Saskatchewan government assistance plan. A concession  $2\frac{1}{2}$  by  $12\frac{1}{2}$  miles in size was granted to the discoverers, who sold it to Transcontinental Resources Limited, a company that also controls the Nicholson and Camray properties. Other nearby concessions were acquired by various individuals and companies.

The mineralization occurs in quartzite and schist mapped tentatively as part of the Tazin Group, of Archaean age. The mineralization occurs in a shear zone and in fractures, all lying almost immediately west of a prominent northeasterly-trending fault, called the Black Lake fault, that has been mapped for several miles.

The occurrence is described as follows by Dr. A. R. Byers,\* who examined it for the Saskatchewan government:

\* Unpublished report quoted by permission of Saskatchewan Department of Natural Resources.

At the time of the writer's visit, two areas of radioactivity approximately one mile apart along the strike of the fault zone, had been discovered.

The south showing consists of a mineralized shear, 18 to 24 inches wide which follows the contact between a foliated, fine grained amphibolite on the west and aphanitic, cherty sediments cut by aplitic and pegmatic stringers on the east. The shear strikes  $045^\circ$  and dips  $85^\circ$  southeast to vertical. The Geiger counter indicated a radioactive content of two times normal background for a length of 25 feet along the shear. This occurrence lies east of the main fault and cuts the sediments belonging to the Tazin group. It is the only occurrence of radioactive mineralization noted east of the fault and lying in these sediments.

The main or north showing, about 800 feet southwest of a large sandy point, is a zone of fractured gneiss approximately 2400 feet long and 200 to 500 feet wide. This zone lies just to the west of and parallels the strike of the main fault.

The majority of the fractures fall into three sets: (1) a set strikes and dips approximately parallel to the main fault and represents small subsidiary planes of shear. (2) a second group of fractures strikes  $065^\circ$  to  $080^\circ$  and dips  $75^\circ$  to  $85^\circ$  northwest. This also represents a direction of shear as indicated by slickensiding and sheared rock along the fractures. (3) a set of transverse fractures strikes  $115^\circ$  to  $125^\circ$  and dips  $75^\circ$



northeast to vertical. The fractures of this set have the characteristics of tension structures.

If the strain ellipsoid is orientated on the basis of the regional fault being a direction of major shear, then the composite fracture pattern formed by the above three sets of fractures agrees closely with the theoretical fracture pattern to be expected.

Individual fractures can be traced along the strike for distances of 50 to 200 feet, a few up to 500 feet. The fractures contain little vein material but are characterized by a reddish-brown to red discoloration of the country rock and a partial to complete loss of definition of the crystal texture of the gneiss. This zone of alteration extends out into the rock on either side of the fractures for a distance of a few inches to several feet.

The only metallic minerals noted are pyrite, cobaltite, and pitchblende. Pyrite is sparsely present in the wallrock along the margins of the fractures. Cobaltite was noted at only two places and appears to be very rare. Pitchblende occurs as small lenses and stringers, a few inches to several feet long and one-eighth to three inches wide, distributed at irregular intervals along some of the fractures.

Ten character samples taken by Dr. Byers assayed from 0.032 to 21.42 pct  $U_3O_8$  equivalent.

### The Camray Deposit

The Camray discovery is at the shore of Theano Point of Lake Superior, 70 miles north of Sault Ste. Marie (fig. 1). It is about  $1\frac{1}{2}$  miles from a highway that extends from Sault Ste. Marie to Montreal River.

The history of the discovery has already been discussed in the historical section of this paper. Because the find was made late in the season, no work has been done yet to expose the showings, but a road has been built to connect them with the highway, and it is expected that exploratory work will begin on this and other discoveries in the neighborhood as soon as the snow melts.

The deposit is described in some detail because it is the only pitchblende occurrence in eastern Canada for which much information is available.

**General Geology:** The region is in the southern part of the pre-Cambrian Shield and the Camray claims are underlain by pink granite and granite-gneiss probably Algoman in age. The rock is porphyritic in places, and the quartz in some specimens is so fine-grained that the rock resembles syenite to the unaided eye, a fact that probably explains LeConte's mention of syenite. The granite contains dykes and irregular masses of pegmatite.

The granite is intruded by diabase dykes, only one of which was studied. It consists of fine-grained, considerably altered diabase, which is much jointed and which tends to weather in blocks 2 to 6 in. in size. The diabase dykes of the region are classed as Keweenawan in age and are of two ages: an older, olivine-free type, and a younger, olivine-bearing type. Study of two thin-sections of specimens from this dyke failed to show the presence of olivine, therefore the dyke is classed tentatively with the older group.

**Mineralization:** The mineralization consists of quartz, pitchblende or coracite, carbonate, mica, and secondary uranium minerals. A sample representative of the 6 in. lens mentioned below, indicated a content equivalent to 8.72 pct  $U_3O_8$ . Coracite is listed

in early editions of Dana's "System of Mineralogy" as a variety of uraninite showing partial alteration to gummite. The seventh edition lists coracite as synonymous with gummite, and states that the name gummite is used in that edition as a generic or field term for substances essentially oxides of uranium. Satterly<sup>18</sup> reports that three specimens from the recent discovery were examined by E. W. Nuffield of the University of Toronto, who obtained X-ray diffraction patterns identical with those obtained from pitchblende from Great Bear Lake. Nuffield and Satterly, therefore, class the Camray mineral as pitchblende. Unless further mineralogical or analytical studies indicate that the name pitchblende is not applicable, it will probably be best to use that term and to discontinue use of the name coracite. The mineral is called pitchblende provisionally in this paper.

**Description of the Deposit:** The discovery consists of zones of pitchblende-bearing stringers that fill fissures in the granite walls immediately adjoining the diabase dyke described above. Because the erosion of the dyke has left a canyon, the stringers are exposed on almost vertical cliffs. The upper parts of the cliffs, which are about 50 ft high, cannot be examined at present, nor can the extent of the stringers into the walls be ascertained. The dyke and canyon strike  $N 80^\circ W$  and have an average width of 35 ft, but at one place the dyke narrows to 18 ft and the remaining width of the canyon is floored by granite. The dyke dips  $70^\circ$  to  $80^\circ$  northward.

Deepening of the canyon at the shore of the lake has caused the formation of a small cove, where the water normally extends inland about 140 ft from the general shoreline of Theano Point. The high-water mark is about 200 ft farther inland. Eastward from the cove the floor of the canyon rises gradually for about 500 ft, where it is perhaps 50 ft above lake-level; beyond this point the floor rises steeply. The dyke outcrops intermittently along the flatter part of the floor, and probably continues much farther inland but is covered by overburden.

The greatest amount of pitchblende is exposed in a lens up to 6 in. wide, which tapers to a thin stringer within a length of about 3 ft. Elsewhere, the stringers range from  $\frac{1}{8}$  to 2 in. in width, most of them being  $\frac{1}{2}$  in. or less. Usually they are only exposed for lengths of 2, 3, and 4 ft. Many of the stringers strike  $N 70^\circ E$  to  $N 85^\circ E$ , and dips range from  $35^\circ SE$  to vertical. Because the pitchblende is brittle and easily eroded, often the fractures are now unfilled for a depth of  $\frac{1}{2}$  in. or more from the face of the cliff; there pitchblende cannot be seen but high counter readings indicate the likelihood of its presence.

The stringers are grouped in two main zones along the footwall or south side of the canyon, the one nearer the shore having an overall length of 275 ft, and the other 100 ft. These are separated by an apparently barren zone 200 ft long. At the east end of the shoreward zone of stringers a small fault appears to displace the dyke about 2 ft. This fault may have some bearing on the narrowing of the dyke nearby, but the relationships are not yet well exposed. No pitchblende was seen in the fault,

but high counter readings indicate that it may be present. This is the only indication that stringers may occur in the dyke.

About 400 ft east of the more easterly zone of stringers three, high counter readings were obtained within a length of 80 ft along a granite cliff that appears to be the continuation of the footwall. No pitchblende was seen, but development work here might disclose another zone of stringers.

On the north or hanging-wall side of the canyon two stringers  $\frac{1}{8}$  to  $\frac{1}{4}$  in. wide and 4 ft long are exposed at a locality about 130 ft east of the head of the cove. One of these is perpendicular to the dyke and the other is parallel with the contact between the dyke and the granite.

Mr. Campbell stated that a high count is obtainable in the bottom of another gorge, about  $\frac{1}{3}$  mile east of Theano Point, that may represent the weathering of a dyke oblique to the direction of the one described above. It would be interesting to trace the continuation of the main dyke to its intersection with this gorge, on the chance that pitchblende might be concentrated there. Snow prevented the writer from checking the reading at this locality.

**Grade:** The deposit is not yet exposed in such a way as to permit sampling of a type that would give any indication of the grade of a ton of the stringer-containing granite. To cut samples along the cliff faces would not be practical for two reasons: because of the weathering of the stringers, and because such samples would be parallel with, rather than across, the probable direction of mining.

The field Geiger counter cannot be used in an absolute, quantitative way for estimating the grade of a deposit such as this, because the needle goes to the limit of the scale when held against the larger stringers, and because, when held against rock a foot or two away from a stringer, the reading represents rays coming from the stringer rather than from the area of contact.

Samples of the stringer material itself indicate the character of the mineralization, but not minable grade, consequently the results of such samples are included in the mineralogical section of this report.

Samples of the granite near the stringers, when tested with a quantitative, laboratory counter, indicated an average radiation equivalent to 0.006 pct  $U_3O_8$ , and specimens of the dyke rock gave an average of less than 0.001 pct  $U_3O_8$ . As counts of this order are commonly obtained from rocks, it does not seem likely that the rock between the stringers will prove valuable. If the deposit proves to be minable, it will probably be necessary to sort the stringer material from the country rock.

**Origin of the Deposit:** The pitchblende probably was deposited from hydrothermal solutions of magmatic origin. The relationship to the dyke seems to be entirely structural. The mineralization may have come from the same magma as the dyke, but the suggestion of pitchblende occurring in a fault displacing the dyke, as mentioned above, points to a younger age for the mineral deposit. It would be of interest to have an age determination made on a sample of the pitchblende, but sufficient unaltered mineral is not yet available.

The relationship to the dyke makes the walls of this and other dykes in the vicinity the most obvious place for further prospecting but, presumably, any fractures, faults, or shear zones that were open at the time of mineralization could contain pitchblende, irrespective of the presence of dykes.

**Conclusions:** Much work will be required to prove the merits of the property, but the discovery is of great importance because it establishes the occurrence of pitchblende in the eastern part of the pre-Cambrian Shield, at a time when the report of the "coracite" discovery had become almost legendary; and because analogous geological conditions occur over a large part of the district, hence other and perhaps more important discoveries may be made.

### Hydrothermal Deposits in British Columbia

Three uranium deposits of the hydrothermal type were discovered in British Columbia in 1948 by testing properties known to contain considerable quantities of cobalt minerals. The discoveries were made at the Gem and the nearby Jewel property, in Bridge River district,<sup>17</sup> and at the Hazelton View property near Hazelton.<sup>18</sup> They show striking similarities: all consist of gold-bearing arsenopyrite associated with cobalt minerals, occurring in granodiorite, all had previously received underground exploration as gold properties, and they contain the largest amounts of cobalt minerals known in the province.<sup>19</sup> The economic possibilities of the discoveries are not yet known.

### "Giant Quartz Vein" Deposits

The so-called "giant quartz veins" that are conspicuous features of the part of the pre-Cambrian Shield between Great Bear and Great Slave lakes are large quartz stockworks consisting of networks of quartz stringers with the intervening rock commonly replaced by silica. Many of them occur in northeasterly-trending faults, are several hundred feet wide, and are traceable for several miles. Many of these "veins" have been reopened by fracturing, and mineralized by later quartz, hematite, and small amounts of pitchblende. In some of these deposits the pitchblende is too sparse or too scattered to be of interest, but others are being developed in the hope that minable ore bodies may be outlined. If these efforts are successful, numerous other giant quartz veins that may be prospected will be found indicated on geological maps.

A large giant quartz vein lying between Hottah and Beaverlodge lakes, 200 miles north of Yellowknife, has been known for several years to contain scattered pods of hematite and pitchblende. Part of it was acquired recently by Gold-Uranium Mines Limited, who did considerable diamond drilling.

Another deposit of this type, known as the Giauque, lies 97 miles northwest of Yellowknife. When examined in 1948, trenches blasted at intervals into the rock had exposed many small fractures mineralized with hematite and pitchblende. These fractures formed a general zone perhaps 25 ft wide, extending down the center of the vein for a length of 2200 ft. Other discoveries were reported to have been made later in the year. Fairly high uranium assays can be obtained from the material in the fractures,



but these are so erratically distributed that samples taken across significant widths are low. Much systematic sampling will be required to outline the better mineralized zones and to estimate their average grade. If a satisfactory treatment process should be available, it is possible that a deposit such as this might be workable as a large, low-grade operation now or at some future time.

### Sedimentary Deposits

Sedimentary strata of different ages, containing relatively small amounts of uranium, are known to occur in several parts of Canada, but most of them have not as yet been investigated in detail because of the much greater interest taken in pitchblende deposits. A small amount of uranium has recently been reported by T. L. Tanton to occur in nodules in the Sibley Series of the pre-Cambrian Shield in Ontario.<sup>20</sup>

The sedimentary deposit that has received the greatest attention in Canada is known as the McLean Bay occurrence, at the southeastern shore of Great Slave Lake. Here the Eldorado company has done considerable trenching, diamond drilling, and sampling. The claims are underlain by steeply-dipping beds of Proterozoic dolomite and quartzite. A zone of dolomite that displays concentric structure of possible algal origin is stained with hematite and carries low values in thorium and uranium, the ratio of the two elements being about 6 to 1. Efforts have been made to determine the thorium and uranium-bearing minerals but they have not yet been successful.

The deposit is not regarded as favorably as most Canadian pitchblende occurrences.

### Secondary Deposits

Canadian occurrences of secondary uranium minerals consist of relatively small quantities of such minerals in the near-surface parts of primary uranium deposits; as coatings on the walls of mine workings and on rock in mine dumps; and, occasionally, in gravel or sand near outcrops of primary deposits, from which uranium was leached by surface water and deposited in nearby overburden. Commercial deposits of secondary uranium minerals are not thought likely to occur in Canada. The bright colors of these minerals are useful in attracting attention to the presence of primary deposits.

### Thorium Deposits

Thorium deposits of present commercial importance are not known to occur in Canada, but several discoveries, of different types, have been made.

Numerous pegmatite deposits in different parts of the country are known to contain minor amounts of thorium-bearing minerals.<sup>21</sup>

Veins consisting of carbonate, quartz, and hematite, of the hydrothermal type, and containing appreciable quantities of thorium in a form as yet unidentified, were discovered in 1948 at the northern edge of the pre-Cambrian Shield near the railway that extends from Cochrane, Ontario, to Hudson Bay.<sup>22</sup> Some geologists who have examined these discoveries believe that the age of the mineralization may be later than Middle Devonian.

The occurrence of considerable thorium in pre-

Cambrian dolomite at Great Slave Lake has already been mentioned in the section of this paper dealing with sedimentary uranium deposits.

Several samples of placer concentrates from British Columbia and the Northwest Territories have proved radioactive. Whenever exhaustive tests were made the radioactivity was shown to be due to the presence of thorium, not uranium. This is to be expected, because of the greater stability of thorium minerals. The thorium minerals concerned were either known or suspected to be monazite. The only thorium-bearing placer as yet examined in detail proved to be too small to be of present economic interest.

### References

- <sup>1</sup> Richard Murphy: Geology and Mineralogy at Eldorado Mine. *Trans. Can. Inst. Min. and Met.* (1946) 49, 426-435.
- <sup>2</sup> J. L. LeConte: On Coracite, a new Ore of Uranium. *Amer. Jnl. Sci.* (1847) 3, 117, 173-175.
- <sup>3</sup> G. L. Nute: Lake Superior. New York, 1944. Bobbs-Merrill Co.
- <sup>4</sup> A. H. Chester: A Dictionary of the Names of Minerals. New York. 1896. Wiley and Sons.
- <sup>5</sup> J. D. Whitney: Chemical Examination of Some Minerals. *Amer. Jnl. Sci.* (1849) 7, 434.
- <sup>6</sup> F. A. Genth: Dr. Genth's Contributions to Mineralogy. *Amer. Jnl. Sci.* (1857) 23, 421.
- <sup>7</sup> W. E. Logan: Geology of Canada. Geol. Survey Canada. *Rept. of Progress* (1863) 504, 702.
- <sup>8</sup> H. V. Ellsworth: Nickeliferous and Uraniferous Anthraxolite from Port Arthur, Ontario. *Amer. Mineralogist* (1934) 19, (9) 426.
- <sup>9</sup> J. Obalski: On a Mineral Containing "Radium" in the Province of Quebec. *Jnl. Can. Min. Inst.* (1904) 7, 245.
- <sup>10</sup> S. Brunton: Investigation of the Occurrence of Radioactive Minerals in Ontario. Geol. Survey Canada. *Summary Rept.* (1914) p. 91.
- <sup>11</sup> S. E. Wolfe and N. Hogg: Some Radioactive Mineral Occurrences in Cardiff and Monmouth Townships, Haliburton County, Ontario. Ont. Dept. of Mines, Press Release 8 (1948).
- <sup>12</sup> D. F. Kidd: Great Bear Lake Area. Geol. Survey Canada. *Summary Rept.* (1932) Part C, pp. 1-36.
- <sup>13</sup> J. F. Henderson: Extent of Proterozoic Granitic Intrusions in the Western Part of the Canadian Shield. *Trans. Royal Society of Canada, 3rd. Series.* (1948) 42, Sec. 4, 41-54.
- <sup>14</sup> D. F. Kidd and M. H. Haycock: Mineragraphy of the Ores of Great Bear Lake. *Bull. Geol. Soc. Amer.* (1935) 46, 879-960, 926.
- <sup>15</sup> C. S. Lord: Mineral Industry of the Northwest Territories. Geol. Survey Canada. *Memoir* 230 (1941) 50.
- <sup>16</sup> J. Satterly and D. F. Hewitt: Report on a Pitchblende Occurrence at Theano Point, Lake Superior, Ontario. Ont. Dept. of Mines. Press Release 9 (1948).
- <sup>17</sup> J. S. Stevenson: Radioactive Investigations at Gun Creek. *Western Miner.* (1948) 21, (10) 86-88.
- <sup>18</sup> E. D. Kindle: Mineral Resources, Hazelton and Smithers Areas. Geol. Survey Canada. *Memoir* 223 (1940) 44.
- <sup>19</sup> A. H. Lang: Notes on Prospecting for Uranium in Canada. Geol. Survey Canada. *Paper* 49-4 (1949) p. 6, 16.
- <sup>20</sup> T. L. Tanton: Radioactive Nodules in Sediments of the Sibley Series, Nipigon, Ontario. *Trans. Royal Society Canada, 3rd Series.* (1948) 42, Sec. 4, 69-75.
- <sup>21</sup> H. V. Ellsworth: Rare-element Minerals of Canada. Geol. Survey Canada. *Econ. Geol. Series No. 11* (1932).
- <sup>22</sup> N. Hogg: Ont. Dept. of Mines. Press Release 7 (1948).

## Coal Cleaning Equipment

by W. W. Anderson

A formula for quantitative efficiency is proposed, in which the efficiency value is a function of the improperly distributed material at the gravity of separation effected by the cleaning equipment. The misplaced material is expressed as a percentage of the total material discharged from the equipment. Numerous examples are given, demonstrating application of the formula to various types of separation, including multiple-product separations.

WEBSTER'S dictionary gives the following definition for "efficiency": "Effective operation as measured by a comparison of actual and possible results."

Engineers think of this definition in terms of the ratio of output to input, and since they learn early in their careers that output cannot be greater than input, they usually consider a machine to be 100 pct efficient if it operates perfectly, and to be 0 pct efficient if it performs no function at all. Hence, values of efficiency between 0 and 100 pct reflect how well or how poorly a machine has performed; and any method of calculation for determining efficiency that does not show 0 pct when no function has been performed, and 100 pct when a perfect job has been done, is looked upon with skepticism.

W. W. ANDERSON, Member AIME, is Technical Director, Western Division, Commercial Testing & Engineering Co., Chicago, Ill.

AIME New York Meeting, February 1950.

TP 2786 F. Discussion (2 copies) may be sent to Transactions AIME before March 30, 1950. Manuscript received Oct. 5, 1949, revision received Nov. 25, 1949.

The literature on performance of coal cleaning equipment contains many references to efficiency of performance, but there is no unanimous agreement among the authors as to how the efficiency should be determined. Numerous methods of calculation have been proposed from time to time, most of which attempt to fulfill the definition of efficiency by comparing output to input, using float and sink data or some other characteristic, such as dry ash content, as a measure. However, to the best of our knowledge, none of the methods advocated in the past meets the requirements of a practical engineer in judging cleaning plant performance, because the formulas previously advocated do not confine numerical values of efficiency between 0 and 100 pct, and the literature does not contain an explanation of the meaning of the formulas under various conditions of operation.

Therefore, the following formula, based on Yancey's concept of misplaced material,<sup>1</sup> is proposed as a measure of efficiency which will conform

with the requirement that the values of efficiency must be confined between 0 and 100 pct:

Quantitative Efficiency =

$$100.0 - (\text{pct wt sink}^* \text{ in cleaned coal} + \text{pct wt float}^* \text{ in refuse})^\dagger$$

\* At gravity of separation effected by cleaning equipment.

† Total product basis of the size of material under study.

It will be noted from this formula that an actual separation between cleaned coal and refuse must be effected by the cleaning equipment before the formula can be used; also, that improperly distributed material in the products must be determined at the gravity of separation, rather than at some arbitrary test gravity.

It will further be noted from this formula that the weight of cleaned coal recovered and the weight of refuse rejected by the equipment have their proper influence, because the sink in the cleaned coal and the float in the refuse are expressed in the equation as a per cent weight of the total products of a definite size discharged from the plant.

This formula has one other important characteristic, in that it requires no knowledge of the feed to the cleaning unit. Comparison of data on the products from a cleaning unit with the feed to the unit can lead to many worthwhile conclusions, but not in regard to the efficiency of separation effected by the equipment. The reason for this is that mechanical fracture of many pieces of the feed occurs during the cleaning process; and in some processes crushers are deliberately incorporated in the flow to produce size degradation to free impurities from cleaned coal, thereby altering the washability characteristics of the material while in process.

When such size degradation occurs in a cleaning process, it is quite possible that this crushing will result in a reduction in ash content in the finer sizes of cleaned coal when compared with the raw coal feed, even though no true separation has been effected. This is due to the fact that the finer sizes have been "sweetened" by the crushing of cleaned coal from some coarser size. Many formulas advocated in the past show high cleaning efficiencies



when a condition such as this occurs, erroneously crediting the effects of crushing to cleaning efficiency.

It is our opinion that size degradation occurs in the handling of all materials through a cleaning plant, and for this reason efficiency values must be calculated in terms of the material discharged from the plant, rather than in terms of feed to the plant. The formula proposed herein is based on this belief.

The concept of efficiency can best be understood by examination of a few sets of data. There are essentially three types, as follows: (1) no true separation and no efficiency, (2) true separation at 100 pct efficiency, and (3) true separation at an efficiency less than 100 pct.

**Type 1, No True Separation and No Efficiency:** If for some reason or other a material should pass through a cleaning unit and be divided into a cleaned coal product and a refuse product, each having the same gravity consist after discharge from the machine, a set of data would result such as is shown in table I.

Table I. Specific Gravity Consist of Products, Type 1 Separation

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	55.0	55.0
1.30	1.40	8.0	8.0
1.40	1.50	3.0	3.0
1.50	1.60	2.0	2.0
1.60	1.70	1.5	1.5
1.70	1.80	1.5	1.5
1.80	2.00	2.0	2.0
2.00		27.0	27.0
		100.0	100.0

In this case there has been a division of the feed into two products, but it is obvious that no true separation in accordance with specific gravity has been effected. Therefore, there can be no efficiency of separation.

**Type 2, True Separation at 100 pct Efficiency:** The opposite extreme from no separation at all is perfect separation, such as is exemplified in table II.

Table II. Specific Gravity Consist of Products, Type 2 Separation

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	0.0	0.0
1.30	1.40	5.0	0.0
1.40	1.50	4.0	0.0
1.50	1.60	1.0	0.0
1.60	1.70	0.0	1.0
1.70	1.80	0.0	4.0
1.80	2.00	0.0	5.0
2.00		0.0	90.0
		100.0	100.0

The foregoing figures demonstrate an effective separation at a specific gravity of 1.60 and at an efficiency of 100 pct, because there is no sink material in the cleaned coal at a gravity greater than 1.60, and there is no float in the refuse at a gravity less than 1.60.

**Type 3, True Separation at an Efficiency Less Than 100 pct:** Types 1 and 2 separations cover extreme conditions, and the simple sets of data presented to exemplify them require no extensive calculations to determine their meaning. However, the significance of data exemplifying a Type 3 separation is not so obvious, despite the fact that simple sets of data can be presented of Type 3 separations. Therefore, the several sets of data presented hereinafter of Type 3 separations must be examined in greater detail; but prior to this examination it seems advisable to point out that Type 3 separations are of two kinds: (a) efficiencies above 50 pct, wherein the cleaned coal contains more float than the refuse; and (b) efficiencies below 50 pct, wherein the refuse contains more float than the cleaned coal.

These two kinds of Type 3 separation will now be considered.

(a) *Efficiencies Above 50 pct:* The values in table III are typical of a very high order of efficiency, approaching perfect separation.

Table III. Specific Gravity Consist of Products, High Order of Type 3 (a) Separation

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	90.0	0.0
1.30	1.40	4.0	0.0
1.40	1.50	3.0	0.0
1.50	1.60	2.0	1.0
1.60	1.70	1.0	2.0
1.70	1.80	0.0	3.0
1.80	2.00	0.0	4.0
2.00		0.0	90.0
		100.0	100.0

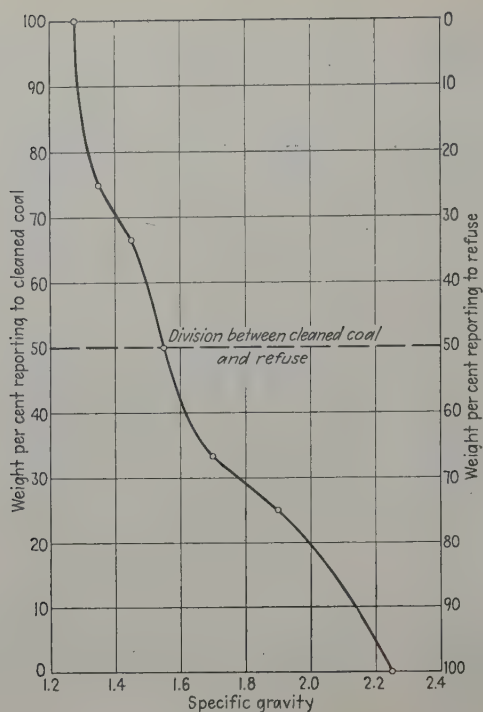
It is apparent from these gravity consists that a true separation has occurred between the cleaned coal and the refuse products, and that the separation has been effected in the gravity range of 1.50 to 1.70. It is also apparent that the separation has not been perfect, because there has been an overlapping of material in the gravities between 1.50 and 1.70. But the exact gravity of separation and the efficiency of separation are not evident.

However, if we assume that the total products are divided equally between cleaned coal and refuse, it will be seen that the gravity of separation occurred at 1.60. Then, the quantitative efficiency of separation at this gravity can be determined to be 99 pct, as follows:

	Pct Weight Misplaced
Cleaned Coal = 50 pct of total products, and contains 1 pct sink at 1.60 gravity	0.50
Refuse = 50 pct of total products, and contains 1 pct float at 1.60 gravity	0.50
Total misplaced	1.00

Quantitative efficiency = 100 — 1 = 99 pct

In contrast to a nearly perfect separation, there is the type of separation which is so poor that the



**Fig. 1—Distribution of total weight between cleaned coal and refuse.**

efficiency is of a low order, approaching 50 pct. For example, consider the figures in table IV.

**Table IV. Specific Gravity Consist of Products, Low Order of Type 3 (a) Separation**

Cleaned coal = 50 pct of combined products Refuse = 50 pct of combined products			
Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	1.0	0.0
1.30	1.40	6.0	2.0
1.40	1.50	10.0	5.0
1.50	1.60	76.0	76.0
1.60	1.80	5.0	10.0
1.80	2.00	2.0	6.0
2.00		0.0	1.0
		100.0	100.0

The weight distribution in the various gravity fractions of the cleaned coal and refuse is shown in table V.

**Table V. Distribution of Material, Low Order of Type 3 (a) Separation**

Specific Gravity		Pct Wt of Fraction	
Sink	Float	Cleaned Coal	Refuse
	1.30	100.0	0.0
1.30	1.40	75.0	25.0
1.40	1.50	66.7	33.3
1.50	1.60	50.0	50.0
1.60	1.80	33.3	66.7
1.80	2.00	25.0	75.0
2.00		0.0	100.0

These values indicate that an actual separation has been made between the cleaned coal and refuse, and one might suspect that this separation occurred

in the gravity range of 1.50 to 1.60, although the exact gravity of separation may not be clear to everyone. Therefore, the values of the weight distribution between cleaned coal and refuse have been plotted in fig. 1.

Examination of this curve of separation shows that the curve crosses the 50 pct weight line at a specific gravity of 1.55. This means that all of the material with a specific gravity of 1.55 has been divided equally between cleaned coal and refuse. The material having a specific gravity less than 1.55 has reported in preponderant quantities to the cleaned coal product; whereas the material with a specific gravity greater than 1.55 has reported in preponderant amounts to the refuse product.

This brief explanation of the gravity of separation is given to demonstrate the correct method for determination of the gravity at which the material was actually separated into a cleaned coal and a refuse product. This method is explained elsewhere in greater detail.<sup>1</sup>

Based on the gravity of separation of 1.55, it will be seen from the data that there was 45 pct sink in the cleaned coal, and that there was also 45 pct float in the refuse. Hence, the quantitative efficiency of separation at a gravity of 1.55 can be determined to be 55 pct, as follows:

	Pct Weight Misplaced
Cleaned Coal = 50 pct of total products, and contains 45 pct sink at 1.55 gravity	22.5
Refuse = 50 pct of total products, and contains 45 pct float at 1.55 gravity	22.5
Total misplaced	45.0

Quantitative efficiency =  $100 - 45 = 55$  pct

(b) *Efficiencies Below 50 pct:* If, for some ridiculous reason, a cleaning process should deliver a



refuse product containing more float than the cleaned coal product, a true separation, nevertheless, could be effected by the cleaning equipment, but the efficiencies of separation would be less than 50 pct. For instance, if the gravity consists of the cleaned coal and refuse products in the previous set of data were switched, we would have a condition in which the refuse would have more float than the cleaned coal product, and the separation would be indicated by the distribution of material shown in table VI.

**Table VI. Specific Gravity Consist of Products, High Order of Type 3 (b) Separation**

Cleaned coal = 50 pct of combined products Refuse = 50 pct of combined products			
Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
1.30	1.30	0.0	1.0
1.40	1.40	2.0	6.0
1.40	1.50	5.0	10.0
1.50	1.60	76.0	76.0
1.60	1.80	10.0	5.0
1.80	2.00	6.0	2.0
2.00		1.0	0.0
		100.0	100.0

The weight distribution in the various gravity fractions of the cleaned coal and refuse is shown in table VII.

**Table VII. Distribution of Material, High Order of Type 3 (b) Separation**

Specific Gravity		Pct Wt of Fraction	
Sink	Float	Cleaned Coal	Refuse
	1.30	0.0	100.0
1.30	1.40	25.0	75.0
1.40	1.50	33.3	66.7
1.50	1.60	50.0	50.0
1.60	1.80	66.7	33.3
1.80	2.00	75.0	25.0
2.00		100.0	0.0

Again, these values demonstrate that a separation was made in the gravity range between 1.50 and 1.60; and if the values were plotted in a similar fashion to fig. 1, the gravity of separation effected would be found to be 1.55. In this case there was 55 pct sink in the cleaned coal and 55 pct float in the refuse at a specific gravity of 1.55; and the efficiency of separation can be calculated to be 45 pct, as follows:

		Pct Weight Misplaced
Cleaned Coal	= 50 pct of total products, and contains 55 pct sink at 1.55 gravity	27.5
Refuse	= 50 pct of total products, and contains 55 pct float at 1.55 gravity	27.5
	Total misplaced	55.0
	Quantitative efficiency = $100 - 55 = 45$ pct	

Furthermore, an extreme example of an efficiency of separation of less than 50 pct would be possible, in which the quantitative efficiency would approach zero. For instance, consider the following conditions:

		Pct Weight Misplaced
Cleaned Coal	= 99 pct of total products, and contains 99 pct sink	98.01
Refuse	= 1 pct of total products, and contains 99 pct float	0.99
	Total misplaced	99.00

Quantitative efficiency =  $100 - 99 = 1$  pct

The foregoing examples demonstrate very simply every type of separation that can be encountered when dealing with separating equipment. However, in order to emphasize these basic types of separation, it is thought advisable to restate them, as follows:

- Type 1. No true separation at zero efficiency.
- Type 2. Perfect separation at 100 pct efficiency.
- Type 3. True separation at an efficiency between zero and 100 pct.
  - (a) Efficiencies above 50 pct, wherein the cleaned coal contains more float than the refuse.
  - (b) Efficiencies below 50 pct, wherein the refuse contains more float than the cleaned coal.

Furthermore, if a machine makes a multiple separation, rather than a two-product separation, the efficiency of separation can be determined for each of the multiple separations and for the machine as a unit. Such values have significance for an engineer, because they are comparable from product to product within the machine, and can also be compared with data on other types of separating equipment working under similar or different conditions.

Again, examination of a few sets of data seems to offer the best means of explanation of the procedure for using the formula in connection with problems of multiple separation. First, in table VIII, a simple three-product separation will be considered, in which the efficiency is of a high order.

**Table VIII. Specific Gravity Consist of Products, Three-product Separation**

Specific Gravity		Cleaned Coal	Middlings	Refuse
Sink	Float	Pct Wt	Pct Wt	Pct Wt
	1.30	85.0	0.0	0.0
1.30	1.40	10.0	0.0	0.0
1.40	1.50	4.0	1.0	0.0
1.50	1.60	1.0	4.0	0.0
1.60	1.70	0.0	90.0	0.0
1.70	1.80	0.0	4.0	1.0
1.80	1.90	0.0	1.0	4.0
1.90	2.00	0.0	0.0	10.0
2.00		0.0	0.0	85.0
		100.0	100.0	100.0

It will be seen from the above set of figures that an actual separation occurred between the cleaned

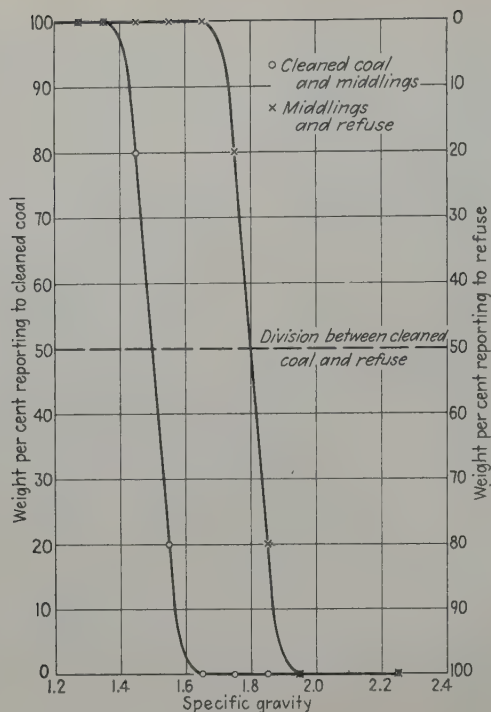


Fig. 2 — Distribution of total weight between products, three-product separation.

coal and the middlings, as well as between the middlings and the refuse, but that there was an overlapping of material in the gravity range of 1.40 to 1.60 in the cleaned coal and middlings, and another overlapping of material in the gravity range of 1.70 to 1.90 in the middlings and refuse. Hence, we know that the separation between cleaned coal and middlings occurred at some gravity between 1.40 and 1.60, and we also know that the separation between middlings and refuse occurred at some gravity between 1.70 and 1.90.

However, in each case the gravity of separation must be determined before the efficiency of separation can be determined. Therefore, in order to make the problem a simple one, it can be assumed, for purposes of explanation, that the total products are divided in equal proportion between cleaned coal, middlings, and refuse. On this basis, the weight distribution between cleaned coal and middlings is calculated to be as shown in table IX.

Table IX. Distribution of Material between Cleaned Coal and Middlings, Three-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal	Middlings
1.30	1.30	100.0	0.0
1.40	1.40	100.0	0.0
1.50	1.50	80.0	20.0
1.60	1.60	20.0	80.0
1.70	1.70	0.0	100.0
1.80	1.80	0.0	100.0
1.90	1.90	0.0	100.0
2.00	2.00	0.0	100.0

In similar fashion, the weight distribution between middlings and refuse can be calculated, obtaining the values shown in table X.

Both sets of values of distribution can be plotted on one chart, as shown in fig. 2, and it will be seen

Table X. Distribution of Material between Middlings and Refuse, Three-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Middlings	Refuse
1.30	1.30	100.0	0.0
1.40	1.40	100.0	0.0
1.50	1.50	100.0	0.0
1.60	1.60	100.0	0.0
1.70	1.70	100.0	0.0
1.80	1.80	80.0	20.0
1.90	1.90	20.0	80.0
2.00	2.00	0.0	100.0
		0.0	100.0

that the gravity of separation between cleaned coal and middlings was at 1.50, and the point of separation between middlings and refuse was at 1.80.

At a specific gravity of 1.50 there was 1 pct sink in the cleaned coal and 1 pct float in the middlings; and at a specific gravity of 1.80 there was 1 pct sink in the middlings and 1 pct float in the refuse. Thus, the efficiencies of separation can be calculated as follows:

		Pct of Total Products Misplaced
Cleaned coal	= One-third of total products and contains 1 pct sink at 1.50 gravity	0.33
Middlings	= One-third of total products and contains 1 pct float at 1.50 gravity	0.33
Refuse	= One-third of total products and contains no float at 1.50 gravity	0.00
Total misplaced		0.66



Therefore, the quantitative efficiency of separation between cleaned coal and middlings =  $100 - 0.66 = 99 \frac{1}{3}$  pct.

Similarly,

		Pct of Total Products Misplaced
Cleaned coal	= One-third of total products and contains no sink at 1.80 gravity	0.00
Middlings	= One-third of total products and contains 1 pct sink at 1.80 gravity	0.33
Refuse	= One-third of total products and contains 1 pct float at 1.80 gravity	0.33
Total misplaced		0.66

Therefore, the quantitative efficiency of separation between middlings and refuse =  $100 - 0.66 = 99 \frac{1}{3}$  pct.

Thus, we find that the efficiency of separation between the cleaned coal and the middlings, and between the middlings and the refuse in each case was  $99 \frac{1}{3}$  pct.

The foregoing example of a three-product separation is an extremely simple one because there was no overlapping of material in more than two products. Therefore, another example of multiple separation should be considered, in which overlapping of material occurs in more than two of the products. The figures in table XI show such a condition.

**Table XI. Specific Gravity Consist of Products, Four-product Separation**

Specific Gravity		25 Pct	25 Pct	25 Pct	25 Pct
Sink	Float	Cleaned Coal	No. 1 Middlings	No. 2 Middlings	Refuse
1.30	1.30	60.0	1.0	0.0	0.0
1.40	1.40	25.0	4.0	0.0	0.0
1.50	1.50	10.0	10.0	1.0	0.0
1.60	1.60	4.0	70.0	4.0	0.0
1.70	1.70	1.0	10.0	10.0	1.0
1.80	1.80	0.0	4.0	70.0	4.0
1.90	1.90	0.0	1.0	10.0	10.0
2.00	2.00	0.0	0.0	4.0	25.0
		0.0	0.0	1.0	60.0
		100.0	100.0	100.0	100.0

In the foregoing table of gravity consists an overlapping of material occurs in three of the four products in the gravity range between 1.40 and 1.90. When such a condition exists, it is not possible to determine the correct gravity of separation by examination of only two of the products, and in such case all products must be considered. For instance, in order to determine the gravity of separation between the cleaned coal and the No. 1 middlings, it is necessary to calculate the weight distribution of the material in all of the gravity fractions as it occurs in the cleaned coal in contrast to the combined value of the No. 1 middlings, the No. 2 middlings, and the refuse.

In other words, for purposes of analysis of the float and sink results in terms of the gravity of separation at which the cleaned coal was separated from the remainder of the material, the No. 1 middlings, the No. 2 middlings, and the refuse must be considered as one product, just as though only a two-product separation had been made.

Likewise, in order to determine the gravity of separation between the No. 1 middlings and the No. 2 middlings, the cleaned coal and the No. 1 middlings must be considered as one product, and compared with the No. 2 middlings and the refuse combined as one product.

This procedure of combining products must always be followed whenever there is overlapping of material into many products, because the gravity of separation is determined in terms of the distribution of the total weight of material in the gravity consist; and if the analysis of data is made excluding a portion of the overlapping material, the gravity of separation so determined will be incorrect. Furthermore, the quantitative efficiency is based on the total weight of material in the products, and all material must be accounted for if the correct efficiency is to be obtained.

Based on this procedure of combining the products properly, the weight distribution between the cleaned coal and all other products is as shown in table XII.

**Table XII. Distribution of Material between Products at Cleaned Coal Discharge, Four-product Separation**

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal	No. 1 and No. 2 Middlings and Refuse
1.30	1.30	98.4	1.6
1.40	1.40	86.2	13.8
1.50	1.50	47.6	52.4
1.60	1.60	5.1	94.9
1.70	1.70	4.5	95.5
1.80	1.80	0.0	100.0
1.90	1.90	0.0	100.0
2.00	2.00	0.0	100.0
		0.0	100.0

The weight distribution between the combined cleaned coal and No. 1 middlings, and the combined No. 2 middlings and refuse is as shown in table XIII.

**Table XIII. Distribution of Material between Products at No. 2 Middlings Discharge, Four-product Separation**

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal and No. 1 Middlings	No. 2 Middlings and Refuse
1.30	1.30	100.0	0.0
1.40	1.40	100.0	0.0
1.50	1.50	95.2	4.8
1.60	1.60	94.9	5.1
1.70	1.70	50.0	50.0
1.80	1.80	5.1	94.9
1.90	1.90	4.8	95.2
2.00	2.00	0.0	100.0
		0.0	100.0

The weight distribution between the combined

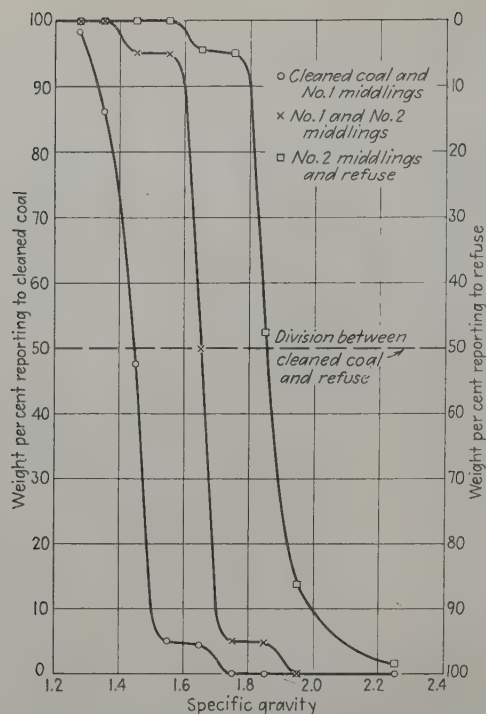


Fig. 3—Distribution of total weight between products, four-product separation.

cleaned coal, the No. 1 and No. 2 middlings, and the refuse is as shown in table XIV.

Table XIV. Distribution of Material between Products at Refuse Discharge, Four-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal No. 1 and No. 2 Middlings	Refuse
1.30	1.30	100.0	0.0
1.40	1.40	100.0	0.0
1.50	1.50	100.0	0.0
1.60	1.60	100.0	0.0
1.70	1.70	95.5	4.5
1.80	1.80	94.9	5.1
1.90	1.90	52.4	47.6
2.00	2.00	13.8	86.2
		-1.6	98.4

The values of weight distribution as shown in the foregoing three tables can be plotted on one chart, as illustrated in fig. 3.

The curves in fig. 3 show gravities of separation of 1.445, 1.65, and 1.855 as the points of separation between cleaned coal and No. 1 middlings, between No. 1 and No. 2 middlings, and between No. 2 middlings and refuse, respectively. Based on these gravities of separation, and an assumption of equal weight of products, the quantitative efficiencies can be calculated as follows:

		Pct of Total Products Misplaced
Cleaned coal	= 25 pct of total products and contains 10.5 pct sink at 1.445 gravity	2.63
No. 1 middlings	= 25 pct of total products and contains 9.5 pct float at 1.445 gravity	2.37
No. 2 middlings	= 25 pct of total products and contains 0.45 pct float at 1.445 gravity	0.11

Refuse = 25 pct of total products and contains no float at 1.445 gravity 0.00

Total misplaced 5.11

Therefore, the quantitative efficiency of separation between cleaned coal and No. 1 middlings =  $100 - 5.11 = 94.89$  pct.

Pct of Total Products Misplaced

Cleaned coal = 25 pct of total products and contains 0.5 pct sink at 1.65 gravity 0.12

No. 1 middlings = 25 pct of total products and contains 10.0 pct sink at 1.65 gravity 2.50

No. 2 middlings = 25 pct of total products and contains 10.0 pct float at 1.65 gravity 2.50

Refuse = 25 pct of total products and contains 0.5 pct float at 1.65 gravity 0.13

Total misplaced 5.25

Therefore, the quantitative efficiency of separation between No. 1 and No. 2 middlings =  $100 - 5.25 = 94.75$  pct.

Pct of Total Products Misplaced

Cleaned coal = 25 pct of total products and contains no sink at 1.855 gravity 0.00

No. 1 middlings = 25 pct of total products and contains 0.45 pct sink at 1.855 gravity 0.11

No. 2 middlings = 25 pct of total products and contains 9.5 pct sink at



	1.855 gravity	2.37
Refuse	= 25 pct of total products and contains 10.5 pct float at 1.855 gravity	2.63
	Total misplaced	5.11

Therefore, the quantitative efficiency of separation between No. 2 middlings and refuse =  $100 - 5.11 = 94.89$  pct.

Consideration will now be given to the combined, or overall efficiency of a multiple-product machine, the operation of which is inherently more difficult to understand than that of a two-product machine. For instance, in the preceding example, four products were obtained at three points of separation, and the combined efficiency is not evident. However, if this equipment had been operated to produce a two-product separation, making a cleaned coal at a specific gravity of separation of 1.445, and the two middlings products had been combined with the refuse as a total refuse product, the efficiency of the machine would have been 94.89 pct.

On the other hand, if the No. 1 middlings had been combined with the cleaned coal, and the No. 2 middlings with the refuse, producing a two-product separation at a specific gravity of 1.65, the efficiency would have been 94.75 pct.

Similarly, the efficiency of separation would have been 94.89 pct if the No. 1 and No. 2 middlings had been combined with the cleaned coal to produce a two-product separation at a specific gravity of 1.855.

However, if none of the four products had been recombined, the overall efficiency of the machine would have been the average of the efficiencies at the three points of separation, namely, 94.84 pct. The overall efficiency must be the average efficiency, because the efficiencies of separation at each point of separation are determined on a proportional basis; hence, the average of the efficiencies at each point of separation can be calculated on a straight arithmetic basis.

Thus, it will be seen that the overall efficiency of any equipment must be considered from the standpoint of the final disposition of the products. In this respect, it should be noted that there are a number of multiple-product machines that are operated to produce numerous products, which are combined into fewer products after removal from the machine. The Rheolaveur washer, air jigs, and water jigs are typical of such equipment; and it is the final combination of products which should be examined to determine the combined efficiency of the units, because combining products automatically reduces the final number of separations and products. This does not mean that study of the separations at intermediate points of separation is not valuable, because many times such study leads to important conclusions in regard to the effect of the intermediate separations on the final products.

Another example explaining why the final products must determine the overall efficiency of any machine is the concentrating table. It is a well-known fact that concentrating tables produce their final efficient separation of products by continuously

effecting inefficient separations at each riffle on the deck of the table; and, strange as it may seem, the net effect of these many inefficient separations at each riffle finally results in a highly efficient separation at or near the corner of the table where the final two-product separation is usually made.

Actually, the reason why the cumulative effect of many inefficient separations is a final separation of a high order of efficiency is because the separation made at each riffle is made on a material different from that treated at the previous riffle; and it is this continuous retreatment which finally results in an efficient separation at the discharge side of the table.

In similar manner, a jig discards a refuse product by means of the No. 1 elevator, and the products discharged at the succeeding elevators are of a different character because the material in the jig is continuously retreated between each successive elevator. The efficiency of separation of the materials at each elevator may not be perfect; nevertheless, the final separation between the combined elevator products and the cleaned coal overflow may be of a highly efficient order.

There are numerous other types of cleaning equipment which likewise continuously extract portions of the feed, and continue to retreat the remainder of the feed before the final separation is completed.

Therefore, it is obvious that inefficiencies at any one point of separation in a unit should not be cause for condemning a piece of equipment as inefficient if the net result of combining products effects a final efficient separation. Too often a machine is termed inefficient because there has been a misunderstanding in regard to the true conditions of separation which actually took place. The important point to be stressed is that frequently, when a satisfactory cleaned coal product is not obtained, the cleaning equipment is incorrectly labeled as being inefficient; whereas many times the actual separation is quite efficient, but the equipment is operating at too high a gravity to obtain a satisfactory analysis in the cleaned coal product.

Finally, a word of caution should be included to warn against the prevalent practice of confusing recovery of products with efficiency of separation. Recovery and efficiency are two entirely separate measures of cleaning performance, as will now be explained.

For instance, using the preceding example, if a two-product separation had been made, and the two middlings products had been discarded with the refuse by some piece of cleaning equipment that was able to make a nearly perfect separation at a specific gravity of 1.445, the efficiency of separation would have been about 5 pct higher than that of the machine that made this four-product separation.

On the other hand, the total recovery of material might very well have been much less than was accomplished by the four-product machine. The reason for this is that the total refuse product from a two-product machine would have included very nearly all of the feed having a gravity higher than 1.45; whereas the four-product machine classified a fairly clean coal and a fairly clean refuse, as well as two middlings products, which could be crushed

and retreated for further recovery, or sold to a different market.

Hence, despite the fact that the overall efficiency of the four-product machine in this example was only about 95 pct, the total recovery of marketable material could be greater than that produced by a more efficient two-product machine. It is true, of course, that greater recovery could be accomplished by installing several very efficient two-product machines in series, or by installing some other multiple-product machine of higher efficiency; but that is a question of economics, and will not be discussed in this paper.

To emphasize the difference between recovery and efficiency, the preceding example demonstrates that the recovery could be increased from 25 to 50 pct by accepting a washing gravity on a two-product separation of 1.65 instead of 1.445; and the efficiency of separation at 1.65 gravity will be 94.75 pct, in comparison with 94.89 pct at 1.445 gravity. This is a tremendous difference in recovery, but an almost negligible difference in efficiency. Of course, the ash contents of the products will not be the same in both cases, but that does not alter the fact that the recoveries are decidedly different, while the efficiencies are practically the same.

Furthermore, the preceding example should not give the impression that multiple-product machines are necessarily less efficient than two-product machines. No inferences should be derived from the examples of data in this paper regarding the relative merits of various types of equipment, because the efficiency of operation of any machine is dependent on the conditions under which the machine is operated. Conclusions in regard to the relative merits of various types of equipment can be deduced only from comparable sets of data obtained on various pieces of equipment when operating with the same type of feed under similar conditions.

Nevertheless, it may be thought that since efficiencies determined by the recovery method are obtained by comparing the actual recovery of cleaned coal with the theoretical recovery as shown by the cleaned coal ash content compared with the feed data, the efficiency by this method is a true measure of cleaning performance. Such a comparison would at first appear valid, but further examination of this method develops four reasons why it is misleading and should not be used:

1. The recovery method of determining efficiency credits effects of crushing to cleaning efficiency.
2. Efficiency values greater than 100 pct can be obtained by use of the recovery method of calculation, and the significance of an efficiency greater than 100 pct has never been explained.
3. The recovery method ignores the quality of the refuse, although the method should be just as valid for the refuse as for the cleaned coal.
4. The use of the method for a problem in multiple separation has never been explained; hence, the method is not generally useful for all types of separation.

In conclusion, it should be pointed out that some aspects of the concept of improperly distributed material have been discussed previously. The literature on the subject seems to indicate the following history of events:

In 1912, David Hancock proposed the Hancock chart,<sup>2</sup> which was probably the beginning of a basic understanding of the separation effected by a washer.

In 1929, Yancey and Fraser<sup>3</sup> discussed efficiencies in relation to the Hancock chart, and incorrectly labeled recoveries as efficiencies.

In 1937, K. F. Tromp<sup>4</sup> suggested that the curve of improper distribution resembled the Gaussian error distribution curve familiar to statisticians.

In 1938, Yancey<sup>1</sup> and his associates discussed improperly distributed material, but concluded that it did not give a satisfactory measure of efficiency because it did not take into account the relative difficulty of separation.

In 1947, Driessen<sup>5</sup> proposed a "relative-error curve" for measuring the faults of washing, but did not convert his values of error area to actual values of efficiency.

The formula for efficiency proposed herein is the result of following the work of previous investigators to its logical conclusion, namely, that a perfect separation, in which there is no improperly distributed material, constitutes a 100 pct efficient separation, and that anything less than a perfect separation can be expressed by 100 pct minus the improperly distributed material at the gravity of separation.

This formula does not attempt to take into account the various factors, such as size consist and size range of feed, gravity consist of feed, rate of feed, and other conditions that influence the efficiency of separation. It is acknowledged that the efficiency of separation is influenced by the conditions under which a machine must operate, but the numerical value of the efficiency should not have to express all of the combinations of operating conditions which caused its result. The operating conditions may be the cause for the result, but they are not the result, and should not be confused therewith.

Much of the above discussion is based on material that is not new in itself, but it is hoped that this re-examination of an old problem offers a practical approach to a better understanding of the relationships between efficiencies and gravities of separation, and that the procedure for analyzing problems of multiple separation will clarify a blind spot in the literature on coal cleaning.

As stated previously, this formula was motivated by the need for a practical means for judging cleaning plant performance, and this paper has resulted from our experience in its use over a period of four years. During this time numerous types of equipment and cleaning plants have been tested and evaluated with various coals; and these tests and evaluations have developed patterns of separation that could not be found in the literature on coal cleaning.

## References

- <sup>1</sup> Performance of a Pulsator-Type Coal-Washing Jig. U. S. Bur. of Mines, R. I. 3372 (Jan. 1938) pp. 9-10.
- <sup>2</sup> Iron Making in Alabama. Ala. Geol. Survey (1912).
- <sup>3</sup> U. S. Bur. of Mines. Bull. 300.
- <sup>4</sup> Glückauf, 73. *Colliery Guardian*, 154.
- <sup>5</sup> *Coal Tech.* (Aug. 1947). *Trans. AIME* (1948) 177.



## Single Retort Underfeed Stokers

by H. A. Baumann,

T. S. Spicer, and

C. C. Wright

Experimental data are presented showing the influence of size consist and firing rate upon the performance of bituminous coal-fired, single-retort, industrial underfeed stokers. Size segregation, degradation, and distributor data are also included. The conclusion is reached that size consist of the fuel as fired, is a critical factor in its performance on this class of equipment.

ALTHOUGH the efficiency of coal utilization has improved steadily since the turn of the century as a result of continued research and development, little of this increase in efficiency can be credited to improvements in the single retort stoker. There are still numerous problems connected with the burning of coal on this type of unit, the solution of which would be a stimulus to coal and stoker sales and a boon to the user. Among the more critical of these problems is that of burning slack bituminous coal of strong caking tendency and containing large percentages of fine material. With the increased trend toward mechanical mining, it may be expected that in the future an even larger percentage of the coal available for use on this class of stokers will contain higher percentages of the fine sizes, thus aggravating an already serious problem.

During the war, the problem of efficient and economic burning of available coals on the single retort industrial stoker, particularly the class 4 stokers (301 to 1200 lb per hr), became quite critical in many localities. In addition to the inherent problems associated with the use of this class stoker, the coals available were frequently not those best suited for use on this equipment and the manpower shortage made it difficult as well as expensive for the consumer to provide the labor necessary to keep the fires in reasonably satisfactory condition. Although exact figures on the number of class 4 stokers are not available, the yearly tonnages of coal consumed are of considerable consequence and have been estimated\* at between 15 and 20 million

\* E. G. Bailey: Trends of Coal Preparation and Fuel Burning Equipment. Proc. Fuel Engineering Conference, Coal Bureau, Upper Monongahela Valley Association, July 24, 1942.

tons. Largely as a result of these considerations,

the stoker research staff of the Mineral Industries Experiment Station of The Pennsylvania State College was requested to study the performance of the single retort industrial stoker and to undertake a program of research on the more efficient burning of bituminous coal on this class of unit. Various

---

H. A. BAUMANN, T. S. SPICER, and C. C. WRIGHT, Members AIME, are Research Engineer, Associate Professor, and Professor, respectively, Division of Fuel Technology, Pennsylvania State College, State College, Pa.

AIME Chicago Meeting, February 1946.

TP 2742 F. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received June 8, 1949.

This paper is a contribution from the Division of Fuel Technology of The Pennsylvania State College, State College, Pa. Investigation jointly sponsored by the Commonwealth of Pennsylvania, the Western Pennsylvania Coal Operators Association and the Central Pennsylvania Coal Producers Association and conducted in cooperation with the Combustion Engineering Co., New York, N. Y.

aspects of overall problem have been investigated since the inception of this program in 1942, only one of which is here reported.

### General Considerations

There are numerous factors which appear directly or indirectly to affect the overall performance of bituminous coal underfeed stokers, such as the caking or matting properties of the coal, the contact relationships between coal and air, the ash fusion characteristics, the burning rate, and the design of the stoker and furnace in which the coal is burned. An analysis of the influence of each of

these factors indicates, however, that all are inter-related and interdependent. For example, the contact relationships between coal and air has a definite bearing upon the caking and matting properties as evidenced by the phenomenon of preoxidation and its influence upon the character of the coke. Contact relationships also influence the burning rate since this reaction is a function of surface area and oxygen concentration. Similarly, the character of the coke influences the contact relationship between air and fuel, and is itself dependent upon the flow of heat to and from the fuel bed, which in turn is influenced by furnace design and by burning rate.

Reducing the problem of combustion on under-

serve as an aid in interpreting actual performance and may point the way to design improvements.

### Experimental Investigations

**Equipment and Installation:** The single retort underfeed stoker employed in the studies herein reported has a nominal rating of 540 lb of coal per hour. The general features of the stoker are shown schematically in fig. 1. The coal is delivered from hopper to retort by means of a screw, and distribution along the retort is accomplished with a ram type pusher; the grate bars are of the so-called movable type in which every third bar moves while the pair between is fixed.

The stoker is installed in a well-insulated brick

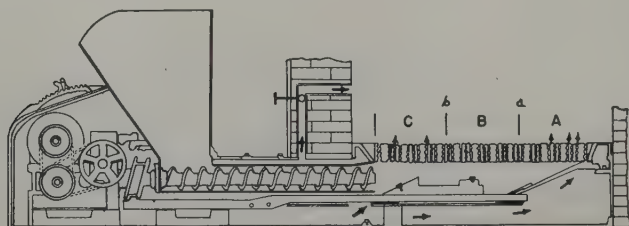


Fig. 1 — Sectional elevation of experimental stoker.

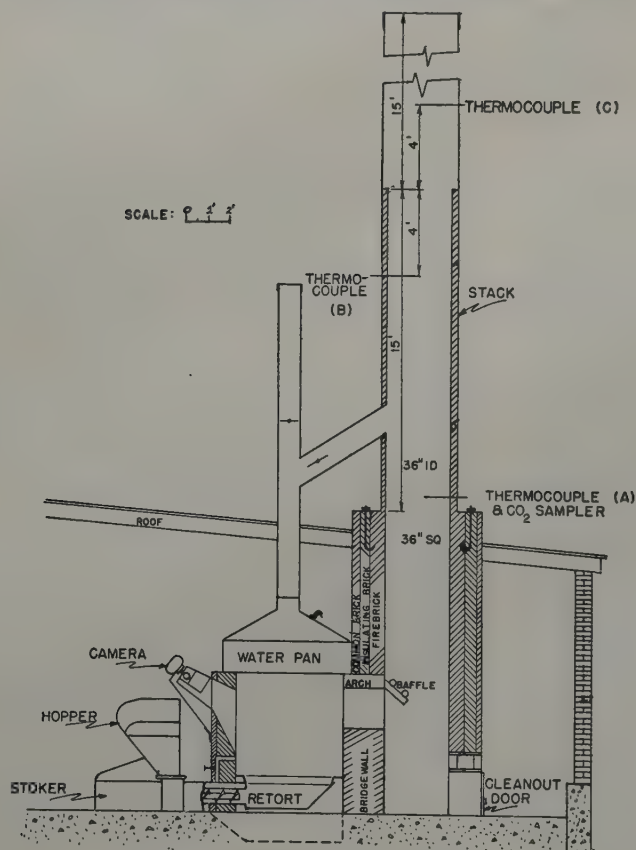


Fig. 2 — Sectional view of industrial stoker experimental equipment.

feed stokers to its essentials, it would appear that one of the primary factors governing performance is the rate of heat transfer to, from, and through the coal. For a given installation, this rate of heat transfer is largely a function of burning rate which is influenced to an appreciable extent by the size consist of the bed. It is the purpose of this paper to present data showing some of the relationships existing between performance, burning rate, and size consist of the coal in the hope that they may

furnace with a water pan replacing the boiler. Although a small percentage of the heat liberated is absorbed by the water in the pan, the major portion is discharged up the stack as sensible heat of the gases. This arrangement is much cheaper to install and permits greater flexibility for test purposes than does the standard stoker-boiler installation. It will be noted in fig. 2 that the front wall of the furnace is so constructed that from the turret above the hopper level it is possible to see



the entire fuel bed and, when desired, pictorial records of the operation can be secured.

Forced air delivered to the stoker is measured in an especially constructed inlet duct by means of a calibrated Pitot tube. The carbon dioxide content and temperature of the stack gases are recorded on a strip chart instrument. The arrangement of the air duct and the points at which air, carbon dioxide, and temperature are measured are indicated in fig. 2 and 3.

**Properties of Coals Tested: Description and Analyses:** Descriptions and analyses of the coals tested are shown in table I. Although these coals were all from the Pennsylvania coal fields, it is believed that the results and conclusions are generally applicable to bituminous coal of comparable quality from other sources.

**Screen Analyses:** Screen analyses for each of the coals tested are presented in graphic form in fig. 4. These analyses were made as follows: a 2000 lb sample of each of the test coals was taken from stock and each was coned and quartered according to the standard ASTM procedure. Alternate quarters were combined and the 1000 lb samples were screened using the Tyler Ty-Lab shaker with 2,

**Table I. Description of Test Coals of Pennsylvania**

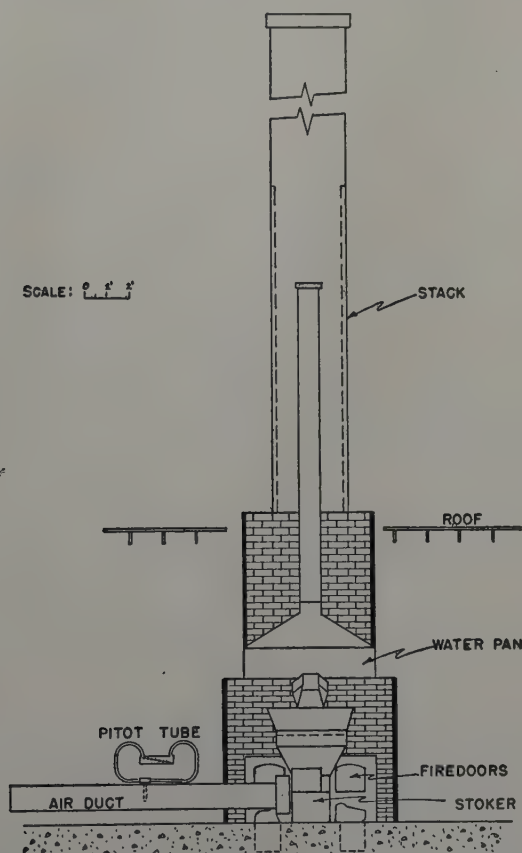
Source and Size of Coals			
Coal	Seam	County	Nominal Size
A	Pittsburgh	Allegheny	2 in. x 0 nut and slack
B	Lower Kittanning	Cambria	2 in. x 0 nut and slack
C	Lower Kittanning	Cambria	¾ in. x 0 slack
D	Upper Freeport	Indiana	2 in. x 0 nut and slack
E*	Upper Freeport	Indiana	2 in. x ¾ in.

**Coal Analyses, As Received Basis**

	Coal A	Coal B	Coal C	Coal D
Proximate Analysis:				
Moisture, pct	1.8	0.8	1.2	0.3
Volatile matter, pct	36.3	21.1	22.7	29.8
Fixed carbon, pct	53.4	71.3	68.9	59.8
Ash, pct	8.5	6.8	7.2	10.1
Sulphur	1.82	2.2	1.8	1.5
Heating value, Btu	13,415	14,430	14,330	13,725
Ash Fusion Characteristics:				
I.D.T., °F	1,930	2,160	2,120	2,250
A.S.T., °F	2,240	2,430	2,450	2,470
F.T., °F	2,360	2,530	2,540	2,740
Bulk density, lb per cu ft	50.5	49.5	48.5	49.0
BSC Swelling Index	7.5	9+	9+	8.0

\* Coal E was prepared from coal D at the mine by passing over a ¾ in. screen.

**Fig. 3—Front view of industrial stoker experimental equipment.**



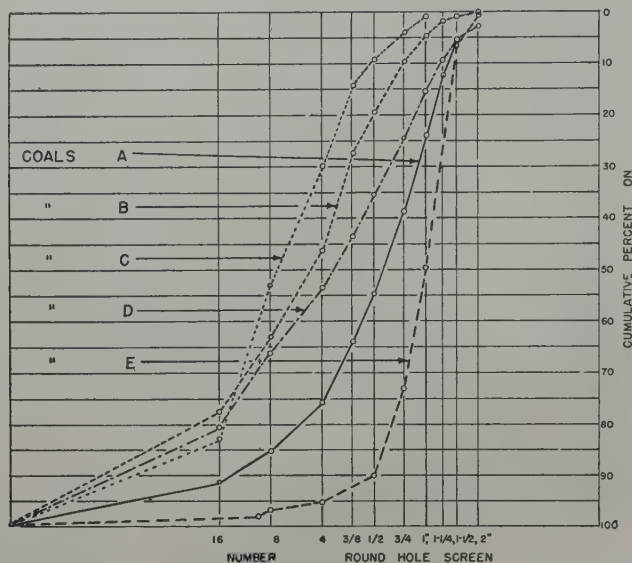
1½, 1¼, 1, ¾, and ½ in. round hole screens, and U.S. screens, No. 4, 8, and 16. The screen fractions were remixed by coning with as little handling as possible and were used in subsequent degradation tests.

**Coal Degradation:** Degradation tests were performed on each coal by feeding each of the previously screened and remixed samples through the stokers at a feed rate of approximately 540 lb per

hr, which is the nominal rating for the unit tested. The coal fed through the stoker was collected as soon as it passed the level of the top of the grate bars. Screen analyses on the coal thus collected were performed in a manner similar to that used in the original screen analysis. The results of these degradation tests together with screen analyses of the original coals are shown in fig. 5, 6, and 7 for coals A, B, and C, respectively.

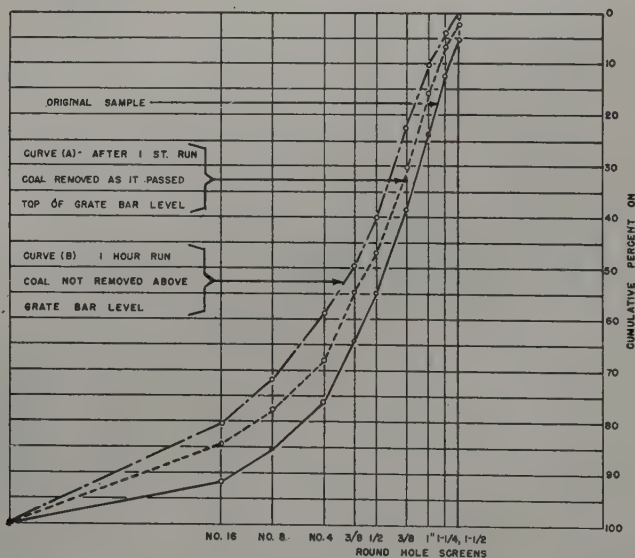
**Coal Segregation:** The coal samples used in size degradation tests were made up to the original screen size analysis by addition or subtraction of fractions of the same coals obtained from screen analysis of the alternate quarters not used in the degradation tests, and were used for segregation tests. Segregation of sizes in passing from hopper to retort of the stoker was determined by dividing the retort into three sections, A, B, and C, of equal

of the three sections carefully removed and its size distribution determined by screen analysis. The foregoing procedure was repeated with the feeding time being continued for 60 min rather than 30 min. Fig. 8, 9, and 10 show the screen analysis of the original coal sample, and screen analyses after the first filling of the retort, after 30 min of continuous operation, and after 1 hr of continuous operation, for the coal in sections A, B, and C of the retort



**Fig. 4 — Screen analyses of test coals.**

**Fig. 5 — Screen analyses of degradation test on coal A.**



area at the grate bar level, as illustrated in fig. 1. Plates were made which could be inserted into the retort at "a" and "b" to permit removal of the coal from the three sections without intermixing of the coal in the three sections. With the plates removed, coal was fed from the hopper until the retort was filled to the level of the top of the grate bars. The stoker was then stopped, the plates inserted at "a" and "b" and the coal from each section carefully removed and its size distribution determined by screen analysis. The plates were then removed and feeding of the coal resumed. After 30 min of feeding coal the stoker was again stopped, the plates inserted at "a" and "b," excess coal above the level of the top of grate bars removed, the coal in each

when using hard structure coal A. Fig. 11, 12, and 13 show similar data for soft structure coal B.

**Coal Distribution:** During the operation of the stoker for segregation and degradation tests, observations were made on the quantity distribution of the various coals over the grate area. No endeavor was made to modify the distribution by changing the arrangement of the pusher blocks once an arrangement had been secured which gave good quantity distribution for the first coal tested. Photographs were made at definite times to record the quantity distribution of the coals.

**Combustion Tests:** A large number of combustion tests have been completed in which various factors such as shape and design of grate bar and



retort, motion of distribution ram, air zoning and related mechanical features as well as such factors as the influence of firing rate, size consist of the fuel and preoxidation of the fuel by various methods have been studied. In each of these combustion tests the following standard operating procedure has been employed.

The fire is lighted using wood and kerosene and

the furnace brought up to temperature over a period of 2 to 3 hr. During this period the air supply is adjusted to approximately 50 pct excess on the basis of predicted coal feed rate and measured air to fan. The true coal feed rate is subsequently determined from the actual weight of coal fed during the test and the reported excess air is calculated accordingly. At the start of the actual test period,

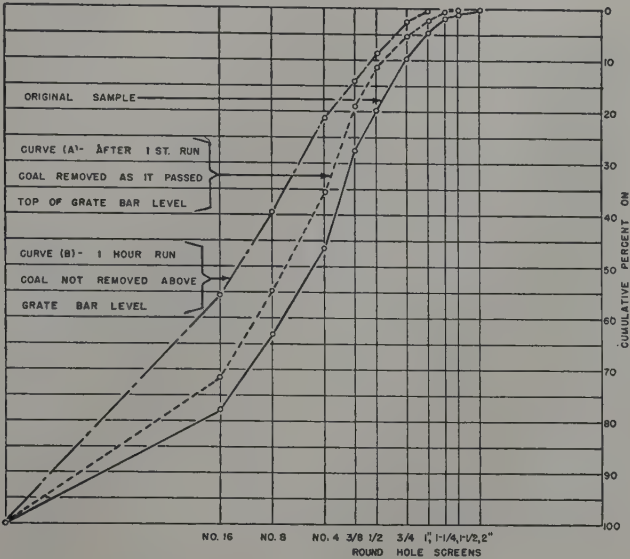


Fig. 6 — Screen analyses of degradation test on coal B.

Fig. 7 — Screen analyses of degradation test on coal C.

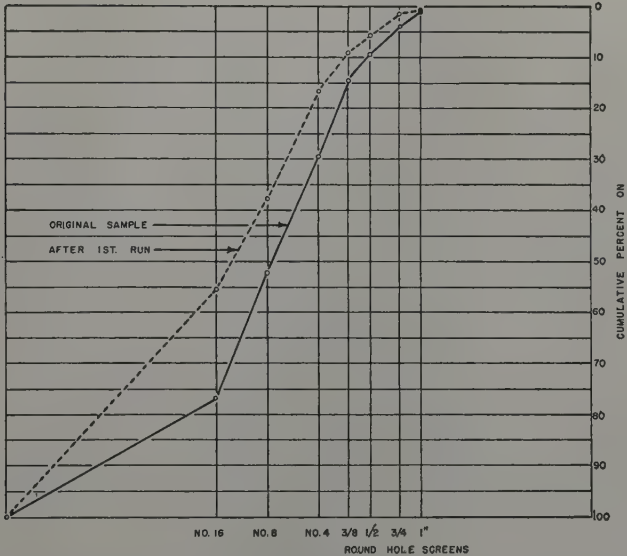


Table II. Combustion Data for Firing Rate Tests

Coals	Normal Rating, Pct	Feed, Lb per Hr	Air <sup>a</sup>		CO <sub>2</sub> Pct			Stack Temp. °F.			Relative Performance
			Excess Corrected, Pct	Total Corrected CFM	Min.	Max.	Avg.	Min.	Max.	Avg.	
A	50	270	54	970	3.8	7.4	5.5	820	1,220	950	Fair
A	100	541	49	1,900	7.2	12.0	9.0	1,280	1,480	1,350	Good
A	150	787	1	1,875	6.0	12.0	10.0	1,520	1,720	1,600	Good
B	50	283	35	975	4.0	7.0	6.5	840	1,220	1,100	Poor
B	100	547	45	1,995	6.4	8.9	8.0	1,150	1,420	1,300	Fair
B	150	788	21	2,380	8.0	12.4	10.5	1,200	1,820	1,500	Fair
C	50	263	48	975	4.0	5.4	4.6	800	980	900	Very poor
C	100	543	48	1,995	5.4	9.6	7.4	1,200	1,660	1,400	Poor
C	150	778	5	2,040	8.0	14.0	11.2	1,600	1,970	1,580	Poor

<sup>a</sup> These data are for measured quantities of air delivered by the fan, corrected to 70°F and 29.92 in. Hg pressure and upon the weight of coal fed.

after the furnace has attained operating temperature, the bed is leveled by raking if such action is required. After this initial raking, when employed, no manual attention is given to the bed for any reason for the equivalent of one shift, or 8 hr from the time the fire was lighted. The relative performance of the coals under the various test conditions is evaluated principally by visual observation and by time-lapse motion picture studies, although

than the actual, which is based on the  $\text{CO}_2$  content of stack gases. Since air leakage was held to a minimum,  $\text{CO}_2$  results appreciably below those corresponding to the measured excess air are an indication of the extent to which the fuel was accumulating in the bed.

The other series of tests was made in order to study the effect of size and of size consist of the fuels upon the performance. In each of these tests,

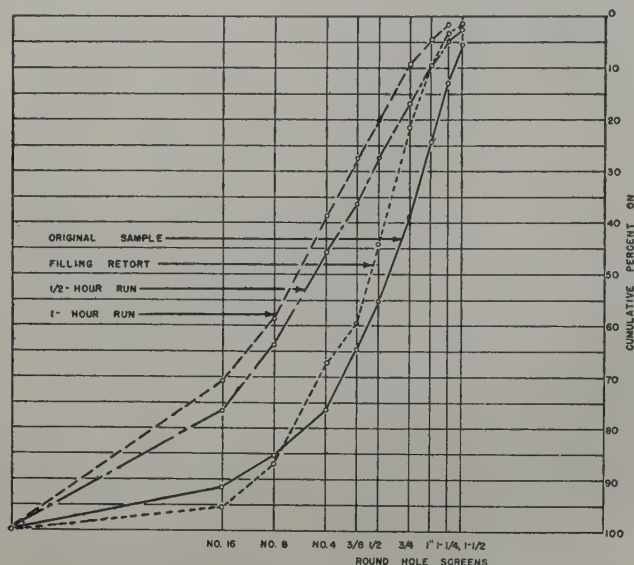


Fig. 8 — Screen analyses of segregation test on coal A, retort section "A."

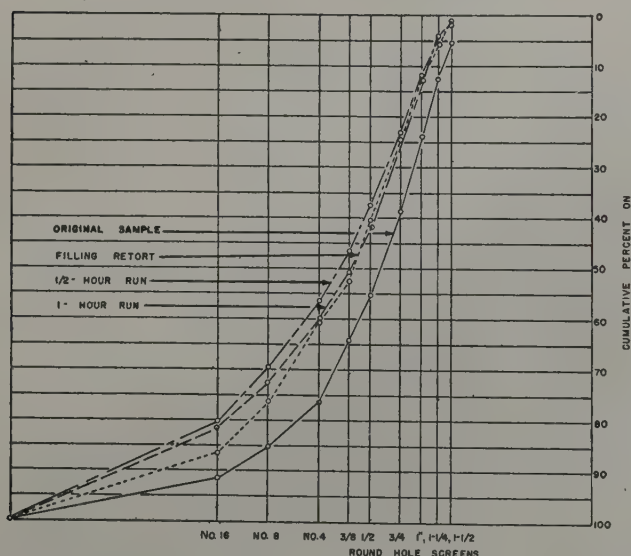


Fig. 9 — Screen analyses of segregation test on coal A, retort section "B."

$\text{CO}_2$  and stack temperature records are usually taken and serve as secondary criteria of performance.

In the present paper, the results of only two series of tests are reported because design features are beyond the scope of this report.

One series of tests was made at three different firing rates; namely, 50, 100, and 150 pct of rating for this particular stoker, in order to study the influence of this factor upon the performance of the various coals. The results are reported in table II. It should be noted that the percentage excess air reported in col. 4 is based on the volume of air delivered to the bed by the fan and the quantity of coal fed. Thus this is the theoretical excess rather

the coal feed rate was maintained at approximately 100 pct of normal rating and 50 pct excess air was employed.

For this series of tests, 5000 lb of coal D was prepared, in which the size consist was made equivalent to that of coal A. Coal A was prepared so that 5000 lb had a size consist equivalent to that of coal B. Coal D was also modified to the extent of screening out the  $3/4$  in. material at the mine; however, a small amount of minus  $3/4$  in. coal remained. It would have been desirable to change the size consist of all coals to conform to each other, but this was not possible with coal C ( $3/4$  in. by 0) and, moreover, coal B had such a small percentage of the larger sizes that very large quantities would



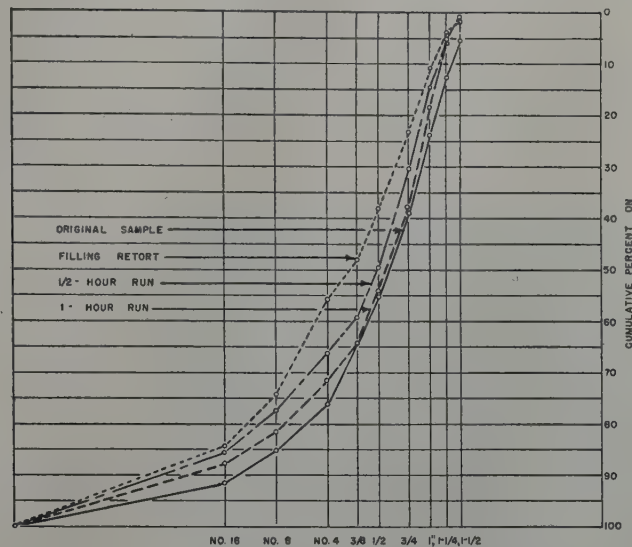
**Table III. Comparative Combustion Performance Rating of Size Consist Studies**

Coals	Results
Coal A (2 in. x 0)	Excellent
Coal D with coal A size consist (2 in. x 0)	Good
Coal A with coal B size consist (2 in. x 0)	Fair
Coal B (2 in. x 0)	Fair
Coal D (2 in. x 0)	Poor (heavy coke masses)
Coal C (¾ in. x 0)	Poor (semi-coke mat)
Coal D (2 in. x ¾ in.)	Very poor

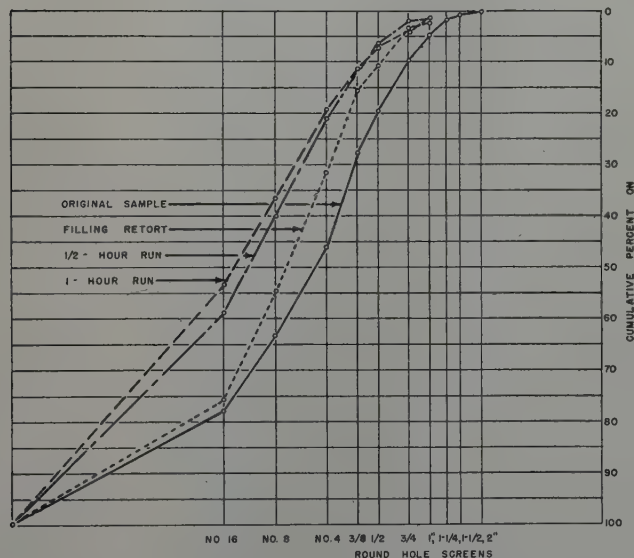
The general lack of information on this subject would suggest, however, that the importance of this factor in stoker performance may have been underrated.

The results shown in curve "A" of fig. 5, 6, and 7 for the degradation resulting when coals A, B, and C discharge freely at the level of the top of the retort and in curve "B" of fig. 5 and 6 where it discharges against the weight of coal above the retort formed by the natural contour of the fuel bed, sug-

**Fig. 10—Screen analyses of segregation test on coal A, retort section "C."**



**Fig. 11—Screen analyses of segregation test on coal B, retort section "A."**



have been required. The qualitative results of these tests, as judged by fuel-bed conditions, general performance, and time-lapse motion picture studies, are presented in table III.

#### Discussion of Results

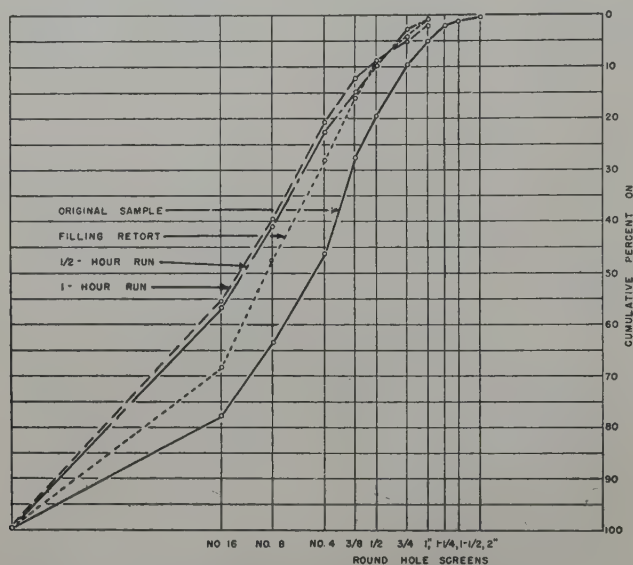
**Degradation:** During the past decade, considerable attention has been focused upon the degradation of coal in domestic underfeed stokers, but little or no mention of this phenomenon can be found in the technical literature concerned with the operation of commercial and industrial underfeed stokers. Obviously the amount of degradation will be a function of the strength characteristics of the particular coal, of the size consist of the coal, and of the design characteristics of the particular stoker.

gest that degradation is a very serious consideration in some cases. It is highly probable that in actual field operation the degradation results will be worse than those shown, because in many cases the mass of coke over the retort will offer more resistance to movement of the entering coal than would the depth of coal formed by the natural contour of the unignited bed. It is apparent that with the soft structure coal B, appreciable degradation occurs as a result of forcing the coal upward against the head of fuel above the retort. The increase in fines (minus ⅛ in. material) is appreciable for both coals.

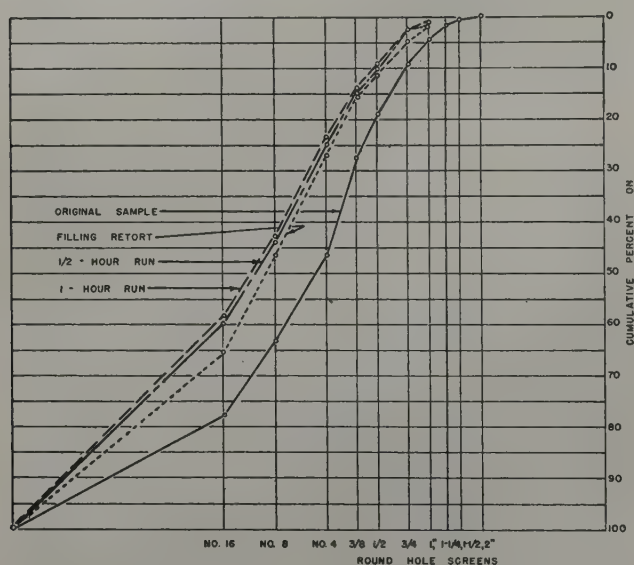
As may be seen from an examination of fig. 1, the stoker in which these tests were performed employed a screw feeder from hopper to retort ap-

proximately 50 in. long, which discharges the coal to the distribution ram which moves the coal along and upward in the retort. No data were secured on the relative amount of degradation produced by the screw and by the ram, but it is believed that the

During the initial filling of the retort there is a tendency for the fines to concentrate toward the hopper end, but with continued operation when the coal is fed against a head of coal, this tendency is reversed and the fines tend to concentrate in the



**Fig. 12—Screen analyses of segregation test on coal B, retort section "B."**



**Fig. 13—Screen analyses of segregation test on coal B, retort section "C."**

distributing ram produces relatively little breakage. It would be of interest to have degradation data for stokers employing only screw feed and for those employing only ram type feed. Such data do not appear to be available, however, at the present time.

**Segregation:** When transporting broken solids one of the most difficult problems encountered is that of preventing segregation of the coarse material from the fines. The problem is especially difficult with the single retort underfeed stoker because the coal must be fed from the hopper into the retort and then distributed over the grate; this involves three changes in direction. The results presented in fig. 8, 9, and 10 for the distribution of coal sizes in various sections of the retort when feeding coal A indicate that considerable segregation may occur.

end away from the hopper. It was observed during these tests and recorded photographically (fig. 14) that the maximum concentration of fines occurred immediately above the point where the retort bottom changed slope, which is also the point where the end pusher block ends its stroke. Whether the change of slope or the pusher block action is the cause of this, concentration cannot be stated definitely but the result of subsequent changes in pusher block arrangements would suggest that the slope is probably the controlling factor.

Similar data for coal B are shown in fig. 11, 12, and 13. During the initial filling of the retort the same tendency as observed with coal A was noted, but after the first 30 min of operation variations in distribution of sizes in the three different sections of the retort were hardly noticeable. Whether this is due to the preponderance of fines being delivered



to all sections or whether it is because the size distribution has approached some consist which minimizes segregation cannot be stated. As with coal A, however, it was observed and recorded photographically that there was a tendency for the fines to attain a maximum concentration in a narrow band across the retort immediately above the point where the retort bottom changed slope.

In subsequent tests, it was demonstrated that it was possible to predict with unfailing accuracy from segregation studies the location along the retort where the worst coke masses and dead spots would occur if such developed. Invariably the trouble would occur at the point of maximum fines concentration.

**Distribution:** As the length of retort and the grate area increase, it becomes correspondingly more difficult to maintain an even quantity distribution of coal on the grates. The problem is further complicated by the fact that the distribution is affected by size consist, moisture content, and such other physical characteristics of the coal as affect its movement. Frequently it will be found that the distribution pattern for burning beds does not follow the same pattern as so-called dry (unignited) beds, because of the segregation phenomenon discussed in the preceding section. The converse, however, is rarely true. If the distribution pattern of the dry bed is unsatisfactory, it rarely will be found to be satisfactory for the bed when burning. It is thus desirable when starting with a new coal to obtain first a satisfactory distribution with a dry bed by appropriate changes in pusher block arrangement and stroke of ram. If results with the live bed are then unsatisfactory, further changes can be made.

It should be pointed out also that the distribution pattern may be markedly different for two coals in the same equipment with the same setting of pusher blocks and stroke of ram.

**Influence of Size Consist on Performance:** The qualitative results presented in table III for the burning of coal D with initial coal A size consist and coal A with initial coal B size consist are indicative of the importance of this factor. It would have been more desirable to run this series of tests using coals having exactly the same size consist at the point where the coal burned, but, as previously shown, the size consist at this point is a function of several factors that are not predictable when the test is started. Consideration was given to using, as a standard, the size consist as delivered at the retort level against no head or against heads ranging up to the equivalent to that produced after 1 hr of continuous feed. In view of the uncertainty as to the exact size consist of coal A delivered at the grate level under actual firing conditions, however, it was decided that the safest standard to adopt would be the size consist of the original coal.

The test results established with reasonable certainty that, at a fixed firing rate in a specific installation, size consist is the most important single factor controlling the performance of the coal. That results comparable to those normally attained with coal A were not attained with coal D is believed to be due largely to the fact that the latter had a less satisfactory size consist at actual burning level.

Similar results were obtained in the test where a

sample of coal A having the size consist of coal B was burned. There appeared to be little doubt that, had the consist of coal A been the same as that of coal B at the actual burning level, the results would have been essentially the same.

The results of the test with the sample of coal D from which most of the minus  $\frac{3}{4}$  in. material had been removed is a further indication of the importance of size consist. The performance of this coal was so poor that it was impossible to maintain combustion in this particular installation. Lack of reaction surface appeared to be the cause for loss of ignition.

**Influence of Firing Rate:** There exists a rather common belief that as long as the stoker is oversized for its combustion assignment the performances will probably be better than if it is under-

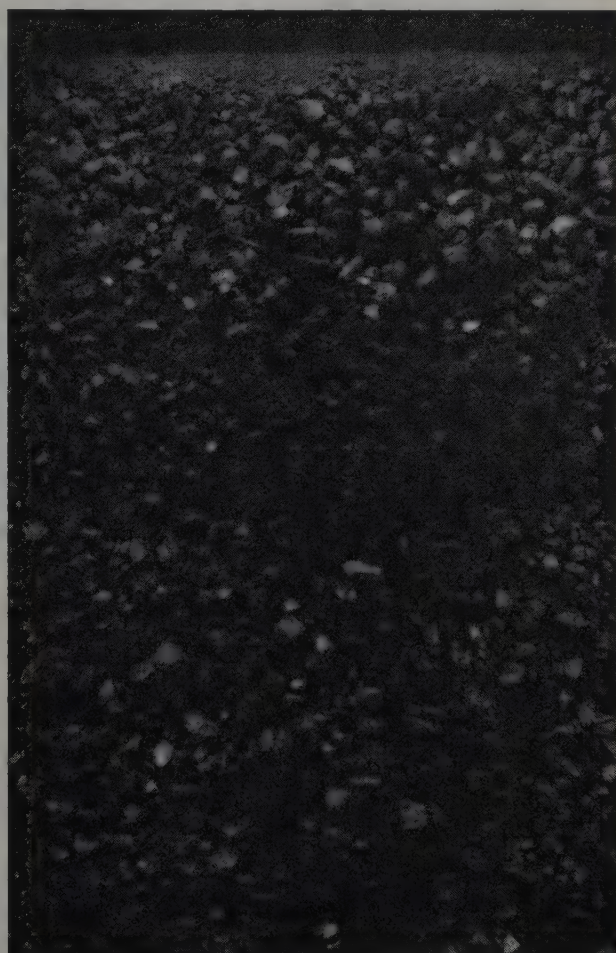


Fig. 14—View of unignited fuel bed showing band of fines concentrated above point where bottom of retort changes slope. Coal A. Top, bridge wall. Bottom, hopper end.

sized. That this may be an erroneous assumption is shown by the data for tests at 50, 100, and 150 pct rating presented in table II. With coal A, the manufacturer's rating gave the best results, 150 pct rating was almost as satisfactory and 50 pct rating gave much poorer performance. With coal B there was little to choose between the 100 and the 150 pct rating and had it been possible, with the installed fan, to supply sufficient excess air it is be-



lieved that 150 pct rating might have been the better of the two. The results of operation at 50 pct rating were very much poorer than at the higher ratings. The relationships between the performance for coal C at the three ratings were similar to those for coal A although performance at all ratings was very much poorer.

These results indicate the importance of the heat transfer problem. At the higher firing rates the overall furnace temperature was higher and permitted sufficient transfer of heat from furnace to fuel bed to insure at least partial coking of the coal which in turn permitted the air to channel through the coke crevices and maintain active combustion. At the low firing rate, however, insufficient heat was radiated to the bed, the character of the coking was altered and instead of producing fissured coke

tent is a function of the firing rate. The probability exists that the best size consist for producing optimum performance of a given coal in a specific installation is not the one that is most economic for the coal producer to supply. In some cases, it may be feasible for the mines to furnish a consist more suitable for use on the single retort stoker, but in general other means of overcoming the difficulty must be found.

Firing rate studies on a limited number of coals indicate that performance can in most cases be improved by judicious selection of firing rates where load and other conditions warrant such a practice. While the optimum firing rate may coincide with the manufacturer's rating, this is not necessarily the case since it is impossible for the manufacturer to predict the performance of his equipment with



Fig. 15—View of ignited bed burning coal C at 50 pct rating. Green coal blanketing 80 pct of total grate area. Top, bridge wall. Bottom, hopper end.

over the retort section, a relatively impervious mat of semifused coal resulted, through which the air could penetrate only with difficulty. This results in most of the air blasting along the grate bars and produces intense localized combustion around the edge of the bed with virtually no combustion over the retort section. In extreme cases, such as with coal C, where in addition to poor heat transfer from the furnace, the size consist is such that heat transfer through the coal itself is poor, the condition of the fuel bed may become extremely poor with green coal pushing up through the center of the bed all along the retort. This mass of green coal spills down over the bed of fused or semifused coal on the grates and may at low firing rates extinguish the fire. A close approach to this condition may be seen in fig. 15 where coal C was fired at 50 pct rating. Green coal blanketed the bed over approximately 80 pct of the total grate area.

#### Summary

Experiments with a limited number of coals in a specific stoker-furnace test installation indicate that at a given firing rate, size consist of the fuel as actually burned is one of the major controlling factors in determining the relative performance of different coals. With a given coal, performance is influenced largely by the rate of heat transfer to, from and through the fuel bed which to some ex-

a wide variety of coals which behave differently. The plant engineer in cooperation with combustion engineers from the coal supplier and equipment manufacturer can do much to establish the best conditions for operation under the specific local conditions. The test results suggest that the so-called coking, caking, or matting characteristics of the coal are not intrinsic characteristics and that the performance of a number of coals, generally considered quite different in these characteristics, is actually quite similar when conditions of size consist, as fired, and of rate of heat transfer are about the same.

Although size consist and rate of heat transfer appear to be controlling factors in determining performance, it appears probable that much could be accomplished in the design of both stoker and furnace which would influence the effect of these factors and permit the efficient burning of bituminous coals which at present do not generally give satisfactory performance on this class of stoker.

#### Acknowledgments

The authors gratefully acknowledge the assistance of George Langsford, Laboratory Assistant, and Carl Neubling, Sr., Analyst, in the performance, screening and analytical tests, and the helpful comments and suggestions of engineers from the Combustion Engineering Company.



# Research in Coal Geology

by Gilbert H. Cady

The present article calls attention to the activities of a committee on coal research of the sister society of Economic Geologists. The field of coal geology crosses the boundary between the two societies and joint meetings are as applicable here as in the metallic and other nonfuel fields of mining and natural resources recovery. The six fields of coal geology are briefly described and some of the ways in which they touch on coal mining, preparation, and recovery briefly indicated.

THE application of geology to problems arising in coal mining engineering and coal preparation has, in general, been somewhat remote, or the geological problems have involved such simple forms of geological knowledge that the attention of specialists in this field has not been commonly thought necessary. Furthermore, the geology of coal beds and coal mining has received but scant attention in our educational system, even in mining education; so that coal mining engineers have become somewhat conditioned against geological thinking. Geologists, on the other hand, have commonly failed to realize that there are geological aspects to coal mining and coal preparation problems. A further reason why geology receives so little consideration in coal mining undertakings in this country lies in the fact

that our coal supplies are, or have been thought to be, almost unlimited, have been easily accessible, and very cheaply mined.

Conditions have been changing within recent years. Failures can be less easily endured. Great inroads have been made on our bonanza deposits of easily mined coal and the increasing cost of coal and the increasing difficulty of finding it in easily accessible and large quantities places greater and greater importance on accurate information and good technique in exploration. It seems probable, therefore, that the opinions and aid of geologists will be more commonly sought than in the past in the exploration for coal and in the development and operation of coal mines, and analyses of mining conditions, preparation, and utilization.

---

GILBERT H. CADY, Member AIME, is Senior Geologist and Head, Coal Division, State Geological Survey Division, Urbana, Illinois.

AIME Columbus Meeting, September 1949.

TP 2774 F. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received Sept. 27, 1949.

---

It may be timely, therefore, to extend to the Coal Division the recognition that geology receives in the Institute, in the fields of hard-rock mining, of petroleum engineering, and of nonmetallic-, non-fuel-resources engineering, evidenced by joint sessions with the sister society of Economic Geologists. It is my personal feeling that at least occasionally, with profit to both engineers and geologists, joint sessions might be arranged between the two societies similar to those set up with other divisions of the Institute, thereby bringing the two groups closer together in the field of common interest. A considerable number of the geologists who are members of the Coal Division are also members of the Society of Economic Geologists and have taken part in the activities of its Coal Research Committee. Such members and doubtless others would profit by sessions of the Coal Division in which geological aspects of coal occurrence, characteristics, distribution, roof conditions, and other matters relating to coal geology are considered. Such joint sessions would likewise provide a much needed stimulus to interest in coal geology in the academic field.

The field of coal geology stretches across the boundary between the two societies. Fundamental research in one group will have its eventual effect upon thinking and activities in the other group. Both societies have a real interest and desire to promote such research. The Committee of Coal Research of the Society of Economic Geologists believes that an account of the deliberations and activities of the Committee will be of interest to the members of the Coal Division. We might even obtain moral support. The main objective of the Committee is to stimulate interest in the geology of coal and to formulate a program of study and fundamental research along geological lines.

Coal geology may be conveniently organized with respect to six varieties of study and research: (1) coal resources investigations; (2) coal mining geology; (3) natural history of coal, origin of coal beds and of coal; (4) geobotanical investigations, coal botany and paleobotany; (5) coal petrography; and (6) coal geochemistry. A thorough understanding of these various branches of coal geology is not to be expected of a single individual but specialists in the different fields will develop if research is adequately supported. The content of each of the six fields of knowledge will be briefly considered.

**Coal Resources Studies:** Delineation and description of the location, distribution, thickness, structural and mining conditions of the coal deposits of areas of the earth have long been regarded as the prime concern of coal geology. It is the duty of the coal geologist engaged in coal resources studies to determine the facts in regard to the coal resources. On the basis of the accumulated information the engineer, or any other person that is qualified to do so, can estimate the reserves available for economic recovery under whatever conditions prevail at the time. The estimate of resources change only as information provides better understanding of the conditions of coal occurrence, distribution, and minability. The estimate of reserves which will usually be less than the resources may change from year to year as conditions determining mining costs

and other factors in the competitive picture change. Within the province of these activities information is assembled relative to folds, faults, floor, and roof conditions, and the local character of and the regional variation in the chemical character of coal.

**Coal Mining Geology:** This phase of coal geology is largely unexplored but will be one of greater and greater importance as it becomes more and more necessary to extend mining operations in the preferred coal beds into areas where mining conditions are less favorable and less well known. Studies need to be extended to include observational and experimental investigations of the geological factors determining strength of roof. Various other aspects of mining geology are involved where beds are much folded or faulted, where water or gas is troublesome, where methods of exploration of coal properties require geophysical methods, and in the development of new properties, and in its interpretation of drill cores.

**Natural History of Coal:** Geological activities are also appropriately extended to include studies concerning the natural history of coal beds and of coal itself. What are the conditions under which plant debris accumulated? What portions of the plants withstood longest the geological vicissitudes of coalification? What parts were most readily "homogenized" to use a term and idea employed by Professor Wilhelm Petrascheck, of Leoben, Austria? These are questions which concern particularly those interested in the low-rank coals. Much European literature exists on the transformation of vegetable matter into coals of low-rank because such knowledge has been found useful in determining the utilization of such coals. It seems inevitable with the increasing relative importance of low-rank coals in this country that problems of initial coalification and early metamorphism will assume increasing importance. Undoubtedly solution of such problems will lead to a better understanding of the constitution of the higher rank coals.

**Geobotanical Investigations:** The use of plants and plant parts in fossil form in coal-bed correlation and the stratigraphic classification of coal measures has long been an established practice. In the past the evidence has consisted very largely of compressed fossil leaves, stems, and seeds found embedded in the shales and sandstones associated with the coal beds. Much information concerning the nature of the fossil plants has been obtained through study of coal balls in which plant material is found in a sort of petrified condition with the cellular parts only moderately compressed or entirely uncompressed. Within very recent years it has been demonstrated that the coal beds themselves provide fossils in the form of spores and pollen that in the case of Illinois coal beds, at least, can be used as type fossils for identifying coal beds. Possibly other fossils useful for identification may also be found in the coal. Such knowledge is of immediate practical importance where the bed being mined is crossed with faults and other beds of similar thickness are present above or below the bed being mined, or where explorations involve two or more closely spaced beds of similar thickness. Such geobotanical studies reveal the nature of the coal-producing flora and its variability. Studies that



have been made<sup>1</sup> point to the variable effect of the

<sup>1</sup>James M. Schopf: *Variable Coalification: The Processes Involved in Coal Formation*, *Econ. Geol.* (1948) 43, (3) 207-225.

coalification process upon the organic material so that it may be found that rank is not determinable by the same criteria for all kinds of coal. In connection with the botanical studies important contributions have been made and will continue to be made in regard to the plant material in coal and the morphological characteristics of plants obtained in a fossilized condition in coal balls.

**Coal Petrography:** Coal petrography or petrology is one of the later developments in the field of coal geology based very largely on the distinctions placed upon the different banded ingredients of coal by Dr. Marie Stopes in 1919.<sup>2</sup> The technique con-

<sup>2</sup>Marie C. Stopes. *On the Four Visible Ingredients of Banded Bituminous Coal*, *Proc. Roy. Soc. (London)* (1919) 99, 470-487.

cerns the composition of coal in a manner analogous to the procedures of rock petrology. It is based primarily upon the recognition of the presence of four varieties of banded ingredients visible with the naked eye: vitrain, clarain, durain, and fusain. Coal petrography and coal botany overlap because it is possible to describe the composition of coal not only in terms of the megascopically-different banded ingredients but also on the basis of the microscopically-different components representing plant parts or secretions which the writer has called phyterals. This overlapping has resulted in a good deal of difficulty in terminology and description. Nevertheless, terms have been fairly well defined and the informed technician can shift from the petrographic to the botanical or phytéral terminology without much confusion.

In describing a coal bed petrographically it is the usual procedure to note the distribution and proportion of the banded ingredients megascopically visible in a freshly broken, or cut and polished, column of the coal bed. Such studies give an idea of the composition of the bed petrographically in terms of vitrain, clarain, durain (splint), and fusain. The effect of variations in the proportion of the ingredients upon preparation, and eventual combustion, has been somewhat explored. Fusain is now recognized as the most important component of dust-forming material derived from the handling of mid-western coals. Vitrain in these coals undoubtedly has the greatest capacity for swelling and agglutinating, whereas durain (particularly splint durain) is noteworthy for its free-burning characteristics and lack of amenability to the process of hydrogenation. In fact, the capacity of a coal for hydrogenation seems to be largely determined by the amount of inert material with high carbon or low-volatile content, such as fusain and durain.

When a technique for the rapid analysis of broken coal in terms of the banded ingredients is developed, using unskilled or semiskilled labor, this method of analysis will doubtless have wider use if its practical value is also demonstrated. Progress in these directions is being made, but it must be admitted that with few exceptions, demand for such information has not developed. As noted above the fusain and the durain (splint) contents of coal have been regarded by some operators and some types of coal users as important and steps are taken to determine

the amounts of these ingredients in some coals. Petrographic analyses made by the Illinois State Geological Survey of different sizes of broken coal shows quite clearly that ingredient composition differs from size to size and differs from the average composition of the bed. It is evident, therefore, that the mining process alone or the mining and preparations processes together are selective with respect to the banded ingredients. It is obvious, therefore, that the amount of the various ingredients present in the coal bed will determine the proportion of the various sizes produced. In some mining establishments this might be a consideration of importance. In general it is probably true that ingredient composition is less important in the higher than in the lower rank coals. Only exploratory studies have been made along this line, but they seem to indicate the importance of knowing something about the effect of mining and preparation processes upon the petrographic characteristics of the various sizes of the coal produced if there is a complete understanding of these processes. This is particularly the case with respect to the coals of medium and low rank. Up to the present there has been very little investigation of the possible effect of petrographic variation upon utilization except for special uses like briquetting and hydrogenation. Certain coal types resulting from petrographic variation, for example, splint, semisplint and cannel coals, have long been recognized in the coal trade and are used frequently for special commercial applications. Two benches of a single coal seam sometimes are mined separately to take advantage of this. Also various sizes from the same mine may go to different uses because of considerable segregation of petrographic constituents according to size.

Petrographic analysis may eventually, however, discover the explanation for differences in the day-to-day variations in coal that has received the same preparation or for the differences between coals produced by the same methods from different mines. In some cases the cause of variations may lie deeper than "ingredient consist," in which case recourse to microscopic examination may reveal botanical or phytéral differences in composition of significant importance. This is obviously an appropriate field of investigation where concern is with the fundamental physical makeup of coal and will probably be of greater and greater importance as more and more low-rank coals enter the market. It is one which might be appropriately explored in academic laboratories, but as far as I am aware is not being so explored. Work of this kind is, I believe, confined to two laboratories in this country, but a third is in the process of organization here in Columbus by the U.S. Geological Survey and is to be placed under the direction of Dr. James M. Schopf.

**Coal Geochemistry:** There is a natural transition from coal petrography to coal geochemistry. Coal petrography reveals a variety of coal components more or less reminiscent of various organic entities initially present in the plant debris from which the coal was formed. As coal is produced from various organic substances by geological processes, it is reasonable to inquire concerning the nature of the geologic processes involved and how they acted upon the raw material to produce the results ob-



tained. The process of metamorphism involves the application of pressure, more or less heat, one or both applied through a period of time; but it also involves the intimate events that transpire in the organic molecules subjected to the vicissitudes of the geological environment. The problem is that of determining the specifications of the environmental conditions and also of determining the reaction of the organic substances to these conditions. One of the natural consequences of a familiarity with the botanical and petrographic heterogeneity of the middle and low-rank coals is inquiry in regard to the part played by geological processes in producing, perpetuating, or eliminating such heterogeneity.

The failure to carry on such an investigation lies wholly with the geologists. The chemists cannot justly be taken to task for failure to conduct geological research. However, no such research can be carried on without the cooperation of chemists and particularly of chemists who have an understanding of and interest in geological phenomena and principles.

It is certain that for the indefinite future coal will be investigated predominantly by the standard procedures of the commercial types of analysis, by the application of various, carefully standardized tests for determining directly the adaptability of coal to various uses, by carefully developed procedures of solvent analysis, and by the conventional methods of organic analysis. A vast amount of extremely useful information will be added to that already existing. This information is of much value to geologists, for it is significant of geological phenomena in spite of its predominant empirical character. Because rank differences are fundamental with respect to coal, items of fact of a wide variety in regard to the properties of coal, even though empirically determined, can be fitted into the picture, and, in turn, be used empirically for description and classification purposes.

In the matter of coal chemistry, coal geologists have been short on a number of counts but I wish to point out two particularly: First, they have not succeeded in establishing the importance of the initial variability of coal-forming material to the extent of demonstrating its significance even in the middle and low-rank coals, where, it is probable such demonstration would be most readily possible. Second, they have not produced tangible criteria for evaluating the geological elements of the environment effecting metamorphism so that they can be used by the physical and structural chemist as a basis for theoretical consideration of their effectiveness in bringing about fundamental molecular changes in the components of coal. It is possible that these faults lie largely in the fact that American geologists interested in coal started at the wrong end of the coal series, where petrographic differences are of less definite significance. Whereas, if the start had been made in the low-rank coals, coal petrography and coal botany could have readily demonstrated the importance of coal heterogeneity and the definite relationship of component variability to variations in the coal. The changes in these components resulting from coalification could then have been followed through to the higher rank coals with the gradual tapering off of the influence of petrographic variability. The nature and import-

ance of petrographic variations in coal would then probably have been better and more realistically understood.

The nature of coal geology has been described in brief outline. The various fields of study and research in many ways touch closely and not uncommonly considerably overlap the fields of coal-mining engineering and coal preparation and utilization. Research in those aspects of coal geology that impinge closely upon the practical fields of engineering in its various phases, such as the delineation and description of coal resources, description of various geological conditions affecting ease and cost of mining, studies in roof geology, and so on, has been fairly well supported particularly by Federal and state agencies. Even the somewhat more academic fields of coal botany and coal petrography occasionally contribute information of practical significance, particularly in the identification of beds and the interpretation of structural irregularities and in some phases of coal preparation and utilization. There is considerable activity in some of these fields and some activity in all, although not nearly as much as might be expected in view of the fact that, in general, colleges and universities, even those located in coal mining regions, give little or no attention to coal geology, particularly to research in this field.

The outstanding neglected field is that of coal geochemistry in spite of its obvious importance to an understanding of the nature of coal. It is primarily for the encouragement of research in this particular field that the Committee on Coal Research of our sister Society of Economic Geologists has declared itself. It appeared to this committee that the outstanding need in the field of coal geology and in the study of coal in general was fundamental research into the constitution of coal in which geologists and chemists would join as a geochemical project. It seemed to the Coal Research Committee, several of whom are members of the Institute, meeting in Pittsburgh in March 1949, in view of the fact that the interests of the two organizations overlap fairly widely, that it would be appropriate to make a statement to the Coal Division of the Institute in regard to the activities of the Coal Research Committee of the sister society, particularly with respect to its proposal for fundamental research. We are asking for no action on the part of the Coal Division. Should any of you be personally interested in the proposal and care to hear more about it, information will be supplied upon request. If you would like to be represented on a coal research committee that is endeavoring to build up general interest in coal geology, you are welcome to join our group, as there are no prerequisites aside from such interest.

I might say that the activities of the Committee extend somewhat beyond that of suggesting and promoting research of a fundamental sort, in that we are endeavoring to obtain wider recognition of coal geology in the programs of scientific and technical societies, and to encourage the inauguration of courses in coal geology in the colleges. To these ends a symposium on low-rank coals of the west was organized to be held at the time of the SEG and GSA joint meetings at El Paso in November 1949.



# Ground Water in California

by J. F. Poland

Annual pumpage of ground water from the alluvial valleys of California now is about 10 million acre-feet. This heavy pumpage has created problems of over-draft and ocean-water encroachment in many valleys. Steps are being taken to protect and increase the available supply but much additional basic study is needed.

**Location of Basins and Geologic Features of Occurrence:** The major ground-water resources of California occur and are stored in the many large alluvium-filled valleys of the state. The deposits of Quaternary age which hold the ground water commonly are called valley fill. Meinzer<sup>1</sup> has discussed the characteristic features of this fill and showed that for western United States its distribution is concentrated chiefly in western Utah, Nevada, central Oregon, California, southern Arizona, and central New Mexico (see fig. 1). For California the

---

J. F. POLAND is District Geologist, Ground Water Branch, Geological Survey, U. S. Department of the Interior, Sacramento, Calif.

Paper published with the permission of the Director of the Geological Survey, U. S. Department of the Interior.

AIME San Francisco Meeting, February 1949.

TP 2746 H. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received May 23, 1949.

---

principal ground-water basins are shown on fig. 2. Except for the Mojave Desert and the "Basin and Range" Valleys (fig. 2, Nos. 23 and 23A), which are not shown individually, the basins outlined on fig. 2 contain all the large reservoirs in valley fill within the state.

The Sacramento and San Joaquin Valleys (fig. 2) together comprise the Great Central Valley of California, which has an area of about 18,000 square miles. The Sacramento Valley contains about 5000 square miles of valley lands, approxi-

mately equal to the area of Connecticut; the San Joaquin Valley embraces about 13,000 square miles, equivalent to the areas of Massachusetts and Connecticut combined. Both the area and the ground-water use in the Central Valley is far greater than for all the other shaded alluvial basins of fig. 2 added together. Except for the Sacramento Valley, all the major ground-water basins of the state are south of Sacramento.

The valley fill in these ground-water basins has been transported by streams and most of it has been laid down in alluvial fans, producing the typical alluvial-fan structure of aquifers radiating as tongues from a prismatic pile of coarse, permeable debris at the canyon mouths; these aquifers are encased down slope by poorly sorted materials consisting of a clayey matrix with imbedded pebbles and boulders, interbedded with flood-plain silt. In the deeper inland structural valleys, the alluvial deposits interfinger at depth in the valley bottoms with horizontally bedded lacustrine sand, silt, and clay. The San Joaquin Valley is a good example. In the coastal valleys exposed to the sea, such as the Los Angeles coastal plain, the alluvial fans pass coastward into lagoonal and thence to shallow marine deposits. In all these basins, the gravel is coarsest and most permeable near the apexes of the fans, but tongues or lenses of clean gravel and sand may extend for many miles down the fan. Thus, these water-bearing structures are in a position to absorb stream flow near the canyon mouths and to transmit it through the buried radiating conduits.

Especially in the valleys of southern California, the structure is complicated by faults which usually extend upward through the deposits of Pleistocene age. Where these faults cut impermeable confining beds, they have initially afforded passage for upward escape of artesian waters. The ascending waters undergo pressure decrease and, because the ground waters of California are characteristically of the calcium-bicarbonate type, carbon dioxide has escaped, causing precipitation of calcium carbonate and other salts together with silica. In this manner ground-water dams have developed even in wholly unconsolidated and highly permeable deposits. Gouge is believed to have been a negligible factor in the formation of these barrier features. In many of the basins of southern California these ground-water dams are primary factors in controlling the quantity of recharge and the position of the water level (and hence the pumping lift). For basins in the process of full development, these may be

**Fig. 1—Map of western part of United States showing principal areas underlain by Quaternary valley fill.**  
(After Meinzer.)



of primary consideration in determining the most effective utilization.

#### **Quantity Pumped and Uses:**

Ground-water development in California began in the Los Angeles area a few years after the end of the Civil War, about 1870. By 1900 approximately 10,000 wells had been put down within the four major valleys that comprise the South Coastal Basin.<sup>2</sup> According to Simpson<sup>3</sup> irrigation north of the Tehachapi Mountains did not become significant until after 1900. By the first World War, in the central as well as the southern part of the state, most of the surface water available from the unregulated streams for irrigation in the dry summer months had been diverted for use. Consequently, most of the development since then has been from the ground-water basins. This has been spurred by the great improvement in turbine pumps and the wide availability of electric power, as well as by the expanding population and the increasing demand for agricultural products.

The total amount of ground water pumped yearly from California's underground basins is not known accurately. However, the pumpage from all the basins outside the Central Valley is estimated to be approximately 2 million acre-feet (1 acre-foot = 325,851 gal). For the San Joaquin Valley, the combined gross pumpage of ground water from about 35,000 wells south of the Merced River for

the one-year period from April 1, 1947, to April 1, 1948, has been estimated by the California Division of Water Resources from electrical energy consumed as close to 6 million acre-feet.<sup>4</sup> Of this, about one million acre-feet is pumped from the deep wells on the west side of the valley. Pumpage in the part of the San Joaquin Valley north of the Merced River and in the Sacramento Valley together is believed to be nearly two million acre-feet. Thus, the total gross pumpage in the state in the year 1947 to 1948 apparently was between 9,000,000 and 10,000,000 acre-feet. Of this total at least three-quarters was pumped from wells in the

Central Valley. These figures suggest that the use of ground water in the state now is roughly equal to that of surface water, including Colorado River water. The pumpage is concentrated almost wholly in the months of little or no rainfall, from April to October. If uniformly distributed throughout the year, the discharge from all well pumps would be equal to about 13,000 second-feet, or about 8.5 billion gallons a day.

A comparison of current ground-water pumpage in California with the flow of main-stem streams and the storage capacity of large surface reservoirs is of use in visualizing its magnitude of draft. For example, the average gauged runoff of the Sacramento River at Red Bluff for the period 1896 to 1939 (water years) is 8,111,000 acre-feet a year.<sup>5</sup> At the other end of the state, the flow of the Colorado River is of critical interest. Morris<sup>6</sup> states



that for the past 50 years the average flow of the Colorado River at the Mexican border has approximated 17,720,000 acre-feet, under assumed virgin conditions. Thus, the estimated current ground-water draft in California is 10 to 20 pct higher than the 44-year average yearly discharge of the Sacramento River at Red Bluff and slightly more than half as great as the 50-year average runoff for the Colorado River.

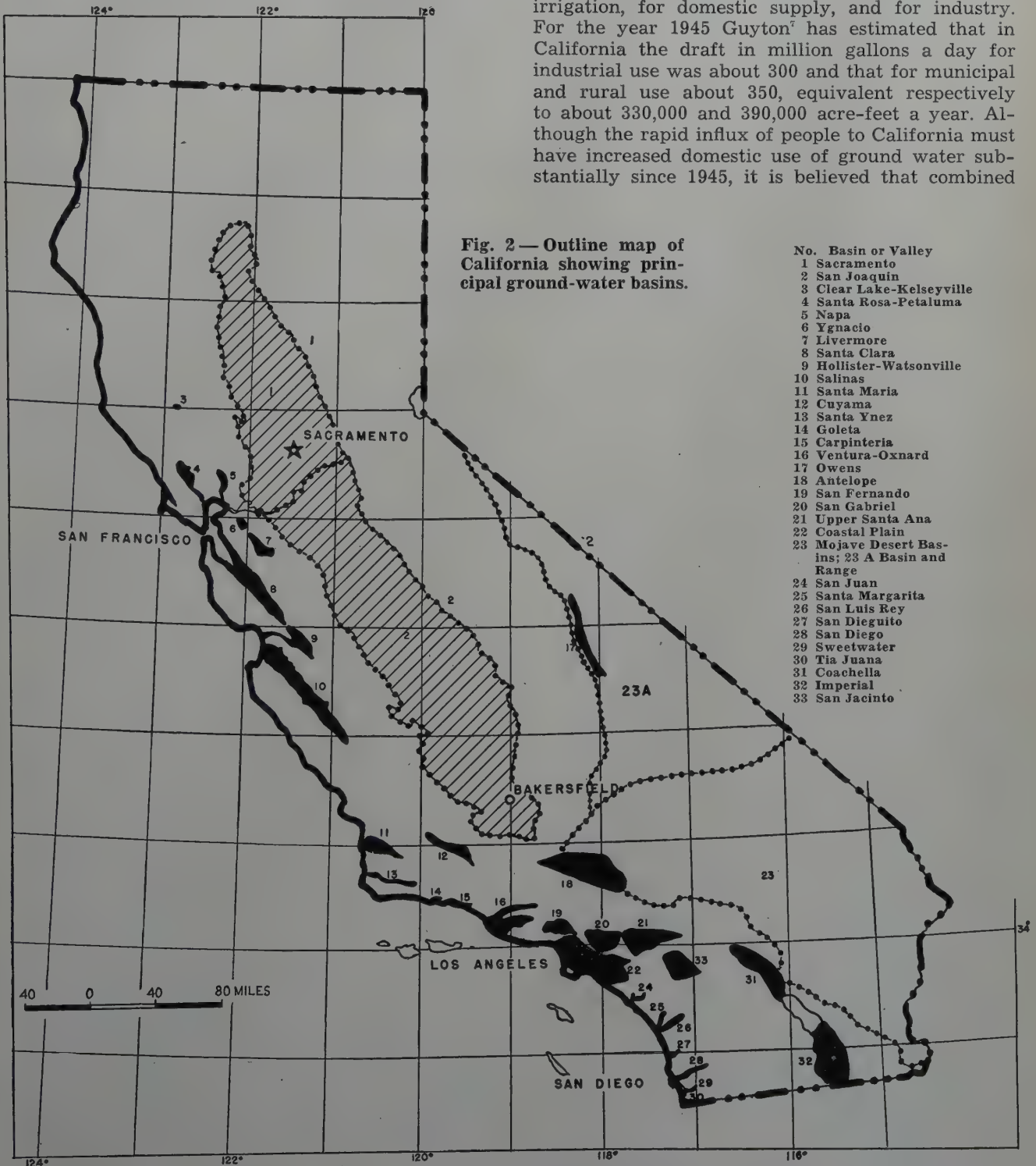
Comparison with the capacity of surface-water reservoirs affords an instantaneous and simpler picture. For example, the current annual pumpage of ground water is about twice as great as the joint

storage capacity of the reservoirs formed by Shasta and Friant Dams.

For the total gross pumpage of ground water, the distribution by major ground-water basins is approximately as follows: Central Valley, at least 75 pct; South Coastal Basin, 8 pct; Salinas Valley, 4 pct; Santa Clara County basins, 3 pct; Ventura County basins, 2 pct; Santa Barbara County basins, 1.5 pct; Antelope Valley and Coachella Valley, each about 1.3 pct; and miscellaneous small basins 1 to 2 pct.

The principal uses of ground water in California are threefold: in order of importance, they are for irrigation, for domestic supply, and for industry. For the year 1945 Guyton<sup>7</sup> has estimated that in California the draft in million gallons a day for industrial use was about 300 and that for municipal and rural use about 350, equivalent respectively to about 330,000 and 390,000 acre-feet a year. Although the rapid influx of people to California must have increased domestic use of ground water substantially since 1945, it is believed that combined

Fig. 2 — Outline map of California showing principal ground-water basins.



domestic and industrial use still is somewhat less than one million acre-feet a year. Evidently at least 90 pct of the ground water pumped is used for irrigation.

The pumpage of ground water in all parts of the United States has increased greatly in the past decade. In 1945, according to Guyton,<sup>7</sup> the average daily use of ground water was about 20 billion gallons, equivalent to 22 million acre-feet for the year, and about twice the draft in 1935.<sup>8</sup> Doubtless the country-wide draft has increased substantially since 1945, but even if it has risen as much as a third, the draft in California is now about one-third of the total. Arizona is second in rank with pumpage of ground water in 1947 estimated by Turner<sup>9</sup> as on the order of 2.6 million acre-feet.

#### **Problems Created by Intensive Development:**

The long-continued and increasing demands on the ground-water supplies of the state have created serious problems in many of the ground-water basins. In some of the basins, water levels have been drawn down many scores and even hundreds of feet; in a few coastal basins the drawdown has pulled water levels many feet below sea level. The principal problems created by this great lowering of water levels and the application of large quantities of ground water and surface water to irrigation of the agricultural lands are those of (1) overdraft, (2) salt-water encroachment, (3) gradual changes in the chemical and physical character of the soils, and (4) long-term changes in the chemical character of ground waters. Only the first two are considered here. Protection of the ground-water supplies from pollution by industrial wastes has become a very important problem in recent years, owing to the great increase in industrial plants and somewhat indiscriminate disposal of wastes. That problem, which is now under investigation by several state agencies, has developed primarily because of rapid industrial expansion.

**Overdraft:** Prolonged and progressive drawdown of water level in a ground-water basin suggests an overdraft, that is, that the average net draft is exceeding the average replenishment. Such a condition is more noticeable if the depth to water is rapidly approaching the economic limit of pumping or if the drawdown is dewatering a substantial part of the saturated thickness of water-bearing beds.

In California, the problem of overdraft is complicated by the distribution of rainfall, which is the most basic factor in recharge. Not only is the rainfall concentrated almost wholly in the months from October to May, but also it varies greatly from year to year and has a rude cyclical distribution, with periods of many years in which the average rainfall is deficient and others in which it is above normal. For this reason both recharge and draft fluctuate not only from year to year but from period to period. For the most completely developed ground-water basins of the state, those in the Los Angeles area, problems of overdraft and the complications caused by this rude cyclical distribution of rainfall have been analyzed in an exhaustive report by Gleason.<sup>10</sup>

An apparent overdraft may be wiped out by

great replenishment in a very few excessively wet years. For example, water levels in wells in the San Gabriel Basin (fig. 2) above Whittier Narrows were drawn down more than 100 ft during the two decades from 1916 to 1936, which was a period of deficient rainfall. Toward the end of this period it was considered by some that an overdraft existed. However, during the wet period from 1936 to 1945, the rainiest 10-year period on record for the Los Angeles area, the excessive runoff and ground-water recharge refilled the basin and restored water levels to those of 1916. The prolonged drawdown of the water table and the long-continued use of stored water prior to 1937 made available a great underground reservoir for retention of subsequent recharge. In this case, haphazard exploitation produced a classic example of the advantages and use of a ground-water basin for cyclic storage of water.

In at least a dozen areas in California a condition of real overdraft now exists. The largest is the San Joaquin Valley. Here the areas of overdraft on the east side are chiefly south of the Kings River recharge area, south of Hanford and Visalia. In this area the overdraft probably is within the range of half a million to a million acre-feet a year. On the west side of the valley, for the 150-mile strip from Mendota to the south end, development of ground water has taken place largely since 1935 and has been most rapid since the early forties. Here the pumpage is now on the order of a million acre-feet and probably more than half of the water is being mined. Compared to the discharge of Sierra streams to the east, the flow of the west side streams is very low and recharge is small.

Other large basins of the state which are now reported as or known to be overdrawn are the Santa Clara Valley fringing San Francisco Bay;<sup>11</sup> the lower east side of the Salinas Valley;<sup>12</sup> the Santa Maria Valley;<sup>13</sup> the Ventura-Oxnard area;<sup>14</sup> the Antelope Valley;<sup>15</sup> the Upper Santa Ana Valley and the main and West Basins of the coastal plain, which are in the South Coastal Basin.<sup>10</sup>

**Salt-water Encroachment:** *Ocean-water Intrusion:* Several of the major and at least a dozen of the minor ground-water basins of California border the coast and their seaward margins are in contact with or extend beneath the Pacific Ocean or its landward salients such as San Francisco Bay. Geologic and hydrologic evidence accumulated to date show that in these basins the seaward ends of the aquifers are in hydraulic contact with the ocean. Under natural conditions of seaward hydraulic gradient, fresh ground water was discharged to the sea or to its inland extensions. In places along the coast where heavy pumping has drawn down the water level below sea level, the hydraulic gradient has been reversed and ocean water has invaded the aquifers either directly or through defective wells.

In this way, salt water has invaded to some degree ground-water basins in the following areas: the Sacramento-San Joaquin delta between Antioch and Pittsburg; the Napa Valley; the Santa Clara Valley from Oakland on the east to San Mateo on the west;<sup>16</sup> the mouth of the Salinas Valley;<sup>17</sup> the Ventura area; the coastal plain of Los Angeles and



Orange Counties, both in the West Basin,<sup>17</sup> and in the main coastal basin;<sup>18</sup> and several of the small coastal basins of San Diego County,<sup>19</sup> especially the Tia Juana Valley.\* The most serious invasion to date

\* Ground-water supply of Tia Juana Valley in San Diego County is now under adjudication and problems of salt-water intrusion are being investigated by the California Division of Water Resources, under the direction of G. B. Gleason.

is that in the coastal plain of Los Angeles and Orange Counties. In the West Basin between Redondo Beach and Santa Monica ocean water is advancing inland along a 10-mile front and has progressed as much as two miles from the coast; in the main coastal basin ocean water has advanced inland as much as two miles in Santa Ana Gap of Orange County, and the potential threat to the Orange County supply is great.

*Interior Contamination:* In several basins, saline contamination has developed by invasion of interior saline waters apart from the ocean. For example, in the vicinity of Marysville in the so-called peach bowl of the Sacramento Valley it is reported<sup>20</sup> that at places excessive drawdown of the water level in wells has brought up saline water from below. Also on the west side of the San Joaquin Valley in the vicinity of Vernalis and to the south, beds containing saline waters locally underlie the fresh-water aquifers. Wells which penetrated too great a thickness of water-bearing beds have had to be plugged so as to seal off the undesirable bottom waters. Locally in the Salinas Valley,<sup>21</sup> intensive drawdown of the pressure surface in confined aquifers has permitted saline shallow waters to move down and contaminate the fresh waters beneath, either through defective well casings or by direct leakage through the confining beds.

**Methods for Increasing or Protecting the Available Supply:** *Importation and Recharge for Cyclic Storage:* Overpumping of a ground-water basin and progressive lowering of the water level inevitably will develop if the average net long-term discharge, both natural and artificial, is greater than the average net recharge. There are only two possibilities for decreasing or eliminating an overdraft: (1) decrease the net discharge, or (2) increase the recharge.

In the early stages of development, general regional drawdown of the water level in a basin may substantially decrease natural discharge, both subsurface outflow and discharge by plants. After this has been accomplished, however, the only means of reducing net discharge still further is by decreasing pumping draft. This never will be a popular action and probably never would succeed unless supported by legal authority or enforced by economic necessity.

Fortunately, the recharge ordinarily can be increased substantially by conservation measures, by catching the runoff in detention reservoirs and spreading it on permeable deposits so that it will go into storage in the ground-water basins. In some basins, however, it is too costly to retain much of the flash runoff in surface retention dams, and in a few insufficient runoff is available. In such cases, importation of water is the only means for supplementing the ground-water supply.

In southern California importation of water from the Colorado River in quantities much larger than the gross ground-water draft of that area is taking care of many of the expanding needs for the present and immediate future, except for basins like the Antelope Valley, which has no immediate prospects for amelioration of its serious overdraft.

The water deficiency in the San Joaquin Valley, which existed as early as 1921, was one of the major incentives for the State Water Plan investigations which were begun in that year. The Central Valley Project has developed from these investigations. As summarized by Hyatt.<sup>22</sup>

This plan included the creation of a large storage reservoir, on the Sacramento River at the head of the valley near Redding, which would regulate the flow of the river, provide flood control, and generate a large amount of hydroelectric power. Waters released from this reservoir during the summer would flow down the Sacramento River, eliminate irrigation deficiencies and assure navigation in that valley throughout the year, overcome the encroachment of salinity in the Sacramento-San Joaquin Delta, and provide sufficient additional water for diversion southward by means of pumps into the San Joaquin Valley. . . . A second reservoir was also planned on the San Joaquin River at the town of Friant, north of Fresno, with large canals leading north and south to supply the distressed areas whose problems were the main reason for the preparation of the plan.

It was concluded for the San Joaquin Valley that the flood runoff could not all be conserved economically in surface reservoirs and that irrigation requirements during years of deficient rainfall could be supplied only by taking ground water from storage.<sup>23</sup> As reported by Livingston,<sup>24</sup> it was pointed out<sup>25</sup> that an assured water supply for the southern half of the east side of the Valley requires the surplus runoff of wet years to be detained underground in cyclic storage for withdrawal and use in dry years. The volumes of ground-water storage required for use in dry periods is very large, as much as 16,000,000 to 18,000,000 acre-feet. It has been assumed that this water can be put underground in a few wet years by excess irrigation, assisted by percolation from stream channels, canals, and ditches.

Little is known as yet about the structure of the water-bearing deposits in the San Joaquin Valley and about the movement and circulation of ground water in the aquifers that lie within that structure. The Central Valley Project is now under construction by the Bureau of Reclamation. The success of that project as it affects the San Joaquin Valley depends fundamentally on the ability to put underground this cyclic storage at rates of as much as 3,000,000 acre-feet or more a year. So far investigators of the Bureau of Reclamation and other agencies have obtained only a part of the basic information on ground-water conditions and on the other factors affecting the practicability of artificial recharge in the large volumes contemplated.

In connection with the problems of overdraft and need for additional water in many of the ground-water basins of the state, a reappraisal of the overall water resources is now under way by the State Water Resources Board through the Division of

Water Resources. The primary purpose is to determine where surplus waters exist, how great they are, and how they can be utilized to best advantage in areas of deficient supply.

**Restraint of Saline Encroachment:** In ground-water basins where no general overdraft exists, drawdown of water levels along the coast may cause a relatively minor intrusion of ocean water that will not pass inland appreciably beyond the area where there is a landward gradient. Such apparently is the case in the Salinas Valley.<sup>12</sup> On the other hand, if a basin is pumped so heavily that there is a basin-wide depression of water levels below sea level, ocean-water intrusion can continue to march steadily across the basin so long as the water levels remain depressed. This is the situation in the West Basin of Los Angeles County.<sup>17</sup> In most ground-water basins, such a condition could be remedied most effectively in the long run by restoring water levels throughout the basin to a sufficient height that the salt water would be displaced seaward. The West Basin is anomalous, however, in that about half its current recharge is derived by underflow through the barrier faults of the Newport-Inglewood uplift, its inland boundary. The recharge by underflow continues only because there is a water-level differential across the faults ranging from 30 to 60 ft, with water levels on the inland side about at sea level and those in the West Basin 30 to 60 ft lower. Thus, if levels on the coastal side (within the West Basin) are raised to or above sea level, this recharge will cease. Therefore, to utilize the basin effectively, it is possible that artificial recharge could be employed to build a fresh-water ridge near the coast and pumping could be concentrated along the inland edge so as to maintain the favorable water-level differentials and the recharge.

Restraint of ocean-water intrusion by artificial recharge and the development of coastal fresh-water ridges ultimately may prove economically feasible for several ground-water basins along the coast. Geologic conditions will determine whether this can be accomplished by spreading water at the land surface for percolation from trenches, pits, or basins, or by the more expensive methods of recharge through wells. An engineering analysis based on the facts would determine whether such measures are economically feasible.

Artificial recharge has been practiced in California for nearly 50 years. Most of the operations, for obvious reasons, have been concentrated at the permeable intake areas at or near the heads of the alluvial fans. In the future, the steps which are taken to increase or protect the supplies of ground water in California doubtless will result in a great increase in the quantities of water placed in cyclic ground-water storage, and necessarily will require much research on methods and techniques of artificial recharge. As a prerequisite to effective progress, the geology and hydrology of the critical ground-water basins should be examined in great detail.

#### References

<sup>1</sup> O. E. Meinzer: The Occurrence of Ground Water in the United States. U. S. Geol. Survey, *Water-Supply Paper* 489, pp. 291-303, fig. 100.

<sup>2</sup> W. C. Mendenhall: U. S. Geol. Survey, *Water-Supply Papers* 137, 138, 139, 142 (1905); also 219 (1908).

<sup>3</sup> T. R. Simpson: Ground Water and Water Rights in California. Calif. Div. Water Resources. Typewritten Report (January 1949) 17 pp.

<sup>4</sup> *Idem.* p.1.

<sup>5</sup> U. S. Geol. Survey, *Water-Supply Paper* 881 (1940) 410.

<sup>6</sup> S. B. Morris: The Colorado River. *Jnl. Amer. Water Works Assn.* (1947) 39 (10) 953.

<sup>7</sup> W. F. Guyton: Industrial Use of Ground Water in the United States. Unpublished paper presented before the Geol. Soc. of Washington, Washington, D. C., 1945.

<sup>8</sup> A. N. Sayre and V. T. Stringfield: Artificial Recharge of Groundwater Reservoirs. *Jnl. Amer. Water Works Assn.* (1948) 40 (11) 1152-1158.

<sup>9</sup> S. F. Turner and others: Pumpage and Ground-water Levels in Arizona in 1947. U. S. Geol. Survey. Mimeographed Report, 3 pp., 1948.

<sup>10</sup> G. B. Gleason: South Coastal Basin Investigation; Overdraft on Ground-water Basins. Calif. Dept. Pub. Works, Div. Water Resources. *Bull.* 53 (1947) 256 pp.

<sup>11</sup> G. W. Hunt: Report to the Honorable Board of Directors of the Santa Clara Valley Water Conservation District on Proposed Lexington Dam and Water Conservation Works. Published Report, 29 pp. August 1947.

<sup>12</sup> T. R. Simpson: Salinas Basin Investigation. Calif. Dept. Pub. Works, Div. Water Resources. *Bull.* 52 (1946) 230 pp.

<sup>13</sup> G. F. Worts, Jr.: Geology and Ground-water Resources of the Santa Maria Valley Area, Santa Barbara County, California. U. S. Geol. Survey. Duplicated Report (1948) 235 pp.

<sup>14</sup> Harold Conkling: Water Supply of Santa Clara Water Conservation District, Ventura County. Typewritten report to the Board of Directors of the Conservation District (Nov. 1947) 81 pp.

<sup>15</sup> G. B. Gleason: Report to the Assembly of the State Legislature on Water Supply of Antelope Valley in Los Angeles and Kern Counties. Calif. Dept. Pub. Works, Div. Water Resources. Mimeographed Report (May 1947) 22 pp.

<sup>16</sup> C. F. Tolman and J. F. Poland: Ground Water, Salt-water Infiltration, and Ground-surface Recession in Santa Clara Valley, Santa Clara County, California. *Trans. Amer. Geophys. Union* (1940) pt. 1, p. 24, fig. 1.

<sup>17</sup> J. F. Poland, A. A. Garrett and Allen Sinnott: Geology, Hydrology, and Chemical Character of the Ground Waters in the Torrance-Santa Monica Area, Los Angeles County, California. U. S. Geol. Survey. Duplicated Report (1948) 472 pp.

<sup>18</sup> A. M. Piper and A. A. Garrett: Chemical Character of Native and Contaminated Ground Waters in the Long Beach-Santa Ana Area, California. U. S. Geol. Survey. Duplicated Report (1946) 354 pp.

<sup>19</sup> A. J. Ellis and C. H. Lee: Geology and Ground Waters of the Western Part of San Diego County, California. U. S. Geol. Survey, *Water-Supply Paper* 446 (1919) 321 pp.

<sup>20</sup> R. Simpson: Oral Communication.

<sup>21</sup> Pp. 136-140 of ref. 12.

<sup>22</sup> Edward Hyatt: Water Development Plans in California. *Jnl. Amer. Water Works Assn.* (1947) 39, (5) 430-442.

<sup>23</sup> A. D. Edmonston and others: San Joaquin River Basin. Calif. Dept. Pub. Works, Div. Water Resources. *Bull.* 29 (1931).

<sup>24</sup> Penn Livingston: Ground-water Features of the San Joaquin Valley, California: A Review of Published and Unpublished Reports and Papers. U. S. Geol. Survey (1944) 48 pp. (Prepared for the U. S. Corps of Engineers.)



# Effect of Freight Rates on Marketing

Northwest Industrial Minerals

by Leslie C. Richards

The competitive position of producers of industrial minerals depends upon the delivered price of their product. Freight charges are a major factor in the sales to consumers. A comparison of freight rates on movements of limestone and silica rock in the Pacific Northwest shows a wide variance in mills per ton mile for like hauls, thereby favoring some localities over others.

**Introduction:** Sooner or later the potential miner of industrial minerals reaches the conclusion that of the various factors he must consider, the marketing of his product is the most important. The sooner he realizes this the better, as successful industrial mineral producers will confirm. Herein lies a difference between industrial mineral mining and metal mining. Often in the latter, technical problems of mining or concentrating are equal to or even surpass those of marketing. This is not meant to imply that there are no problems in mining and beneficiation of industrial minerals, but these usually are outweighed by the problem of marketing.

Recently, an official of a nationally known corporation in the electro-metallurgical industry made

must be able to deliver to the market at a competitive price. It must be remembered that his largest cost factor is rail transportation, and that he is limited to markets that he can reach competitively.

Since the cost of rail transportation is so important, let us compare various freight rates in the Northwest and attempt to determine their influence on marketing. A study of all rates on all industrial minerals transported in this area, is beyond the scope of this paper. For the purposes of comparison, let us consider limestone (broken, crushed, or ground), and silica rock. These are widely used

LESLIE C. RICHARDS, Member AIME, is a Mining Engineer, Portland, Oregon.

Third Annual Northwest Industrial Minerals Conference sponsored by the Columbia Section in cooperation with the Oregon and the North Pacific Sections, AIME, May 1949.

TP 2781 H. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received May 26, 1949.

an industrial mineral survey in the Pacific Northwest. In the course of his investigation he interviewed producers and potential producers of the various industrial minerals scheduled for use in the plant now being constructed in this area. He stated that he was interested in only three factors: namely, (1) Can your product meet our specifications? (2) Are you able to supply our tonnage requirements? and (3) What will be the delivered price? If there are several sources of supply of sufficient grade and tonnage, as is often the case in the Northwest, then the third factor becomes the important and deciding one.

While industrial minerals are usually sold f.o.b. cars with the freight being paid by the buyer, it is the delivered and not the f.o.b. price that determines the competitive position of the producer and his industrial mineral product. Unfortunately for both the consumers and producers of industrial minerals in the Northwest, the sources are generally a considerable distance from the markets. This necessitates long rail hauls, for which transportation charges make up the largest share of the delivered cost and the delivered price of the product.

Is it any wonder then, that rail rates play such an important part in the marketing of industrial minerals? In order to sell his product, the producer

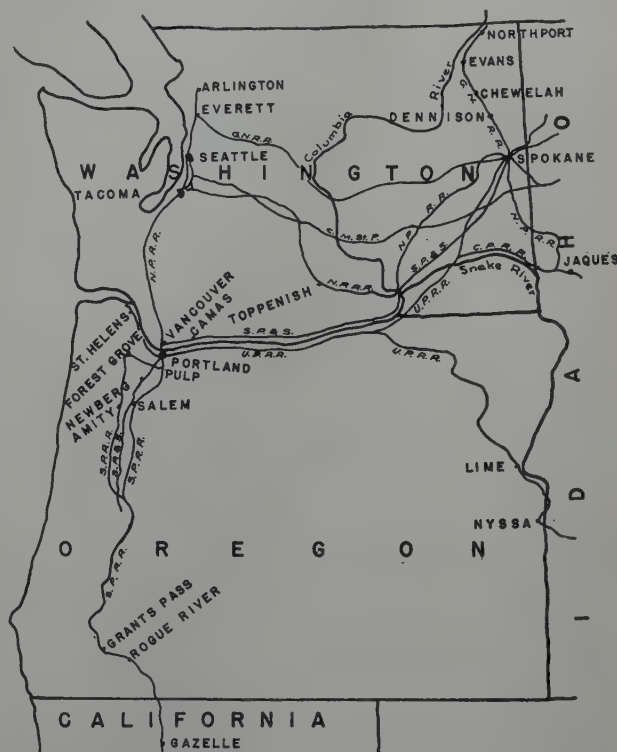


Fig. 1—Railroads that serve Washington and Oregon.

industrial minerals, and are carried by several different rail routes from the sources of supply to the centers of consumption in the Pacific Northwest.

**Markets and Sources of Supply:** Limestone has a number of uses in western Oregon and Washington. Among them are agricultural lime for the soils of the Willamette Valley and southwestern Washington; paper rock for the mills in such cities as West Linn, Oregon City, Newberg, St. Helens, and Salem, Oregon, and Camas and Everett, Wash.; chemical and metallurgical-grade limestone for the metallurgical plants centering around Portland, and the Tacoma-Seattle areas; and sugar rock for the sugar refineries at Toppenish, Wash., and Nyssa, Oregon. Since limestone for cement manufacture is generally processed near the quarry, transportation

Table I. Railroad Rates on Limestone, Pacific Northwest<sup>a</sup>

To	From	Use	Miles	Tariff	Minimum Carload, Lb	Railroads	Mills Per Ton Mile
Amity, Ore.	Gazelle, Calif.	Ag. lime	344	\$3.45	100,000	SP	10.0
Amity, Ore.	Grants Pass, Ore.	Ag. lime	231	2.90	80,000	SP	12.5
Salem, Ore.	Gazelle, Calif.	Ag. lime and paper rock	357	3.45	100,000	SP	9.7
Salem, Ore.	Grants Pass, Ore.	Ag. lime and paper rock	244	2.90	100,000	SP	11.9
Salem, Ore.	Jaques, Ida.	Ag. lime and paper rock	426	4.50	100,000	Camas P. RR, UP, SP	10.6
Salem, Ore.	Northport, Wash.	Ag. lime and paper rock	557	4.45	100,000	GN, SPS, SP	8.0
Salem, Ore.	Evans, Wash.	Ag. lime and paper rock	534	5.05	100,000	GN, SPS, SP	9.4
Salem, Ore.	Arlington, Wash.	Ag. lime and paper rock	300	4.13	60,000	NP, SP	13.8
Oswego, Ore.	Lime, Ore.	Ag. lime	394	2.87	5 cars	UP, SP	7.3
Forest Grove, Ore.	Gazelle, Calif.	Ag. lime	448	3.85	100,000	SP	8.6
Forest Grove, Ore.	Grants Pass, Ore.	Ag. lime	335	2.95	100,000	SP	8.8
Forest Grove, Ore.	Jaques, Ida.	Ag. lime	411	5.40	100,000	Camas P. RR, SP	13.1
Vancouver, Wash.	Gazelle, Calif.	Ag. lime and chemical grade	420	5.70	100,000	SP, UP	13.6
Vancouver, Wash.	Grants Pass, Ore.	Ag. lime and chemical grade	307	3.50	100,000	SP, UP	11.4
Vancouver, Wash.	Lime, Ore.	Ag. lime and chemical grade	385	3.25	80,000	UP	8.4
Vancouver, Wash.	Jaques, Ida.	Ag. lime and chemical grade	373	3.25	80,000	Camas P., UP	8.7
Vancouver, Wash.	Northport, Wash.	Ag. lime and chemical grade	496	3.85	80,000	GN, SPS	7.8
Vancouver, Wash.	Evans, Wash.	Ag. lime and chemical grade	471	3.85	80,000	GN, SPS	8.2
Vancouver, Wash.	Arlington, Wash.	Ag. lime and chemical grade	237	2.25	60,000	NP	9.5
Vancouver, Wash.	Orcas Island	Ag. lime and chemical grade		3.85	60,000	Barge and NP	
Pulp, Ore.	Gazelle, Calif.	Paper rock	392	3.85	100,000	SP	9.8
Pulp, Ore.	Grants Pass, Ore.	Paper rock	279	2.90	80,000	SP	10.4
Pulp, Ore.	Jaques, Ida.	Paper rock	391	3.85	100,000	Camas P. RR, UP, SP	9.8
Pulp, Ore.	Northport, Wash.	Paper rock	522	4.45	100,000	GN, SPS, SP	8.5
Pulp, Ore.	Evans, Wash.	Paper rock	499	4.45	100,000	GN, SPS, SP	8.9
Pulp, Ore.	Arlington, Wash.	Paper rock	265	3.58	60,000	NP, SP	13.5
Newberg, Ore.	Gazelle, Calif.	Paper rock	380	3.85	100,000	SP	10.1
Newberg, Ore.	Grants Pass, Ore.	Paper rock	267	2.90	80,000	SP	10.9
Newberg, Ore.	Jaques, Ida.	Paper rock	393	4.05	100,000	Camas P. RR, UP, SP	10.3
Newberg, Ore.	Northport, Wash.	Paper rock	524	5.80	80,000	GN, SPS, SP	11.1
Newberg, Ore.	Evans, Wash.	Paper rock	501	5.80	80,000	GN, SPS, SP	11.6
Newberg, Ore.	Arlington, Wash.	Paper rock	267	3.69	60,000	NP, SP	13.8
St. Helens, Ore.	Grants Pass, Ore.	Paper rock	324	4.50	100,000	SP, SPS	13.9
St. Helens, Ore.	Evans, Wash.	Paper rock	509	3.85	80,000	GN, SPS	7.6
Camas, Wash.	Arlington, Wash.	Paper rock	252	2.55	100,000	NP, SPS	10.0
Camas, Wash.	Gazelle, Calif.	Paper rock	435	5.70	100,000	SP, SPS	13.1
Camas, Wash.	Grants Pass, Ore.	Paper rock	322	3.50	100,000	SP, SPS	10.9
Camas, Wash.	Jaques, Ida.	Paper rock	398	3.25	80,000	Camas P. RR, NP, SPS	8.2
Camas, Wash.	Northport, Wash.	Paper rock	479	3.05	100,000	GN, SPS	6.4
Camas, Wash.	Evans, Wash.	Paper rock	456	3.15	100,000	GN, SPS	6.9
Camas, Wash.	Orcas Island	Paper rock		3.85	60,000	NP, SPS	
Portland, Ore.	Gazelle, Calif.	Chem. and met. grade	410	3.85	100,000	SP	9.4
Portland, Ore.	Grants Pass, Ore.	Chem. and met. grade	297	2.95	100,000	SP	10.0
Portland, Ore.	Lime, Ore.	Chem. and met. grade	385	3.25	80,000	UP	8.4
Portland, Ore.	Lime, Ore.	Chem. and met. grade	385	2.87	5 cars	UP	7.4
Portland, Ore.	Jaques, Ida.	Chem. and met. grade	373	3.25	80,000	Camas P. RR, UP	8.7
Portland, Ore.	Northport, Wash.	Chem. and met. grade	504	3.85	80,000	GN, SPS	7.6
Portland, Ore.	Evans, Wash.	Chem. and met. grade	481	3.85	80,000	GN, SPS	8.0
Portland, Ore.	Arlington, Wash.	Chem. and met. grade	247	2.25	60,000	NP	9.1
Portland, Ore.	Orcas Island	Chem. and met. grade		4.05	60,000	Barge and NP	
Seattle, Wash.	Jaques, Ida.	Chem. and met. grade	423	3.25	80,000	Camas P. RR, NP	7.7
Seattle, Wash.	Northport, Wash.	Chem. and met. grade	454	3.85	80,000	GN	8.5
Seattle, Wash.	Evans, Wash.	Chem. and met. grade	431	3.85	80,000	GN	9.0
Seattle, Wash.	Arlington, Wash.	Chem. and met. grade	61	1.40	100,000	NP	23.0
Seattle, Wash.	Kendall, Wash.	Chem. and met. grade	140	2.35	100,000	CMStP, NP	16.8
Seattle, Wash.	Orcas Island	Chem. and met. grade		2.20	100,000	Barge	
Everett, Wash.	Jaques, Ida.	Paper rock	456	3.85	80,000	Camas P. RR, NP, GN	8.4
Everett, Wash.	Arlington, Wash.	Paper rock	30	1.40	100,000	NP	46.8
Everett, Wash.	Kendall, Wash.	Paper rock	110	2.35	100,000	CMStP, NP	21.3
Toppenish, Wash.	Jaques, Ida.	Sugar rock	243	2.45	80,000	Camas P. RR, NP	10.0
Toppenish, Wash.	Northport, Wash.	Sugar rock	340	3.85	80,000	GN, NP	11.3
Toppenish, Wash.	Evans, Wash.	Sugar rock	317	4.45	60,000	GN, NP	14.0
Nyssa, Ore.	Lime, Ore.	Sugar rock	155	1.28	100,000	UP	23.3

<sup>a</sup> Rates include Ex-162 and 166 increases but not Ex-168.

Tariff references: Item 7455 SFTP 1-S; Sp Co. 233-H; SP Co 235-H; Bohons 5-U, I-1662; SPS 820; Bohons 2-K; Bohons 65-N; Bohons 1-N; Puget Sound Freight Lines 23-C.

is not a major problem. However, limestone for the other uses mentioned either is or has been shipped from the following localities: Gazelle, Calif., south of Yreka; Grants Pass, Oregon; Lime, Oregon, near Huntington; Evans and Northport, Wash., north of Spokane; Arlington, Wash., north of Everett; Jaques, Idaho, east of Lewiston; and the San Juan and Orcas Islands in Puget Sound.

Silica rock for the Portland-Vancouver electro-metallurgical industries is shipped from Denison, Wash., north of Spokane, and Rogue River, Oregon, near Grants Pass.

Briefly then, the sources of supply of limestone

and silica rock, with few exceptions, are in southern and eastern Oregon, northern California, Washington, and Idaho, while the principal markets for these sources are in northwestern Oregon and western Washington. The rail hauls involved range from 200 to 500 miles.

**Railroads Serving Area:** The states of Oregon and Washington are served by the lines of five major railroads, and by several local roads that are, for the most part, subsidiaries of, or at least controlled by, the major roads (fig. 1). The five major railroads are the Southern Pacific, Union Pacific, Northern Pacific, Great Northern, and the Milwaukee



(Chicago, Milwaukee, St. Paul and Pacific). The Southern Pacific, serving western Oregon, enters the state from California and terminates at Portland. The main line of the Union Pacific crosses the northern part of Oregon from the east to Portland and then turns north to Tacoma and Seattle. A branch of this railroad also reaches Spokane from eastern Oregon. The Northern Pacific, Great Northern, and Milwaukee roads enter the Puget Sound industrial area from eastern Washington. The first two continue south to the Portland-Vancouver area.

Local lines of interest are the Spokane, Portland and Seattle, the Camas Prairie, and the Oregon Electric, now a division of the S.P. and S. The S.P. and S. is owned jointly by the Northern Pacific and the Great Northern. It follows the north bank of the Columbia River and is a direct route from Spokane to Portland. The Camas Prairie serves the central Idaho region from Riparia, Wash., to Grangeville, Idaho. It is controlled jointly by the Union Pacific and the Northern Pacific, and connects with the lines of both. The Oregon Electric is an entry into the Willamette Valley for the Northern Pacific and the Great Northern. As Portland is the terminus for the major rail lines, joint line-haul and switching charges are such that, in general, shippers from the east and north cannot compete in markets south of Portland, and shippers from the south cannot compete north of Portland.

**Freight Rate Structures:** The average individual has little or no knowledge of the mechanics of freight rates. This is not surprising, as freight rate structures, particularly commodity rates, of the railroads are so complex as virtually to defy description except in terms of particular commodities. Freight moves under either a class or a commodity rate. Where there are no published commodity rates, limestone and silica rock move under a class E rate, the lowest of the class rates. The great bulk of traffic moves under, and most of the freight revenue is produced by, commodity rates. These are specific rates quoted directly, rather than through the medium of classification, on designated services, that for one reason or another require special rates. A commodity rate is usually lower than the class rate it displaces and applies to large shipments moving long distances, to low-grade articles taking short hauls, and to especially competitive traffic. The Interstate Commerce Commission, Class Rate Investigation, Docket No. 28300, exhibit No. 228 (1942) states: "Of the carloads of freight originating on September 23, 1942, on all railroads in the

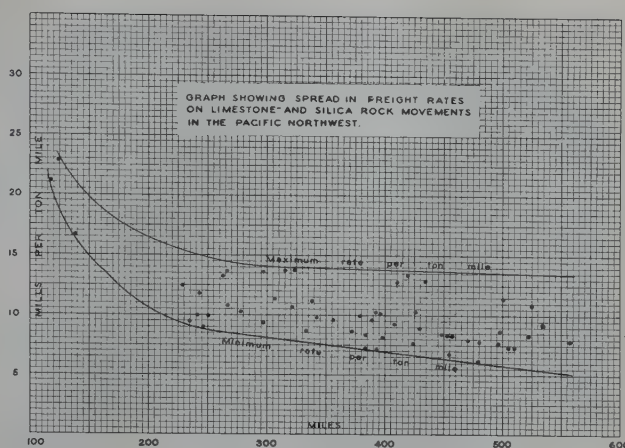


Fig. 2—Spread in freight rates.

U.S., 85.2 percent moved under commodity rates. Commodity rates accounted for 77.6 percent of the revenue from carload traffic."

Since we are interested in commodity rates and the way they affect shipments of limestone and silica rock in the Northwest, the question arises as to who sets these rates and on what basis. A shipper of limestone or silica rock, if he can show that there will be a sufficient movement of his commodity, can request a rate from the railroads involved to the point of destination. The railroads' freight rate-making agent in the Northwest, which is the North Pacific Coast Freight Bureau, then sets and publishes a rate under the supervision of the Interstate Commerce Commission. Some freight rate factors used are: (1) cost of service to the carrier, (2) value of service to the shipper, (3) value of the article, (4) nature of the article, (5) risk in handling the article, (6) distance of the haul, (7) bulk and weight of article, (8) whether special facilities or extra services are required, (9) volume of traffic and periods of movement, (10) rates on similar articles under similar circumstances and conditions, (11) competition between producing centers, (12) whether or not movement will be based upon rate, and (13) prospects of car being returned loaded or empty.

In practice, rates have not always been fixed at the best levels, nor is it likely that they ever will be exactly scientific. Rate making is an art and involves the exercise of judgment. Absolute standards are not available and between the upper and lower limits of reasonableness may lie a consider-

Table II. Railroad Rates on Silica Rock, Pacific Northwest<sup>a</sup>

To	From	Use	Miles	Tariff	Minimum Carload, Lb	Railroads	Mills Per Ton Mile
Portland, Ore.	Rogue River, Ore.	Met. grade	306	\$2.90	100,000	SP	9.6
Portland, Ore.	Dennison, Wash.	Met. grade	401	3.15	100,000	GN, SPS	7.8
Portland, Ore.	Chewelah, Wash.	Met. grade	439	3.45	100,000	GN, SPS	7.8
Vancouver, Wash.	Rogue River, Ore.	Met. grade	316	4.25	60,000	SP, SPS	13.4
Vancouver, Wash.	Dennison, Wash.	Met. grade	391	3.15	100,000	GN, SPS	8.1
Vancouver, Wash.	Chewelah, Wash.	Met. grade	429	3.45	100,000	GN, SPS	8.1
Seattle, Wash.	Rogue River, Ore.	Met. grade	498	5.45	60,000	SP, GN	10.9
Seattle, Wash.	Dennison, Wash.	Met. grade	350	2.85	100,000	GN	8.2
Seattle, Wash.	Chewelah, Wash.	Met. grade	378	2.85	100,000	GN	7.6
Spokane, Wash.	Dennison, Wash.	Met. grade	20	1.25	100,000	GN	63.0
Spokane, Wash.	Chewelah, Wash.	Met. grade	58	1.25	100,000	GN	21.6

<sup>a</sup> Rates include Ex-162 and 166 increases but not Ex-168. Tariff references: SP 233-H; Bohons 2-K; Bohons 5-U.

able zone. Rates are said to be governed by the three C's: competition, comparison, and compromise.

Former Commissioner Eastman of the Interstate Commerce Commission expressed his idea on the origin of rates when he said in 1925, "The railroad rate structure of the United States is the product of the efforts of freight-traffic managers seeking maximum revenue of their respective railroads, modified to some extent by public regulation." (100 I.C.C. 513611.)

**Comparison of Rates:** A study of local rail rates on limestone and silica rock shows a wide variance in mills per ton mile even for hauls of like distances. Fig. 2, with the charge per ton mile plotted against the distance hauled, clearly shows the spread in freight charges on limestone and silica rock movements in the Pacific Northwest. Most of the movements fall within the range of 250 to 500 miles. For these distances, the maximum charges per ton mile are roughly twice the minimum charges for the same distance.

These differences raise a question in the minds of laymen, particularly when other factors appear the same, such as one-line or joint-line hauls. Here note the following examples:

Commodity hauled: Limestone.

	Distance, Miles	Tariff	Mills Per Ton Mile	Railroad
Grants Pass to Amity, O.	231	\$2.90	12.5	SP
Arlington to Vancouver, W.	237	2.25	9.5	NP
Grants Pass to Salem, O.	244	2.90	11.9	SP
Arlington, W., to Portland, O.	247	2.25	9.1	NP

In some cases joint hauls are lower than single hauls of like distances. If one were to believe that the various freight-making factors were used to determine the tariffs, the reverse should be true, but note the following examples:

Commodity hauled: Limestone.

	Distance, Miles	Tariff	Mills per Ton Mile	Railroad
Jagues, I., to Seattle W.	423	\$3.25	7.7	Camas P. & NP
Northport to Camas, W.	479	3.05	6.4	SP & S & GN
Gazelle, C., to Portland, O.	410	3.85	9.4	SP
Lime to Portland, O.	385	3.25	8.4	UP

In some cases, rates on joint hauls to greater distances are the same as rates on a single haul for part of the distance, and in other cases the reverse is true, as can be seen from the following examples:

Commodity hauled: Limestone.

	Dis- tance, Miles	Tariff	Mills Per Ton Mile	Railroad
Arlington to Vancouver, W.	237	\$2.25	9.5	NP
Arlington to Camas, W.	252	2.55	10.0	NP & SP&S
San Juan Is. to Vancouver, W.		3.85		NP*
San Juan Is. to Camas, W.		3.85		NP & SP&S*

\* By barge to Seattle.

Other examples of inconsistencies are:

(1) The comparison of hauls from Northport, Wash., and Jagues, Idaho. The tariff from Northport to Camas is \$3.05, and to Portland \$3.85; while the tariff from Jagues to Camas is \$3.25 and to Portland it is also \$3.25. Both hauls are handled by the SP&S railroad.

(2) The tariff from Northport, Wash., to Camas,

Wash., is \$3.05. That from Evans, Wash., to Camas is \$3.15. Both loading points are on the same rail route. The only difference is that Evans is 23 miles closer to Camas.

(3) In the case of hauls from Denison and Chewelah, Wash., of silica rock the rates to Spokane are both \$1.25. They are both \$2.85 to Seattle, but to Portland, the tariff from Denison is \$3.15, while that from Chewelah is \$3.45.

## Conclusion

Limestone and silica are low value commodities which are handled by railroads in large volume with standard equipment at minimum risk and expense. Both products are shipped in bulk in the raw or semifinished condition. Silica rock produced in one section of the Northwest is almost identical to that quarried in another in handling characteristics, and value f.o.b. cars. The same is true of the various limestone operations. It would appear, therefore, that since little or no cost differential exists between the several movements of silica rock and limestone in the Northwest from the standpoint of handling charges, value, bulk and weight, risk of handling, and value of service to shipper, that the rate for each of these commodities should be determined primarily by the distance hauled. Volume of traffic and periods of movement should have some effect on the tariff although they should represent only a relatively small modification.

Limestone and silica rock are sold f.o.b. cars in the Northwest for two to three dollars per ton, and a freight differential of a dollar or less on such low-cost products is a dominant marketing factor. Variation in freight tariffs, as shown in tables I and II, clearly indicates the competitive position of the various producing localities for the markets.

Although this paper is concerned with limestone and silica rock, the variance in rates for like hauls can be duplicated in other industrial minerals. In general freight rates make up the major part of the cost of producing industrial minerals and therefore the freight rate usually determines whether or not such a project is economic. It is evident that establishment of rates is unscientific and usually on the basis of "all the traffic will bear." This brings into focus the extreme need for reviewing freight rates on industrial minerals to obtain more uniformity in such rates and less power to the railroad to make or break a shipper or prospective shipper.

## Bibliography

Truman C. Bighan: Transportation, Principles and Problems (1947). McGraw Hill Book Co.

The Official Guide of the Railways. National Railway Publication Co.

Lewis D. Cannell: The Freight Rate Structure and Its Effect on the Price and Movement of Northwest Wheat. State College of Washington. Bur. of Econ. and Bus. Res. Bull. 2. (1945).

Ralph L. Dewey: The Long and Short Haul Principles of Rate Regulation. Ohio State University (1935).

Marshall T. Hunting: Directory of Washington Mining Operations (1949). Division of Mines and Geology. I.C. 17.

Freight Traffic Red Book (1948).

J. E. Allen: Reconnaissance Geology of Limestone Deposits in the Willamette Valley, Oregon (1946). G.M.I. Short Paper No. 15.



# Fuel Economy in the Lepol Kilns

by R. A. Kinzie, Jr.

In a conventional cement plant, the drying and heating of the raw material takes place in the upper section of a cylindrical kiln where the heat exchange is poor.

In a Lepol kiln this part of the process is carried out on a traveling grate where the heat exchange is efficient. The increase in fuel economy justifies the greater complexity of the installation.

THE major operating costs in a cement plant are labor, power, and fuel. The opportunities and methods of savings in labor and power parallel other industries. Because our industry's use of fuel is large and its method of use is unique, the devices adapted for saving fuel are of interest.

R. A. KINZIE, JR., Member AIME, is Superintendent, Santa Cruz Portland Cement Co., Davenport, Calif. AIME San Francisco Meeting, February 1949.

TP 2775 H. Discussion (2 copies) may be sent to Transactions AIME before March 31, 1950. Manuscript received May 6, 1949.

In normal practice the raw material in the form of a dry powder or a mud is fed to a rotating inclined cylinder where it is dried, calcined, and clinkered. In a Lepol kiln the raw material is rolled up into little balls with water in the nodulizer, fed to a traveling grate on which it is dried and partially calcined and then to a short kiln where the process is finished.

The more usual methods of saving heat have been the use of long kilns with chains in wet process plants and the use of waste heat boilers in dry plants (fig. 1). At the Santa Cruz plant we have used a Lepol system kiln and achieved a fuel economy of 4.7 gal per bbl of clinker or about 704,000 Btu per bbl. This can be compared with our old kilns using the same fuel and raw material which consume 7.8 gal per bbl or 1,130,000 Btu per bbl.

These are gross Btu and not the net figures usually used in reports of foreign companies.

Our average consumption for 1948 was 708,000 Btu per bbl of Lepol clinker, based on cement sold, and oil purchased. The overall fuel consumption for 1948 was 6.61 gal per bbl.

The Lepol system consists of a nodulizer, a grate, and a kiln. There is the usual accessory equipment

which is similar to that used in other cement plants. The following description covers the equipment in operation at our plant.

The nodulizer is a cylinder 8 ft 10 in. in diam by 18 ft long set on a slope of  $\frac{1}{2}$  in. per ft, and rotated at about 10 rpm. It is driven by a 50 hp wound rotor motor drawing about 9 kw. The inside of the shell is cleaned by a continuously operating scraper which draws 0.1 kw. The dry raw material is fed to the nodulizer where, being sprayed with water, it rolls into small spheres about 90 pct being between  $\frac{1}{2}$  and 1 in. in diam and containing about 15 pct water.

The nodules fall on the traveling grate where the hot gas from the kiln is pulled down through the

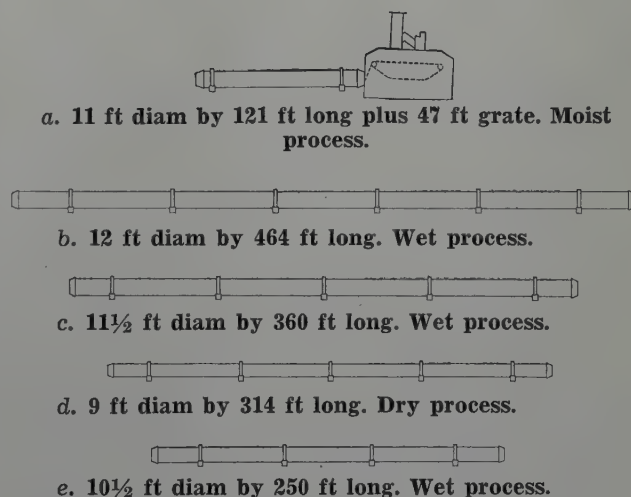
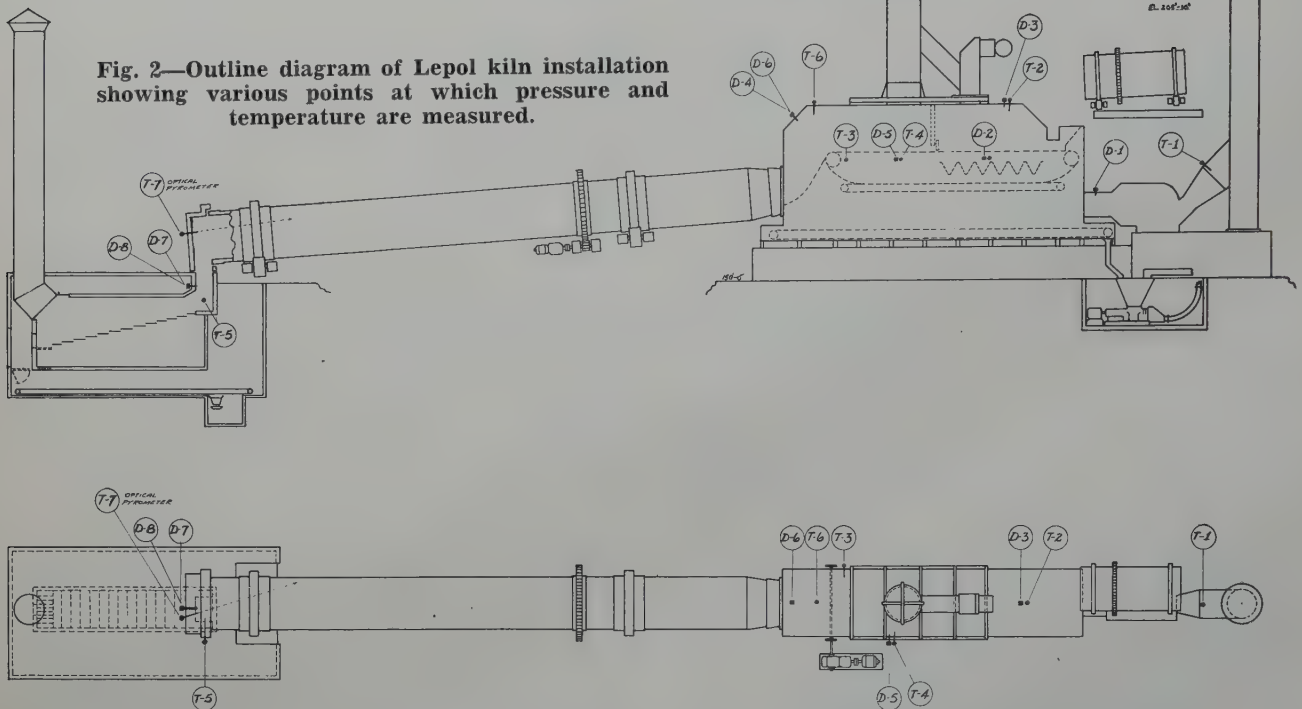


Fig. 1—Size comparison of typical wet and dry process kilns with the Lepol grate and kiln.

nodules and grate and then exhausted. The grate is 47 ft 6 in. between centers and in width 12 ft 5 in. between the centers of the outside chains. The open area of the grate is 40 sq ft which is equivalent to a circle 7 ft 2 in. in diam. At present, the grate travels 32 in. per min and carries a 6 in. bed of nodules.

Fig. 2—Outline diagram of Lepol kiln installation showing various points at which pressure and temperature are measured.



The grate is driven by a 15 hp BTA motor drawing 5 kw. On the return run the grate is supported on a motor-driven chain which supports its weight.

The power for this chain is included in the power given for the grate motor.

The material from the grate falls into the kiln of standard design. The kiln is 112 ft long by 11 ft in

This will compare favorably with standard kilns equipped with chains or waste heat boilers.

The kilns do require more labor than normal kilns but this should not exceed \$0.03 per bbl of clinker.

The repairs chargeable to the Lepol system are low and the lost time on separate kiln runs has varied from 24 to 42 pct of the total lost time.

The instruments on our Lepol system are greater in number than on a normal kiln. Our old cylindrical kilns have only a thermocouple indicating stack temperature.

On the Lepol the temperatures at the inlet to the exhaust fan, the top of each end of the grate, and the bottom end of the calcining end of the grate are measured by thermocouples and recorded (fig. 2). A ray-o-tube indicates optically and records the burning zone temperature of the rotary kiln. These are all Leeds and Northrup instruments.

Draft gauges indicate the pressure at the inlet to the exhaust fan, the differential through each end of the grate, the differential through the kiln, and the pressure at the kiln hood. The differential through the kiln is used to control the damper at the exhaust fan. The speed of the grate is recorded as well. The speed of the grate is interlocked with the kiln speed and one of three values can be selected by the burner. The nodulizer is independently controlled by the nodulizer operator.

**Material Weights:** The weight of clinker produced is calculated from the clinker storage measurements adjusted to sales of cement.

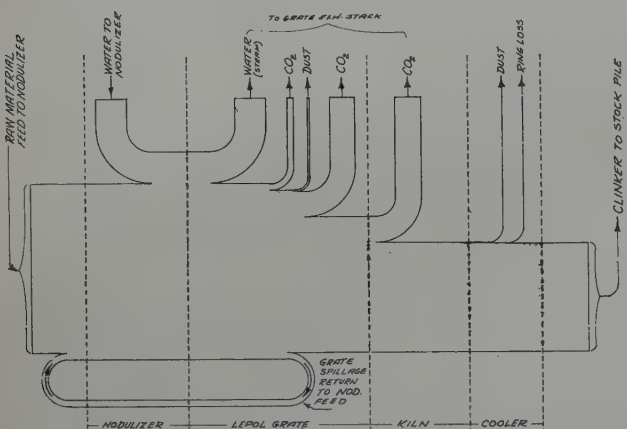


Fig. 3—Material balance, Lepol kiln.

diam supported by two riding rings and driven by a 40 hp wound rotor motor drawing 21 kw. The slope of the kiln is 7/16 in. per ft.

The main fan is driven by a 125 hp motor drawing 76 kw.

The overall power consumption of the nodulizer, grate, kiln, and exhaust fan is 1.25 kw-hr per bbl.



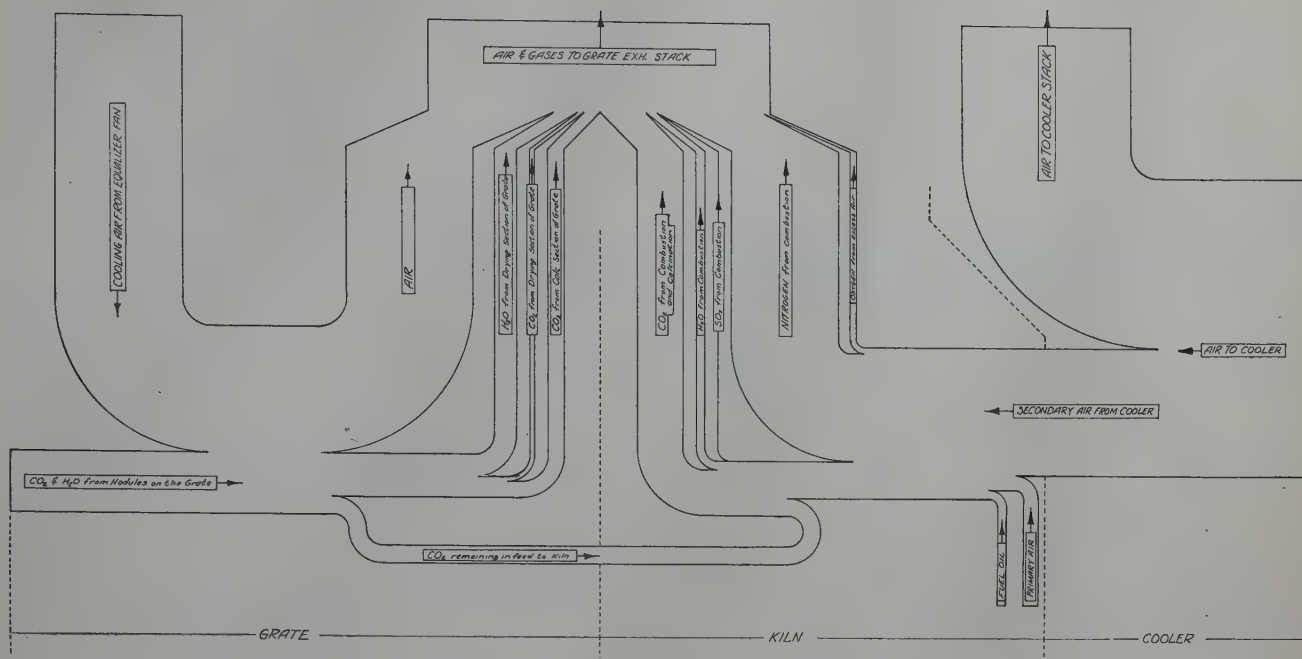


Fig. 4—Gas weight balance, Lepol kiln.

The weights in other parts of the circuit are believed to approximate quite closely the actual values (fig. 3).

**Gas Weights:** The weights of gas involved have as a basis of calculation the measured volumes in the cooler intake and exhaust, the primary air, and the tempering air drawn into the grate. The volume of gas of combustion is calculated from the analysis and weight of oil burned (fig. 4).

The final weight of air exhausted is a little less than that which has been measured on occasions. No attempt was made to balance the difference by adding a leakage to the calculations.

The weight of oil is an average which agrees within  $1\frac{1}{2}$  pct with the average for 1948.

**Heat Balance:** The values of heat content of the gases and clinker were taken from the "Table of Heat Content" published by the Portland Cement Association. The heat of formation of clinker is that calculated by Martin. The air was assumed to have a 50 pct relative humidity (fig. 5).

Because of difficulty in measuring the gas tem-

perature entering the kiln from the cooler, the heat content was calculated by subtracting the heat leaving the cooler from the heat content of the clinker entering the cooler. The heat leaving the cooler, because of the lower temperatures involved, is easier to measure. The temperature of the clinker entering the cooler was determined by optical means.

The weights of gas involved were taken from calculations illustrated in a previous chart.

The 24,000 Btu unaccounted for includes the loss from the grate housing, the cooler housing, and miscellaneous small losses.

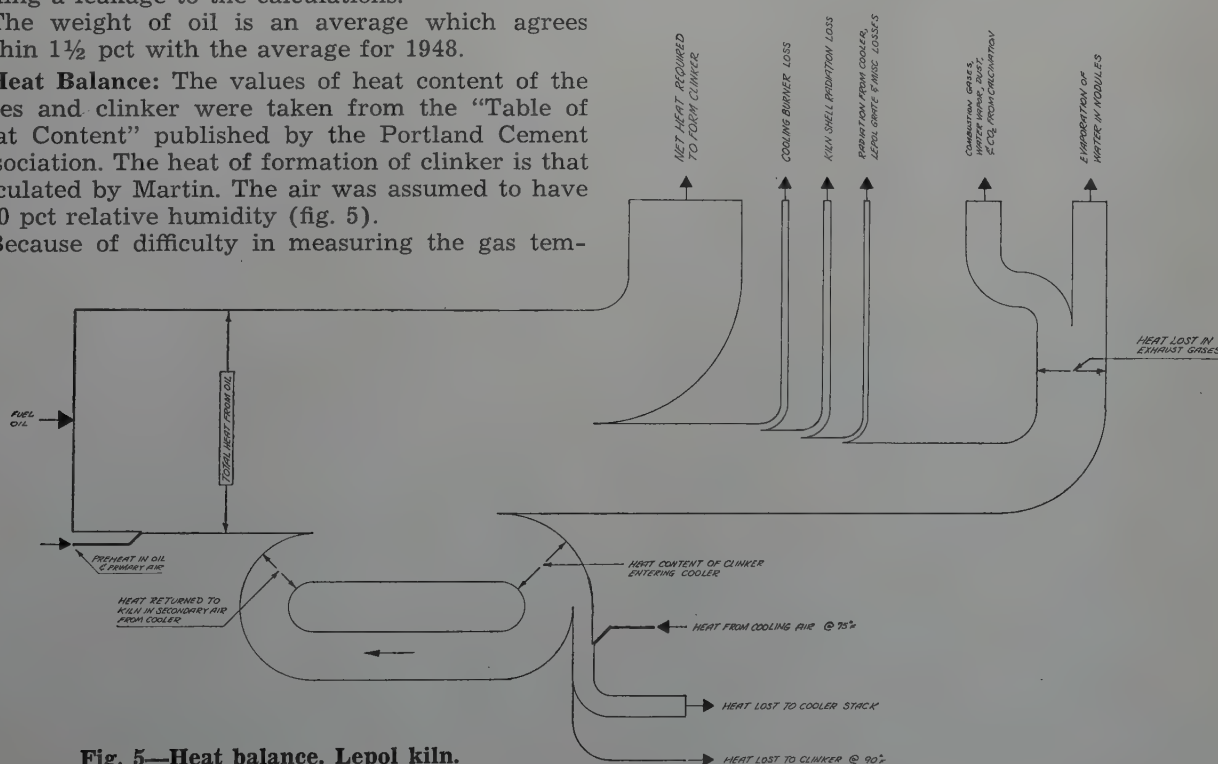


Fig. 5—Heat balance, Lepol kiln.

# Industrial Mineral Economics and the Raw Materials Survey

by Raymond B. Ladoo and C. A. Stokes

This paper summarizes the economic problems of the industrial mineral industries which are essentially different from those of the metals and the fuels. Failure to understand and evaluate such factors as markets, place value, and substitutions of one mineral for another may be disastrous. The engineer should be as responsible for profitable operation as he is for technical efficiency. The Raw Materials Survey of Portland, Oregon, is described.

IT is unfortunate that the word "economics" has come to mean, in the minds of many people, a sort of half-baked mixture of New Deal philosophy and bookkeeping. It may mean anything from mine cost keeping to the theory of gold as a monetary standard. As applied to the industrial mineral industries as a whole it means to us an integrated

---

RAYMOND B. LADOO, *Member AIME, is a Consulting Engineer, Newton, Mass., and C. A. STOKES, Member AIME, is Director of Research and Development, Godfrey L. Cabot, Inc., Boston, Mass.*

*AIME San Francisco Meeting, February 1949.*

TP 2772 H. Discussion (2 copies) may be sent to Transactions AIME before March 30, 1950. Manuscript received March 3, 1949.

---

study of all of the technical, geographical, market, financial, and other factors which determine whether or not a given enterprise has a sound chance of making a business success.

Of course it is impossible to generalize too freely in this large group of industries which go to make up the nonmetallic minerals. This group includes such dissimilar products as crushed stone and ca-

pacitor mica, phosphate rock and spinning fiber asbestos, roofing granules and potash salts, borate minerals and monumental granite. Yet in many respects their economic problems are similar and, at the same time, far different from those of the metals and of petroleum and coal. These basic differences are almost never recognized by people outside the industries and are rarely appreciated even by metal mining engineers. But in many instances these economic factors are fully as important as, if not more important than, technologic problems.

It has been a long fight to get an intelligent and wide-spread appreciation of sound technology in the nonmetallic mineral industries. The importance of the economic side of the problem has yet to be widely understood and appreciated. The problem was outlined in considerable detail by one of the authors some 18 years ago in a long series of articles\* and he has since prepared numerous talks

---

\* Raymond B. Ladoo: Economics of the Non-Metallic Mineral Industries. *Rock Products* (Feb. 28, 1931 to June 18, 1932).

and papers on the subject. But it has seemed like a dry, uninteresting matter. Little has been done to spread the gospel where it is most needed. But it is



as much the duty of the engineer to give sound advice on economic problems as it is on technical problems. This has been recognized in most descriptions of the function of the engineer. Yet that phase of engineering is usually overlooked. Too many men who call themselves engineers have little or no ability to evaluate economic problems. Efforts are now being made to remedy this situation by giving courses in mineral economics in some of our universities. The Division of Minerals Economics at Pennsylvania State College is an outstanding example.

Every industry must be governed by economic considerations. But there is little point in repeating again the well-known and often tabulated economic factors common to most businesses. In this class are such factors as labor supply, efficiency and cost, climatic factors, interest rates, local taxation, and a host of others. Other factors peculiar to the mining industry in general are well known to most engineers and are to be found in books on mine valuation and in handbooks on mining engineering. Except as they have unusual characteristics in the industrial mineral industries they will not be discussed here. It is our purpose to think about, talk about and popularize those factors of minerals economics that are peculiar to the industrial minerals.

**Economic Factors Peculiar to the Industrial Minerals:** While it is difficult to give short titles properly descriptive of some of these factors we can attempt to list some of them as follows:

1. Lack of free markets.
2. Importance of place value.
3. Variable mineral composition and inaccurate nomenclature.
4. Interchangeability of minerals for some uses.
5. Time required for product and market developments and effect on capital needs.
6. Nonmetallic mineral "alloys," natural and artificial.
7. Synthetic minerals and mineral substitutes.
8. Inadequate application of research and development to production, processing, and application problems.
9. Influence of technologic developments.

These factors are discussed briefly below.

**Lack of Free Markets:** By this we mean that few industrial minerals are such uniform, standardized articles of commerce that they can be sold and used interchangeably, anywhere, at any time. At one end of the scale of marketability is gold, which can be sold or exchanged freely anywhere in unlimited quantities. At the opposite extreme are many nonmetallic minerals such as feldspar which must have definite consumer acceptance before they can be sold and then only in limited quantities. Few industrial minerals can be used directly by the ultimate consumer. They are not consumer goods. Except by finding new uses total consumption can grow only as industrial users' markets are developed and grow.

**Place Value:** Many industrial minerals have such low unit value or are so abundant that proximity to markets is more important (with obvious reservations) than quality or quantity. Thus a large deposit

of high grade building sand near large and growing industrial centers may be very valuable, while a similar deposit in the middle of a Nevada desert is valueless.

But place value is something which is constantly changing due to a wide variety of factors. Among these are:

1. Changing geographical distribution of population and of industries, bringing formerly worthless deposits within economic shipping radius of consuming industries.

2. Improved production processes may improve quality or reduce costs or both, so as to offset, in part, high transportation costs.

3. Improved transportation methods, from deposits to railroad, make remote deposits increasingly accessible economically. Among these should be noted the airplane (asbestos and jade from interior Alaska), good roads and motor trucks, pipelines (clay in slurry form pumped from deposits to mills in Georgia), and long cross-country hauls of cement raw materials by belt conveyor.

4. New market demands, both for quantity and quality, gradually justify the working of remote deposits.

5. As the chemical and physical processing of minerals becomes more complicated and extensive, plants tend to become larger and energy needs much greater. The future may see a greater trend toward development of minerals near sources of cheap energy.

6. As costs for labor and plant construction in the process industries rise, raw material costs constitute a smaller percentage of total costs. Thus the paying of higher transportation charges on raw materials from remote sources, may become relatively less objectionable.

This list could be extended but enough has been said to demonstrate the great importance of location of deposits with respect to markets.

**Variable Mineral Composition and Inaccurate Nomenclature:** Among the many factors which make the economics of this field difficult to grasp and understand is the wide variability in composition and chemical properties of many materials which we call by a single mineral name.

While this is true in the whole field of mineralogy it is not so important in the group of metallic ore minerals. Of course, the mineralogy of an ore has an important bearing on the ease and cost of ore beneficiation and of smelting and refining. But the product finally marketed is relatively pure, uniform, and standardized. The salable end products, whether they be copper, lead, zinc, or tin, are standard regardless of the mineralogy of the original ores.

But in the nonmetallic mineral field the names we call our final products may be no guides at all to the chemical composition or the physical and chemical properties. Any number of minerals could be used as examples, asbestos, mica, clay, graphite, bentonite, feldspar, ocher. But as a fairly simple example let us take talc.

The mineralogies give us a definite chemical formula and percentage composition for the pure material, of which, of course, there is very little

produced commercially. The talcs of commerce vary greatly from the pure material, both in chemical composition and in physical properties. The so-called fibrous talcs of St. Lawrence County, New York, differ so greatly from the talcs of Vermont and Georgia that they find almost wholly different uses, they are noncompetitive. In fact, recent careful work by the U. S. Geological Survey on the New York talcs have proved that much of the material sold as talc contains as little as 5 pct talc (or even none at all in one case), the balance being a mixture of tremolite, serpentine, anthophyllite, and occasionally diopside, quartz, calcite, and other minerals.

But U. S. Bureau of Mines statistics of talc production include all material sold as talc, plus finely ground soapstone, plus pyrophyllite. How can a prospective new producer of talc from, say, a newly discovered deposit in Michigan, gain an adequate idea of the size of the potential markets for his talc from published statistics? Worse still, how can he know that the statistics he finds mean very little without experienced interpretation?

Even the average metal mining engineer would not suspect that published production statistics were not a good and safe guide. Otherwise able metal mining engineers have made some very costly blunders when they have, with overconfidence due to ignorance, tried overnight to be experts on industrial minerals. They did not know the problems and they did not know that they did not know.

How can we as industrial mineral engineers and technologists educate the rest of our own profession at least to the realization that our problems are different?

*The Interchangeability of Minerals:* The interchangeability of minerals or minerals and chemical products, complete or partial, for many important uses is an economic factor, the importance of which is often overlooked even by experienced producers. Consider feldspar. The feldspar producer not only must compete with other feldspar producers but also for many uses, with aplite, nepheline syenite, artificial mixtures such as Gar-Spar, feldspathic sands, and, to some degree with pyrophyllite and talc. This list could be extended. Unless he realizes this situation, a present or prospective feldspar producer may seriously err in his judgment of market possibilities. This type of competition may or may not be very serious.

From the standpoint of a whole industry, feldspar again for example, this substitution of one product for another may not be too serious. The industry as a whole may show a satisfactory rate of growth. This may be due to fortuitous increases in use by present consuming industries, feldspar in glass for example, or to entirely new uses. But the change in geographical distribution of markets may be disastrous to individual producers. Old producers may have to drop out and new ones, more favorably located, take their places.

The extent of this type of competition can here be only touched upon briefly, but it includes competition not only with other minerals, rocks, and mixtures of materials, but also with chemically prepared products. An example is chemically precipitated calcium carbonate as a paper filler and coating material to replace clay.

This interchangeability of materials is at once a danger to established producers and an opportunity, for it may extend and widen markets for their own materials. In any event it further emphasizes the basic need for mineral producers to study markets. It is a challenge to the ingenuity and imaginativeness of all executives and engineers in the mineral industries.

In studying markets it is not enough to know that the A.B.C. Co. today uses 5000 tons of talc per year. We must know what kind of talc, why and how it is used, exactly what physical or chemical function it performs in the user's products or processes. Is there any other mineral or chemical product or mixture of materials which will perform this essential function satisfactorily? Can such a product be obtained anywhere else in adequate supply, of proper quality and uniformity, and at an equal or lower delivered cost?

To obtain and understand this kind of data an engineer must have a good background of industrial chemistry and acquire a fairly detailed knowledge of many industrial processes. Often plant production men, engineers and chemists, have an astonishing lack of knowledge of raw materials and their functions, and the mineral producer himself may have to dig out the information he needs.

*Time Required for Product and Market Development and Effect on Capital Needs:* Due to variability in composition and lack of free markets for many industrial minerals a new producer must usually spend considerable time in developing methods of production to make uniform products acceptable to the trade. It may take months to open up enough working faces in the mine or quarry to supply daily mill requirements. Even if the final products appear to be identical with established products, consumer acceptance may be slow except in times of shortages.

The result of these delays is that a new project may have to wait two years or five years or even ten years, before there are any profits. To tide over this nonprofit period may require large capital expenditures for production labor, power, supplies, taxes, research and market development, and other items. Many new producers and, often, experienced producers overlook this factor and fail to provide sufficient capital at the start. Too often additional capital is not available when needed and the project, sound enough in itself, fails before it is well started.

In contrast with this, some metal mining and milling operations are so standardized and markets are so well established that a new mill may be built and operating to capacity in from six months to a year.

*Nonmetallic Mineral "Alloys," Natural and Artificial:* Some industrial minerals useful for their chemical properties or for both chemical and physical properties, such as ceramic raw materials, are mixtures of two or more minerals, somewhat similar to metallic alloys, but the ingredients occur in natural mixtures and in variable proportions. Feldspar is a good example. Practically all commercial feldspars are mixtures of potash, soda and lime feldspars, plus silica and minor accessory minerals, and no two are exactly alike.

Natural cement rock is a mixture of lime, silica,



and alumina minerals in such proportions that, after calcining, the product has the properties of a hydraulic cement. Blast furnace slag used for cement making is an artificial mixture of these same ingredients.

Such mixtures are always of variable composition and physical properties and require special techniques to make uniform, salable products. The production of pure metals, such as gold, silver, copper, lead, and zinc, requires no analogous processing.

*Synthetic Minerals and Mineral Substitutes:* Chemical processes are used today to make numerous synthetic and artificial products which replace natural mineral products to a greater or lesser extent.

Silicon carbide is a good example of an artificial mineral, not found in nature, which has nearly driven from the market natural abrasives of certain types. Electrically-fused pure aluminum oxide similarly replaces most of the natural corundum and emery formerly produced for abrasive use. These two artificial abrasives together have also made nearly obsolete the production of many types of grindstones, pulp stones, and sharpening stones of all kinds.

Synthetic ruby and sapphire have nearly replaced natural stones for jewel bearings.

Quartz crystals suitable for radio oscillator plates and sheet mica of capacitor grade have been made artificially, but as yet the synthetic products are no threat to the natural minerals.

Artificial graphite has replaced much natural graphite for certain types of uses.

While we are not yet able to make spinning fiber asbestos synthetically, glass fibers are constantly being improved and may in time replace asbestos.

In the metal field, no one has yet been able to make synthetic gold or copper or tin, commercially, although necessity has forced widespread substitutions.

It is of interest to note that most mineral substitutes are themselves made from other minerals.

*Inadequate Application of Research and Development to Production, Processing, and Application Problems:* While it is true that the last two decades have seen great progress in the application of engineering and technologic principles by the industrial mineral industries, there are still many industries which could profit by more research and development work.

One of the trends in research and development today is the seeking of more specific functionality even at considerably increased costs, in other words, sharp-shooting the consumers' needs. An example of this is the shipping of clays and precipitated calcium carbonate in slurry form in tank cars. Other examples are the pelletizing of dusty fillers and pigments, coprecipitation of pigments and fillers with rubber, and master-batching with plasticizers.

Another tendency related to the above is that toward more chemical and physical modification of minerals. Outstanding examples are the micron size fractionation and the bleaching of clays for paper fillers and coatings. Crude clays are used today

which, until recently, were considered worthless for these uses. The addition of surface coatings to pigments and fillers is another example.

*Influence of Technologic Developments:* Improved methods and equipment for mining, treatment and testing of industrial minerals have economic effects far beyond savings in costs and improvement in quality.

In the past few years there have been many new inventions and technological improvements in processing methods and machines which have had a profound effect on established industries. Some of these are noted below:

1. Froth flotation
2. Sink-and-float or heavy-media separation
3. Greatly improved air separation
4. Centrifuging, either alone or with electrophoresis
5. Micronizer—fluid energy grinding
6. High intensity magnetic separation
7. Improved electrostatic separation
8. Practical bleaching methods for white fillers
9. Spray drying
10. Air conveying
11. Electric eye controls
12. High gravity artificial grinding pebbles
13. Humphreys spiral concentrator.

Other devices could be cited. Many of these, of course, are not limited to the nonmetallic minerals.

These improved working tools lead to the use of lower grade or alternative minerals, thus increasing reserves and widening the geographic distribution of reserves.

We now have new and improved equipment and techniques for examining and testing minerals and mineral products, such as the electron microscope, X ray diffraction techniques, and spectroscopic methods of analysis. These all contribute to a better understanding of the composition and properties of minerals and tell us why certain minerals behave as they do in use applications. With the help of the basic knowledge that these tools contribute, we can find other minerals or combinations of minerals and chemicals as substitutes which may do a given job better or cheaper.

The geographic distribution of consuming industries, of population, and of per capita wealth, due largely to war developments, hydroelectric plant construction as in the T.V.A., Hoover Dam and Bonneville areas, greatly increased freight rates, and the Supreme Court basing point decision on portland cement, all have very great influence on industrial mineral production and markets. These changes have created new consumer markets, leading to the building of localized plants to serve those markets. This, in turn calls for new sources of raw materials, local if possible, but at least available at permissible costs.

Technological advances together with new consuming areas to provide markets make deposits commercially valuable which once were of no interest.

**The Raw Materials Survey:** Much of the preceding discussion has been upon the subject of markets for mineral products and their influence on the value and workability of deposits. In the past our supplies of mineral products have been sufficiently abundant and low enough in price so that producers have had to seek markets. Until recently, industrial buyers have not felt the need to worry much about sources of raw materials. Market surveys have become commonplace, but only recently have we begun to hear about the reverse of the picture, raw materials surveys.

An outstanding development of this new concept is The Raw Materials Survey, of Portland, Oregon, an organization conceived in 1946 and founded in 1947, to study industrial raw material problems of the Columbia River basin in the Pacific Northwest. It was an outgrowth of special studies of sources of certain mineral raw materials made by the senior author and sponsored by the Industrial Committee of the Portland Chamber of Commerce. The concrete plans for a continuing agency to study such problems were made by Paul B. McKee, president of Portland Gas and Coke Co., Chester K. Sterrett, chairman and manager respectively of the Industries Committee, and the senior author.

This nonprofit organization is supported by Chambers of Commerce, utility companies, steamship and railroad companies and port authorities of the area. Its president is General Thomas M. Robins and the managing engineer is A. O. Bartell.

The purpose of the survey is to gather and furnish to interested companies information on sources of raw materials needed by present and potential industries of the area. The need for this work arose from the fact that there was no single agency which could supply accurate, essential information of this character. It had been sometimes said that while the Pacific Northwest had ample water, electric power, and agricultural and forest product resources, it was deficient in mineral and other raw materials essential to a balanced industrial economy.

One obvious answer to this is that while Connecticut has no copper or zinc deposits, its brass industries have grown and prospered for nearly 200 years. Another answer is that substitute raw materials, locally produced, may replace some conventional raw materials and even perhaps do a better job. A third answer is that foreign raw materials, brought in by water, may possibly be delivered in Portland or Vancouver or Longview more cheaply than from domestic sources. Our eastern seaboard industrial economy in the early days was built largely on foreign trade.

The Raw Materials Survey does not duplicate the work of any other agency. While one of its functions is to make special studies for which it gathers much original data, its daily function is to gather, from every possible source, information on sources, qualities, freight rates and prices on a wide variety of raw materials, mineral and nonmineral, and to study raw material requirements of present and possible future industries. The results of this work are either published as special surveys or in a Monthly Newsletter, or used to answer inquiries.

This work requires knowledge of the requirements of the process industries, strong imagination

and resourcefulness, and a keen eye for ferreting out odd bits of seemingly unrelated data and piecing them together.

The underlying policy of this Survey is not to boom local raw materials, good or bad, but to give consuming industries honest, reliable facts as to where they can get essential raw materials. Industries do not care whether the sources are local, domestic, or foreign provided they can get what they want in adequate quantity of proper and uniform quality at a price they can afford to pay.

At times, it has been thought that this policy was detrimental to local and state mineral industries. But this is not a valid criticism. If local industries can produce what is needed on a competitive basis, they will get the business, but there is no point in artificially bolstering up a local weakling just to satisfy local pride. In the long run sound consuming industries are of more value to a state than uneconomic raw material producers.

**Stockpiling Problem:** An economic problem much in the public eye today is that of stockpiling of minerals essential to war. Here again, the case of most nonmetallic minerals differs from that of the metals. First, most nonmetallic minerals occur in relative abundance in this country, notable exceptions being radio-grade quartz crystals, capacitor-grade mica and mica splittings, industrial diamonds, crucible-grade graphite, and a few others. Second, for many uses there are acceptable or partial substitutes, or by research substitutes can often be found. Contrast this with the metals, where there are only moderate opportunities to develop substitutes for iron, copper, lead, zinc or tin.

For such nonmetallic minerals as we have in fairly adequate supply, the only reason for stockpiling is that war demands may come at a rate much faster than our ability to produce. For really abundant minerals this seems hardly an adequate reason.

For materials like fluorspar, where our reserves are adequate for only a short term and where the rate of extraction is limited by physical factors, stockpiling, if from foreign deposits, has the great additional merit of extending our reserves.

The alternative to the stockpiling of minerals which we cannot produce here, such as quartz crystals, is research to make them synthetically or find adequate substitutes for them. As is well known, such research is now going on. But it cannot be depended upon to solve our problems quickly and stockpiling for military needs is essential.

**Conclusions:** It should be evident that the engineer and the producers in the industrial minerals field should be as cognizant of economic problems as they are of technical and engineering problems. In most instances, the economic problems are of first importance, because if products cannot be sold at a profit, clever mining or processing is of no avail; the project fails. The engineer is responsible for the overall success or failure of the project. It has been impossible here to give more than a very brief and incomplete outline of the problems, but its importance deserves the thought and attention of many engineers and producers.



# Use of Concrete Underground

Use of concrete underground at No. 4 mine, Cornwall, Pa., has become increasingly important. Starting with small amounts at finger openings, occasional raises, piers, its use has expanded to entire slushing drifts and haulages. Besides eliminating costly timber repairs, concreting costs of slushing drifts have proved cheaper than timbering. Higher haulage-concreting costs are offset by no maintenance charges during the life of the haulage.

by Joseph Bernhardt

**T**HE Cornwall Ore Mines, Division of the Bethlehem Steel Co., at Cornwall, Lebanon County, consists of two separate magnetite ore bodies, approximately one mile apart. The one ore body was an outcrop in which open-pit mining operations were begun in 1742 and continued without interruption. This mine is also served by a shaft for mining that portion of the ore body not recoverable by open-pit methods.

The other property had no outcropping and is entirely an underground operation. This mine, captioned No. 4 mine, is serviced by two shafts 220 ft apart. The No. 4 shaft is for hoisting rock and ore, and the No. 5 shaft is for handling men and ma-

JOSEPH BERNHARDT is Superintendent, No. 4 Mine, Bethlehem Steel Co., Cornwall, Pa.

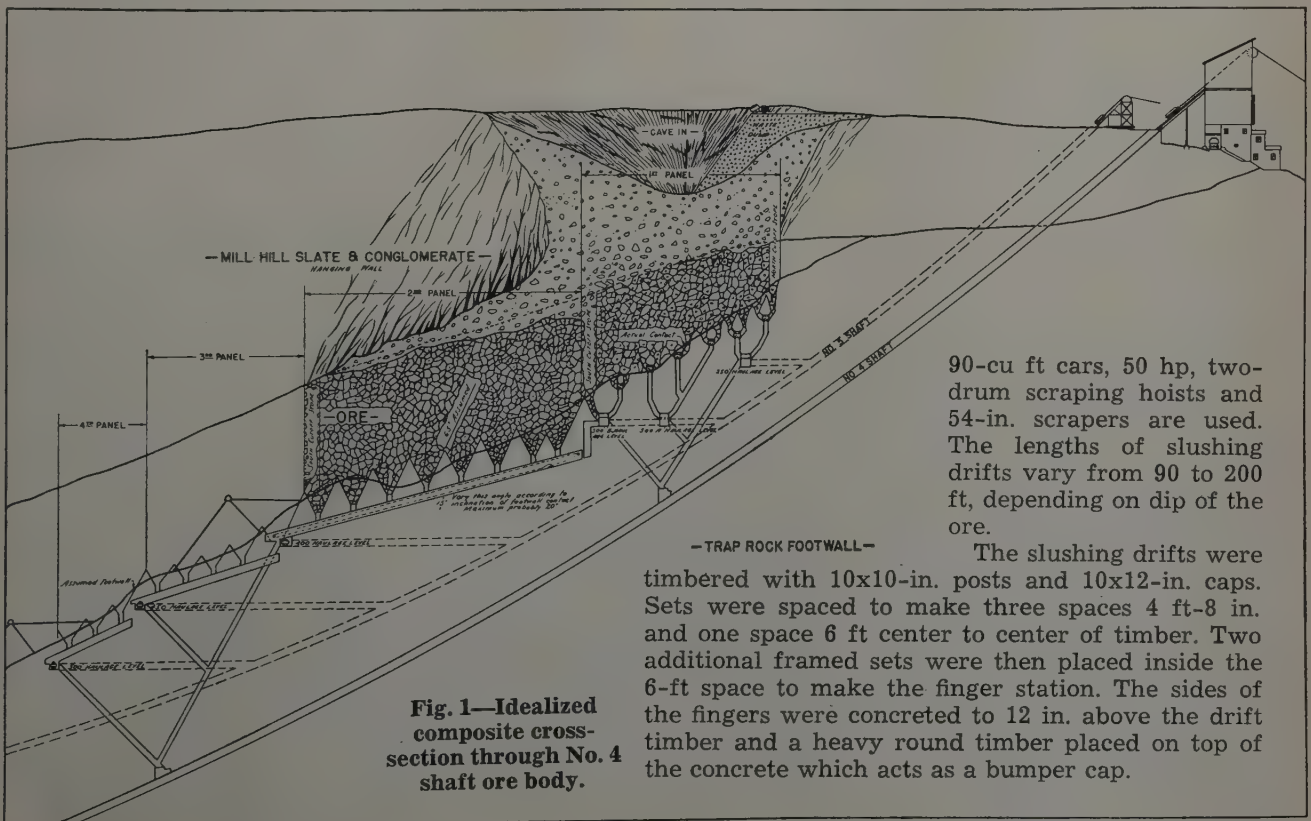
AIME Columbus Meeting, September 1949.

TP 2830 A. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Aug. 22, 1949.

terials. The shafts are on a catenary curve, closely following the dip of the ore body. Slope of the shafts at the collar is  $36^\circ$ , flattening to  $26^\circ$  at the bottom.

The footwall at Cornwall is a traprock which varies structurally from solid to highly fractured. Grizzly stations at the top of each chute raise were reinforced by setting timber sills on concrete. Occasionally it was necessary to concrete the entire chute raise. Concreting was done by using  $\frac{1}{2}$ -yd electric mixers set up in the breaking subs. Sand, stone, and cement were hoisted up through the chute raises to the breaking subs.

Late in 1937 it was decided to eliminate chute raises and breaking subs in future development. Haulage levels are 50 ft apart vertically. When ore is below a  $40^\circ$  dip, slushing drifts are driven normal to the haulages and on 40-ft centers. In a few areas, ore dips greater than  $40^\circ$  and slushing drifts are then placed parallel to the haulage drifts. Finger raises are opposite each other and on 20-ft centers making 20x20-ft draw areas (fig. 1). For loading



90-cu ft cars, 50 hp, two-drum scraping hoists and 54-in. scrapers are used. The lengths of slushing drifts vary from 90 to 200 ft, depending on dip of the ore.

The slushing drifts were timbered with 10x10-in. posts and 10x12-in. caps. Sets were spaced to make three spaces 4 ft-8 in. and one space 6 ft center to center of timber. Two additional framed sets were then placed inside the 6-ft space to make the finger station. The sides of the fingers were concreted to 12 in. above the drift timber and a heavy round timber placed on top of the concrete which acts as a bumper cap.



Until 1943, all concrete was mixed at the mine. Since then, ready-mixed concrete delivered from Lebanon, which is about 6 miles away, has been used exclusively. Other uses of concrete were for the shaft back, level landings and ore-pocket raises. A pneumatic ball type concrete placer was used for placing concrete in the shaft back and level landings.

**Concreting Slushing Drifts:** In all cases, timbered slushing drifts had to be repaired before being drawn empty. The timbering in many of the drifts had to be completely replaced and sometimes second repairs were required. (Fig. 2 shows a slushing drift that required extensive repair.)

In 1945 it was decided to concrete slushing drifts. It was expected that the initial cost of concreting against cost of timbered drifts would be higher, but if timber repairs were eliminated, considerable savings would result. The concrete would also eliminate the hazardous work of timber repair.

The following equipment was ordered and received in November of 1945:

1— $\frac{3}{4}$ cu yd pneumatic concrete placer	\$3,250.
1—lifting frame	400.
7— $\frac{3}{4}$ cu yd concrete hoppers	770.
7—5-ft sections collapsible steel concreting forms including 4 travelers	4,025.
1—20-ft rubber 6-in. concrete hose	400.
2—90° shooting boxes	300.
<b>Total</b>	<b>\$9,145.</b>

The linings of the first drifts were poured solid without finger openings. Mining of the finger openings through concrete was costly and it was decided to form out the finger openings.

The slushing drifts are mined 7 ft-6 in. wide and 8 ft-9 in. high. If the rock is good, fingers are mined in the ribs of the drift. Three 40-lb rails on wood ties are installed in the floor of the drift. The two outside rails are placed on 30-in. gauge to suit the carrying travelers of the steel forms. The three rails, later, make the scraper bottom of the drift.

Four, 5-ft sections of form placed on two travelers are used for each concrete pour. With fingers on 20-ft centers the joint is placed between fingers. The finger openings are formed using prefabricated wood on the side arc of the steel forms. Sides of the finger openings are concreted so that restriction is at the inside of the drift (42 in. wide by 54 in. high). A rail is placed across the top of the opening to give added strength; no other reinforcing is used.

The concreting cycle consists of pouring 20 ft one day, stripping and moving the forms up the next day, and pouring again the third day. Concreting is usually carried on in two adjacent drifts or two drifts on separate levels. This allows a 20-ft pour every day. The minimum thickness of concrete is 12 in. on the top and 9 in. on the sides.

Photographs illustrating some of the slushing drift concreting work are as follows: fig. 3 is a view looking up a 30 pct grade with forms in place, fig. 4 is looking down a 30 pct grade showing the pipe arrangement and forming at the back end.

When the first drifts were concreted, the placer was set up underground as close as possible to the job to be poured (fig. 5). Concrete was taken down the shaft in  $\frac{3}{4}$  cu yd hoppers. This procedure kept our material skips too busy and it was de-

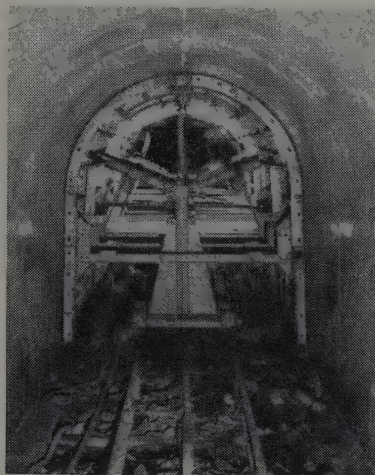


**Fig. 2—Slushing drift that required extensive repairs.**

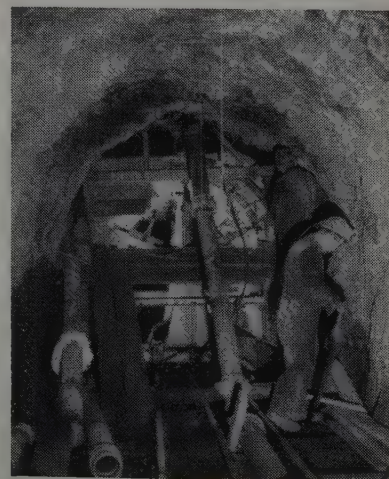
cided to set the placer on the surface. A second placer was purchased and set up on the level in which slushing drifts were being poured. A 6-in. pipeline was placed in the manway of our No. 5 shaft. This line at present runs to our 820-ft level and has take-off pipes at the 650, 700, 760 and 820 ft levels. (Level designation is vertical distance below shaft collar.) The placer underground handles slushing drifts on two haulage levels, blowing uphill to one and downhill through manways to the slushing drifts on level below.

Later it was decided to try using only one placer on the surface, blowing direct to the forms. In a slushing drift which was concreted recently, the concrete traveled 1350 ft down the shaft, 1316 ft on the haulage level and through a vertical distance of 30 ft to get into the upper end of the form, or a total of nearly 2700 ft. At this distance the concrete was somewhat dry and showed some segregation using a 1-2 $\frac{1}{2}$ -3 $\frac{1}{2}$  mix. A mix of 1-2-3 gave better results. Sand and stone are crushed limestone. Per bag of cement, 4 $\frac{1}{2}$  gal of water is used, with a wetting agent added.

Three sets of costs on completed concrete-lined slushing drifts are available to date and are given in table I.



**Fig. 3—View looking up 30 pct grade with forms in place.**



**Fig. 4—View looking down 30 pct grade showing piping arrangement and forming at back end.**

**Fig. 3 and 4—Slushing drift concreting work.**





Fig. 5—View of placer setup underground close to the job to be poured.

Table I. Costs of Concrete-lined Slushing Drifts

	1 550-ft Level 2 drifts	2 600-ft Level 4 drifts	3 600 and 650- ft Level 14 drifts
Lineal feet	162.0	335.5	1,388.5
Total manshifts	271	504.5	1,607.5
Cubic yards concrete	199	560.5	1,910.5
Manshift per foot	1.67	1.50	1.16
Cubic yards per foot	1.23	1.67	1.38
Labor cost per foot			13.73
Material cost per foot			14.15
Total cost per foot concrete			27.88
Total cost per cu yd concrete placed			20.20

(1) First two drifts were poured solid and were somewhat experimental causing high labor cost.

(2) Next four drifts were using two placers and included forming and concreting the finger openings. The first six drifts were 7 ft wide by 7 ft 6 in. high over top of rail.

(3) The size of the drifts was reduced to 6 ft wide by 7 ft high over top of rail. Ten of these fourteen drifts were poured using one placer on surface. Labor rates averaged \$11.84 per manshift. Concrete at present costs \$10.75 per yard at the shaft collar.

The average cost per foot of timbered drift including the cost of pouring finger opening by hand was \$33.45 per foot as compared with \$27.88 per lineal foot of concrete lined drift. Concreting the slushing drift has actually made a saving of \$5.57 per foot.

**Concreting Slushing Drifts in Heavy Ground:** A very good example of using concrete in heavy

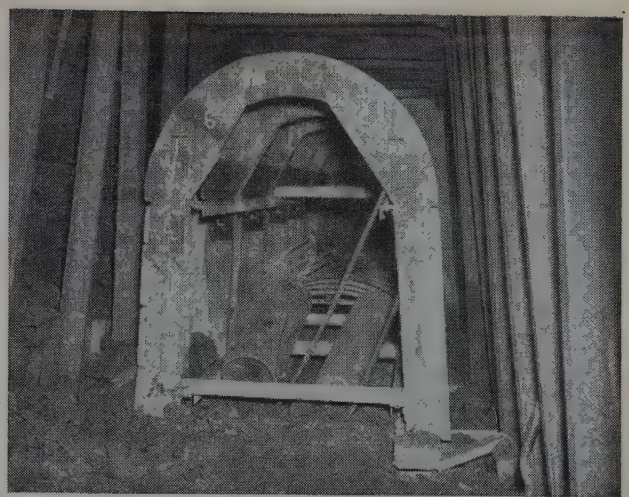


Fig. 7—Rail sets installed in place of timber and concrete forms placed.

ground occurred in 1948. Three 100-ft slushing drifts were driven in ore and limestone on 30-ft centers. Ten by 10-in. posts and 10x12-in. caps were placed on 2½-ft centers. The timber was placed while driving. Upon completion of the drifts, but before mining was started, the timber failed (fig. 6). The timber was taken out and rail sets were installed for a distance of 25 ft. Concrete forms were then placed (fig. 7) and a 24-ft section was concreted. This process was repeated until the drifts were finished (fig. 8). The inside size of these drifts was 4½ ft wide by 6 ft high. Special forms were used in this location because the forms had to be taken up 4-ft diam raises. The concrete held up well permitting 170,930 net tons of ore to be drawn from these three drifts.

**Lining of Ore Pocket Raises:** The concreting of ore pocket raises has been a major project at Cornwall. Each ore pocket serves three haulage levels. The ore pocket raises consist of a main raise 200 ft long and two branches. Raises were driven on a 55° slope and enlarged to 7 ft. Steel tubes 5-ft in diam made up of 1-in. steel plate on the bottom and ½-in. plate on the top were concreted into place. These wore out when approximately half the required tonnage had been drawn. The steel liners which were in 4 ft long sections loosened after the bottoms were worn through. This caused many dangerous hangups.



Fig. 6—Timber failure after drifts were completed and prior to mining.

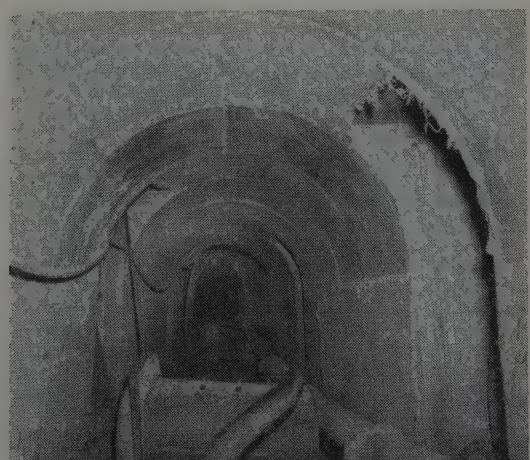


Fig. 8—Completed drift with rail sets and concrete forms.



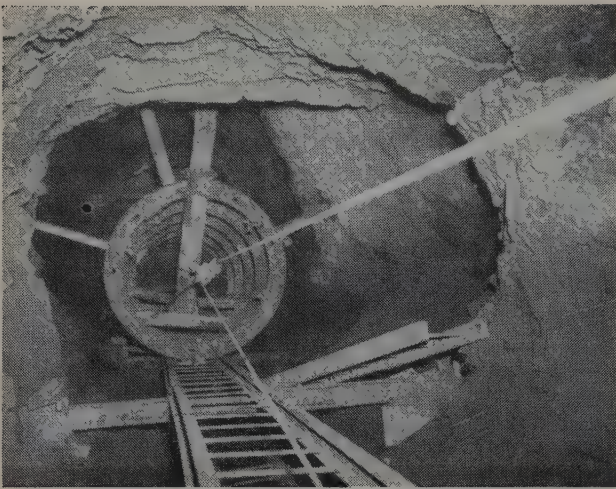


Fig. 9—View of concrete form in longest raise.

The latest loading station located on our 880-ft level has three separate raises emptying into a scraping trench 90 ft long. The scraping trench is 10 ft wide and 14 ft high. These raises will service the 700, 760 and 820-ft levels. It is expected that elimination of the branch raises and resultant lower tonnage through each raise will enable the concrete linings to handle the required tonnages. Fig. 9 is a view of the concrete form in the longest raise (200 ft) and shows the very weak rock structure. A rail ladder was used for alignment of 24 ft of collapsible 5-ft diam forms. An aggregate of trap rock and silicon sand was used instead of the usual limestone aggregate. The concrete was a minimum of 18-in. thick on the bottom and 9-in. on the top and sides (mix 1-2-3).

**Concreting Haulages:** During late 1948 and 1949 some work was done on concreting haulage levels. Although 650 ft has been concreted to date, costs were kept on only 150 ft. This 150 ft of concrete was placed in 240 ft of haulage driven with an arched roof section, while the balance was driven with a rectangular section for conventional timbering (table II).

Table II. Cost of Concreting Haulage Drifts

Lineal feet	150.
Total manshifts	85.
Total cu yd placed	232.
Manshifts per foot	0.57
Cu yd concrete per foot	1.55
Labor cost per lineal foot	6.75
Material cost per lineal foot	16.66
Total cost per lineal foot	23.41
Total cost per cu yd placed	15.10

Concreting of the haulage consists of lining 25 ft of drift and leaving 15 ft unlined for the slushing-drift loading station. This is repeated throughout the length of the drift. These unlined 15-ft sections are temporarily supported with one timber set blocked to the concrete. This timber is removed when loading stations are required.

For concreting haulage drifts there is very little hand forming required. The ends of the sections are closed off and in the center of each 25-ft section poured, a man safety station 3 ft wide by 2 ft deep by 6 ft high is formed. The size of the concreted drift is 8 ft wide by 7 ft-6 in. high over the top of the rail. Fig. 10 shows some completed haulage drift in addition to a concreted intersection.



Fig. 10—Concreted haulage drift and intersection.

The following several advantages in addition to those mentioned previously have become apparent as this work in concreting haulage and slushing drift progresses.

1. Less experienced timbermen are required for both original placement, and occasional repairs to finger openings.

2. Development of haulage and slushing drifts can be far in advance of that work which is actually required with no deterioration of the development workings. This can be especially advantageous during shutdowns.

3. The driving of both slushing and haulage drifts with an arched section has permitted the drifts to stand unsupported until they could be lined with concrete.

4. The fire hazard from timber is practically eliminated.

It is interesting to note that the same forms can be used for either small size slushing drifts or larger haulage drifts by changing the bottom plate of the form. When expanded for smaller width as it is used in the slushing drift, the sides are not vertical, giving a slight horse-shoe effect.

In conclusion, a few figures regarding yardage and costs are given. From January 1948 to the end of June 1949, 3752 ft of haulage and slushing drift was concreted using 5314 cu yd of concrete. No. 4 shaft extension and miscellaneous construction underground required an additional 2850 cu yd or a grand total of 8164 cu yd of concrete.

Frequently more than one job is poured during the same shift. A record daily pour of 92 cu yd was made including the blowing of 55 cu yd from the surface to two different levels and 37 cu yd taken into the mine in  $\frac{3}{4}$ -yd buckets to a placer setup underground and blown to the scraper trench construction.

To maintain our expected monthly tonnage, 210 ft of slushing drift and 80 ft of haulage must be concreted per month. This has easily been attained, since as much as 420 ft of haulage and slushing drift have been poured in a month.

At No. 4 mine an average level 2000 ft long would have 6000 ft of slushing drifts, requiring 10,200 cu yd of concrete costing \$196,500 in place. Based on an average height of 90 ft of ore, the cost for concreting the haulage and slushing drifts would be 8.2 cents per ton of ore.



# Radiotracer Studies on the Interaction of Dithiophosphate with Galena

by G. L. Simard, J. Chupak, and D. J. Salley

Radiotracers were demonstrated to be of considerable value in a study of the interaction of dithiophosphate with galena. The interaction had characteristics of both chemisorption and chemical reaction, being complex because of the nature of the galena surface. Sorption was rapid and markedly reversible, and flotation occurred at relatively small amounts of dithiophosphate sorbed.

**D**ITHIOPHOSPHATES and xanthates are the principal collectors for sulphide minerals, and consequently any knowledge of mineral-collector systems of this type is of value. In the present investigation an attempt was made to obtain information on the interaction of a typical dithiophosphate with galena. In carrying out this study, radioisotopes, so much discussed in the past few years,<sup>1,2</sup> were extensively employed. By using radioactive dithiophosphate synthesized from radioactive phosphorus, a rapid and sensitive analytical procedure

G. L. SIMARD is in the Research Division, Stamford Research Labs., American Cyanamid Co., Stamford, Conn.; J. CHUPAK, formerly in the Research Division, Stamford Research Labs., is now at Camp Detrick, Frederick, Md.; and D. J. SALLEY is in the Research Division, Stamford Research Labs.

AIME New York Meeting, Feb. 1950.

TP 2815 B. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Oct. 17, 1949.

was at hand. This permitted determination of such important quantities as the rate of uptake of dithiophosphate by the mineral, the amount existing at equilibrium on the mineral surface and in the solution, and the desorption of the agent from the surface. Such measurements were possible even on single crystals of galena. In addition the exchange of dithiophosphate between the solution and the sorbed phase was examined, a matter which could be accomplished only by the use of isotopes.

It is desired to point out at this time that throughout this paper the term "sorption" has been employed to designate the uptake of agent by mineral, without implication as to the nature of the process by which the uptake was accomplished.

## Experimental

**Materials: Galena:** The galena was from the Tri-State district. For studies on ground mineral four preparations were used during the course of the investigation. These were prepared from selected large crystals by wet grinding in order to reduce surface oxidation. Alcohol was chosen as a convenient medium for this purpose. The ground mineral was then fractionated by sedimentation in alcohol, dried by evacuation, and stored in a nitrogen-filled desiccator. Data on the preparations are tabulated in table I. The size analysis indicated that the areas of the preparations were of comparable

magnitude, even though the absolute values may be somewhat incorrect.

**Dithiophosphates:** Nonradioactive dithiophosphate (di-isopropyl or di-secondary butyl) was obtained by purification of a commercial product. An aqueous acid solution of the agent was extracted with petroleum ether, the ether layer dried, and the dithiophosphate precipitated as ammonium salt with anhydrous ammonia. Several repetitions of this process resulted in a nearly colorless, flaky product of good purity. (General formula  $(RO_2)PSSNH_4$ ).

Radioactive dithiophosphate was synthesized by heating radioactive elementary red phosphorus\*

\* For early experiments (1943 to 1944), the elementary red phosphorus was obtained from the cyclotron group at the Crocker Radiation Laboratory of the University of California, Berkeley, through the courtesy of Dr. Joseph W. Hamilton. For more recent work (1946 to 1949), the elementary radioactive phosphorus has been supplied by the Oak Ridge National Laboratories on allocation from the U. S. Atomic Energy Commission.

with sulphur at 270° to 300°C to produce radioactive  $P_{25}$ . The latter was then treated with the desired purified alcohol (isopropyl or secondary butyl) at 60° to 80°C to form the dithiophosphoric acid derivative. Purification was effected in the same manner as for the nonradioactive material. The effectiveness of the purification method was established by the isotopic dilution method.<sup>3</sup> The same technique was used to show that only negligible decomposition of neutral or of carbonate solutions of the dithiophosphate took place over a period of a day; this was true whether or not galena was suspended in the solutions.

**Procedures: Radioactivity:** The activity of a solution was obtained by counting with a small glass-jacketed, silvered Geiger counter, using a conventional scaling circuit.<sup>4</sup> Crystals and other solids were counted under a bell-shaped, mica-window counter tube. Sufficient counts were made so that the probable error of counting was of the order of  $\pm 1-3$  pct.

The specific activity in terms of counts per minute

Table I. Data on Galena Preparations

Galena No.	Size, Bulk by Volume Micron	Area <sup>a</sup> cm <sup>2</sup> per g	Pb, Pct
2	2.5-20	1,800	85.7
3	10-30	950	85.1
4	10-20	1,130	85.7
5	5-25	1,100	86.0

<sup>a</sup> By Fischer sub-sieve sizer, except for No. 2, which was from microscopic size analysis. Cu, Fe, and Ag below 0.01 pct in all cases.

per mol was measured by determining the activity of a solution of known concentration. For solids, the specific activity was obtained by drying and counting an aliquot of a solution of known concentration. With the counters used, the specific activity of the most active dithiophosphate preparations was of the order of  $10^{+18}$  counts per minute per mol, thereby allowing detection as low as  $10^{-12}$  mols.

**Sorption:** To determine the uptake of agent by ground mineral, a weighed sample (0.85 to 1.0 g) was shaken in air for a specified time with a given volume of solution of known initial concentration of the radioactive agent. The mixture was then centrifuged (which presumably stopped the sorption process), the clear supernatant solution was pipetted out, and then was drawn through a cotton wad into the counter tube. The observed count and the known specific activity of the radiodithiophosphate allowed calculation of the residual concentration of the dithiophosphate. The difference in amount present before and after the experiment gave the quantity of agent taken up by the mineral.

In certain instances, the ground mineral sample was leached with water before a sorption measurement. The leaching was carried out merely by shaking with 30 to 50 cc of distilled water for a 30 to 60-min period, centrifuging, and pipetting off the supernatant as completely as possible. The sorption procedure was conducted immediately without intermediate drying of the galena.

Although most of the runs were carried out in the air, some trials were made in nitrogen. In these cases, boiled solutions stored under nitrogen were used, and the air in the dead space of the shaking tubes was displaced by nitrogen.

For sorption studies on galena single crystals, well-formed crystals with faces about 1 to 2 cm square were selected. A crystal was freshly cleaved and both halves were immediately immersed in a

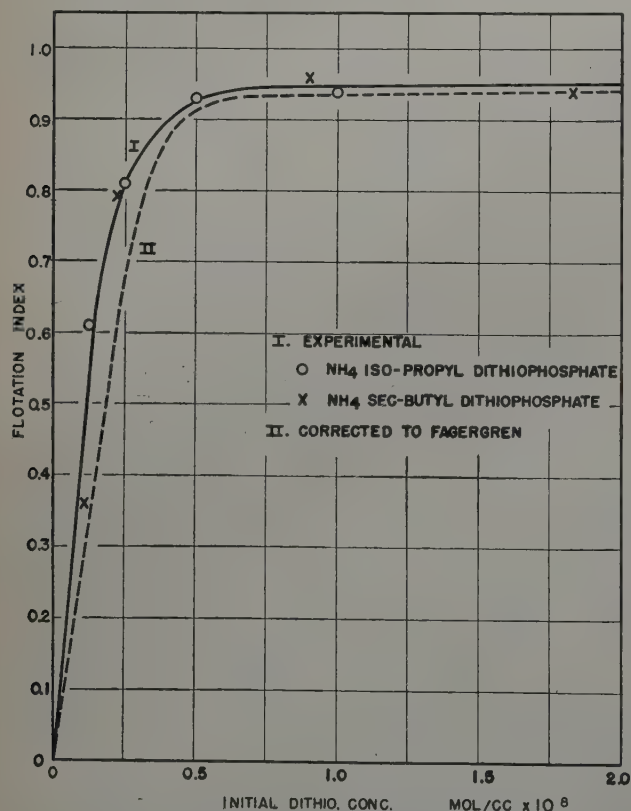


Fig. 1—Flotation curves of galena No. 2.  
( $20 \times 10^{-8}$  mol per cc  $\text{Na}_2\text{CO}_3$ ).

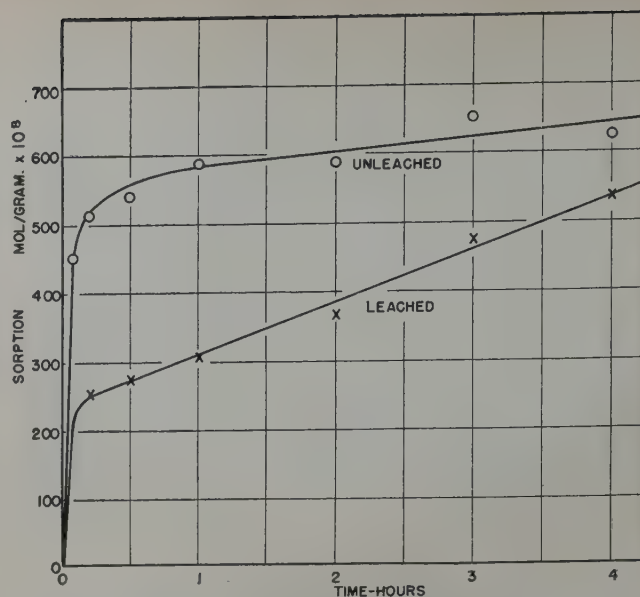


Fig. 2—Rate of uptake of dithiophosphate on leached and unleached galena No. 5.

Initial concentration: dithiophosphate— $50 \times 10^{-8}$  mol per cc.  
Temperature,  $25^\circ\text{C}$ . ( $20 \times 10^{-8}$  mol per cc  $\text{Na}_2\text{CO}_3$ ).

solution of the radiodithiophosphate for a specified time, after which they were removed and sponged dry immediately with Kleenex tissue. Crystals which gave irregular faces or cleavage were rejected. By use of a lead mask having a central hole, a nominal area of crystal was defined for count. Since the amount of material taken up by the crystal was relatively small, no sensible decrease in the concentration of the solution occurred.

**Desorption:** For desorption measurements with ground mineral the uptake of radiodithiophosphate was first carried out by the usual procedures, and then immediately a given volume of sodium carbonate solution ( $20 \times 10^{-8}$  mol per cc) was added to the moist galena. After shaking this for a period, the amount of dithiophosphate desorbed was obtained from the count which appeared in the carbonate solution.

In measuring desorption from single crystals, the crystal bearing the radiodithiophosphate was merely submerged in water for a specified time, removed, dried with Kleenex, and counted.

**Exchange:** For an exchange measurement\* be-

\* For a general discussion of exchange, see p. 250 of ref. 1.

tween dithiophosphate sorbed on galena and dithiophosphate ions in solution, the usual procedure for measuring the consumption of dithiophosphate was first performed. After determining the concentration of the radiodithiophosphate solution left from the sorption run, a nonradioactive solution of identical concentration and volume was added to the galena. This mixture was shaken for a specified time and the count appearing in solution was measured. From the knowledge of the amounts of dithiophosphate present in each phase, a calculation was made of the count expected in solution if statistical distribution of the radioactive molecules between the sorbed and the solution phase had taken place. The ratio of the observed count to that expected gave the so-called "fraction exchanged."

In carrying out exchange between lead dithiophosphate and dithiophosphate ions, a freshly precipitated lead dithiophosphate was first formed by mixing lead nitrate with radiodithiophosphate. The



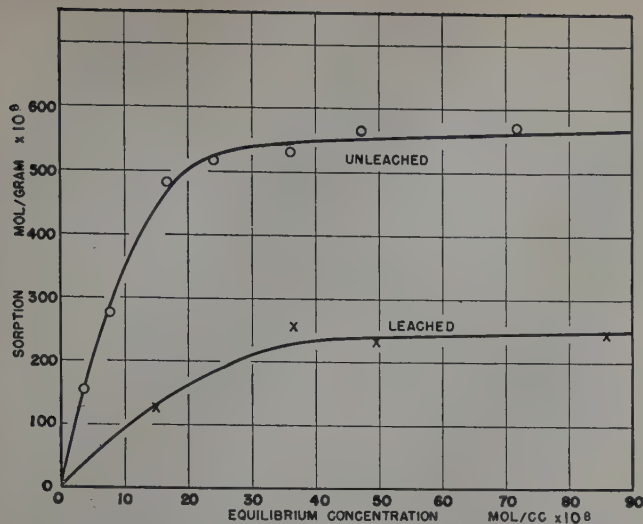


Fig. 3—Sorption isotherms of dithiophosphate on leached and unleached galena No. 5.

12 min sorption time. Temperature 25°C. ( $20 \times 10^{-8}$  mol per cc  $\text{Na}_2\text{CO}_3$ ).

washed precipitate was then suspended in a solution of known concentration of inactive dithiophosphate, stirred for a specified time, centrifuged, and the count which had appeared in the supernatant determined. Calculation of the fraction exchanged was again made by comparing the observed count with that expected on the assumption of complete statistical distribution of radioactive molecules between both phases.

**Flotation:** For flotation experiments a pneumatic flotation cell designed to operate on one-gram samples was constructed, using a fritted-glass disk as the air-dispersing element in a manner similar to that described by Herd and Ure.<sup>5</sup> The cell above the fritted disk was 3 cm in diameter and 5 cm in depth. In making an experiment, 25 cc of the solution to be tested was placed in the cell with the gram sample of the mineral. (On certain occasions 0.4 ppm of octyl alcohol was added as a frother.) A slow stream of air, insufficient to cause overflow, was then passed through the mixture for a conditioning period (usually 10 min). The air velocity was increased in order to produce flotation and the floated mineral was collected on a filter disc, dried, and weighed. Flotation was expressed by "flotation index":

$$\text{F.I.} = \frac{\text{weight of mineral floated} - \text{weight of blank floated}}{\text{weight of sample} - \text{weight of blank floated}}$$

The relationship between the flotation index values for the one-gram pneumatic cell and for a 100-g Fagergren machine was determined by making comparative tests. Fig. 1 shows typical flotation curves for the small pneumatic cell, as obtained experimentally and as corrected to conform to the Fagergren.

### Results

The following sections outline the results obtained on the rate of uptake of dithiophosphate from solu-

Table II. Calculated Constants for the Isotherms

Galena, No.	Maximum Uptake, Mol per g $\times 10^8$	$a$ Cc per g	$n$
4	800	2.06	0.81
5 (unleached)	560	1.21	0.79
6 (leached)	250	1.25	0.89
2	(70) <sup>a</sup>	0.57	0.92
Single crystal	(50) <sup>b</sup>	0.012 <sup>c</sup>	0.70

<sup>a</sup> The highest value observed; the isotherm was not carried to the point where it leveled off.

<sup>b</sup> For 1000 cm<sup>2</sup> per g on the basis of  $0.05 \times 10^{-8}$  mol per cm<sup>2</sup> at a concentration of  $100 \times 10^{-8}$  mol per cc.

<sup>c</sup> On the basis of 1000 cm<sup>2</sup> per g.

tion, on the isotherms, on the desorption, and on the exchange, and include work on both ground mineral and single crystals.

All measurements were made in solutions buffered with  $20 \times 10^{-8}$  mol per cc of sodium carbonate, unless otherwise specified. The pH was constant at  $7.8 \pm 0.1$  over a dithiophosphate concentration range of 0.5 to  $75 \times 10^{-8}$  mol per cc. It is of interest that tracer measurements made with radiocarbonate indicated that little or no carbonate ion could be removed by galena in the presence of dithiophosphate, although in its absence some carbonate abstraction (namely  $140 \times 10^{-8}$  mol per g on galena No. 4) was possible. These findings were qualitatively confirmed by the fact that X ray diffraction examination showed no carbonate present in the precipitate formed on adding lead ion to dithiophosphate solution buffered with carbonate. In the absence of dithiophosphate, basic lead carbonate was precipitated.

**Rates:** Curves of typical form for the rate of sorption of dithiophosphate from solution by ground galena are shown in fig. 2 for leached and unleached samples of preparation No. 5. Both curves show that there occurred a rapid initial uptake which soon terminated on unleached galena but which continued at a much reduced rate, sometimes negligible, on the leached. This secondary process was absent when the experiments were conducted in nitrogen instead of air. Many additional runs with other galena preparations and other solution concentrations showed that the rapid initial process was 90 to 100 pct complete in the first 10 to 12 min although the total quantity taken up in that period varied widely. It was clearly evident that, for the times usually involved in flotation, the fast initial process was the one of interest. Evident also was the fact that a large amount of agent was consumed by the leachable coating of galena.

**Isotherms:** The dependence of the amount of dithiophosphate taken up on the concentration of the solution is shown in fig. 3 for unleached and leached mineral, again from batch No. 5. Since these data were taken for a 12-min period, on the basis of the previous rate studies they are concerned only

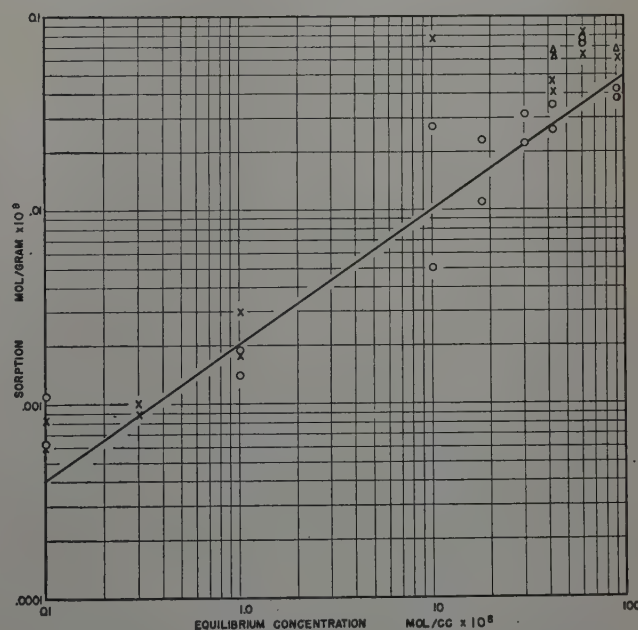


Fig. 4—Uptake of dithiophosphate on single galena crystals.

10 min sorption time. Temperature 25°C.

**Table III. Desorption of Dithiophosphate from Unleached Galena No. 4**

Time of uptake, 12 min  
Time of desorption, 20 min  
Desorption into  $20 \times 10^{-8}$  mol per cc  $\text{Na}_2\text{CO}_3$

Sorption		Desorption	
Solution Conc., Mol per Ce $\times 10^8$	Dithiophos. Taken Up, Mol per G $\times 10^8$	Dithiophos. Desorbed, Mol per G $\times 10^8$	Conc. of Resulting Solution, Mol per Ce $\times 10^8$
3.2	42	19	0.8
12.1	170	71	3.0
30.5	468	101	4.3
46.9	613	90	3.8
56.4	804	85	3.6

**Table IV. Desorption of Dithiophosphate from Single Crystals**

Time of uptake, 10 min  
Desorption into  $\text{H}_2\text{O}$

Sorption		Desorption	
Solution Conc., Mol per Ce $\times 10^8$	Dithiophos. Taken Up, Mol per $\text{cm}^2 \times 10^8$	Time, Min	Dithiophos. Left on Surface, Mol per $\text{cm}^2 \times 10^8$
10	0.0083	7	0.0012
30	0.062	2	0.041
		7	0.029
		22	0.023
		16 hr	0.003

with the rapid initial process. The figure also serves to bring out clearly the larger uptake by the unleached mineral.

For all the galena preparations the isotherms were of the same type but the magnitude of the uptake was variable. They were conventional in shape, rising initially as the concentration increased but eventually leveling off. Although some could be fitted to an equation of the Langmuir type, it was found that in general the initial portion, up to  $20$  to  $40 \times 10^{-8}$  mol per cc equilibrium concentration, gave reasonably straight lines on a log-log plot; above this concentration region, the amount taken up was practically independent of concentration.

The fact that the initial portion fitted a log-log plot is expressed by the usual Freundlich equation<sup>6</sup>:

$$M = aC^n$$

where  $M$  is the mols taken up per gram,  $C$  is the concentration, and  $a$  and  $n$  are constants. It appeared that the value of  $n$  was sensibly constant ( $\pm 10$  pct) for all galena samples but factor  $a$  was altered; table II gives values which were calculated. The constancy of  $n$  indicated that the curves were all essentially of the same form, which suggested that although the preparations were quantitatively different they were qualitatively the same.

**Single Crystals:** Experiments of the type described above show in an indirect way the quantity of agent taken up by the mineral. It was one of the advantages of the radiotracer method that its sensitivity permitted direct measurement of the amount of agent on the surface of a single galena crystal  $0.2$  to  $1 \text{ cm}^2$  in nominal area. For this purpose, freshly-cleaved, single galena crystals were employed, and the results are shown in fig. 4 on a log-log plot of the amount of dithiophosphate (ammonium diisopropyl) taken up per  $\text{cm}^2$  versus the concentration of the solution (unbuffered).<sup>\*</sup> It is noteworthy

<sup>\*</sup> Mention may be made here that a few similar experiments were carried out on single crystals of sphalerite. Within the limit of error no agent was consumed unless the mineral had been first activated in copper sulphate. Although this was as expected, the radiodithiophosphate used was unfortunately of low specific activity so that small amounts which might have been taken up by the unactivated sphalerite would not have been detected.

that a thousandfold concentration range was covered, and that definite amounts of dithiophosphate were taken up by the galena even from solutions of extremely low concentration. The results scatter very considerably, not even the complementary faces

of a given cleaved crystal agreeing in many instances. This scatter may have been due in part to uncertainties in the area counted and to cracks, and so on, although it is thought that the macro area represented the true area within a factor of two. In general, however, the scatter was correlated with the presence of stained areas on the freshly cleaved faces, for high radioactive counts were always associated with visible stains. In spite of the scatter, it is evident that the data here, as with the ground mineral, can be approximately represented on the log-log plot by a straight line. Of considerable interest is the fact that here also the value of the exponent,  $n$ , approximates that obtained in the case of the ground minerals, as is shown in table II, although the value of  $a$  is again different.

**Desorption:** In addition to the above studies on the uptake of dithiophosphate, an investigation was made of the desorption of the consumed dithiophosphate from the galena, both for ground galena and for single crystals. Here again, it may be emphasized that without the radioactivity method, experiments such as these could scarcely have been accomplished. Results are recorded in table III which includes the conditions under which the uptake was effected as well as the amount desorbed and the concentration of the resulting solution. Except for the one instance where the starting amount was small, it is evident that the quantity desorbed was independent of the amount initially on the surface. Time of sorption, up to 60 min at least, did not alter this result. Whether or not greater desorption would have occurred had the time been extended is uncertain; experiments on another galena showed approximately the same amount given up in 3 hr as in 20 min. In the latter case, however, the final concentration of the desorption solution was about  $16 \times 10^{-8}$  mol per cc, about 4 or 5 times larger than obtained in the trials of table III.

Typical desorption results from single crystals are given in table IV. They show that the agent was partially desorbed from the surface in the time permitted. Qualitatively, therefore, these experiments are in line with the results with ground mineral.

**Table V. Exchange of Dithiophosphate between Solution and Sorbed Phase**

Galena No. 4  
( $20 \times 10^{-8}$  mol per cc  $\text{Na}_2\text{CO}_3$ )

Dithiophos. Conc., Mol per Ce $\times 10^8$	Total Mols Present, $\times 10^8$		Exchange Time, Min	Fraction Exchanged
	In Solution	On Galena		
17.0	339	674	2	0.46
7.2	143	410	2	0.67
4.5	90	189	2	0.78
4.3	86	196	10	0.92

**Exchange:** The exchange of dithiophosphate ion in solution with radiodithiophosphate that had been taken up by galena was examined to provide further information on the nature of the interaction product. This is a type of experiment which is only possible by tracer techniques. Table V records some typical data on the "fraction exchanged" (ratio of observed count to count expected at statistical equilibrium). It is evident that exchange was not instantaneous but required 10 min or more for completion. None the less it is clear that a dynamic exchange of ions between solution and sorbed phase was occurring during times of the same order of magnitude as are involved in the rapid initial sorption process and in flotation. This general finding was confirmed by reverse exchange experiments, in which inactive di-



Table VI. Exchange of Dithiophosphate with Lead Dithiophosphate

Dithiophos. Conc., Mol per Ce $\times 10^6$	Total Mols Pres- ent, $\times 10^6$		Exchange Time, Min	Fraction Exchanged
	In Solution	In Precip.		
5.4	107	468	10	0.24
5.4	107	221	10	0.23
5.4	107	103	10	0.20
5.3	80	29	10	0.25

Table VII. Effect on Dithiophosphate Concentration on Flotation

(20 $\times 10^{-8}$ mol per cc $\text{Na}_2\text{CO}_3$ )		
Equil. Conc., Mol per Ce $\times 10^8$	Dithiophos. Taken Up, Mol per G $\times 10^8$	Flotation Index
Galena No. 2		
0.16	0.5	0.7
0.35	0.7	0.9
Galena No. 5 (leached)		
0.64	4.2	0.7
1.4	8.0	0.9

thiophosphate was present on the galena and radio-dithiophosphate ions were in the solution.

In order to view the above results in better perspective, similar exchange experiments were carried out with freshly-precipitated, lead radiodithiophosphate and dithiophosphate ion; table VI shows representative results. The data were necessarily obtained at higher total mols in the precipitate than existed on the galena but the ratios of the mols in solution to those in the precipitate approximated those of table V. It is evident that appreciable exchange did occur between freshly-precipitated, lead dithiophosphate and dithiophosphate ion in solution. Although it would appear that the rate was somewhat slower, it is difficult to make a direct comparison of the rates since the extrapolation of the findings here at high concentration to those at the lower concentrations prevailing in the experiment with galena is made uncertain by the complexity of heterogeneous exchanges of this type.<sup>7</sup>

**Flotation:** Variations in flotability of the different mineral preparations were found, although the procedure employed gave reproducible values of flotation index for a given batch. The notable point was that nearly complete flotation was accomplished at low solution concentration and at low amounts of agent taken up. Table VII illustrates dithiophosphate quantities found necessary to produce flotation index values of 0.7 and 0.9 for two galena preparations.

#### Discussion of Results

Numerous investigations attest to the chemical heterogeneity of the galena surface.<sup>8</sup> Either in air or in solution galena can oxidize to compounds of the general type  $\text{PbSxOy}^9$ . In support of this, electron diffraction patterns made during this investigation on weathered single crystals identified the surface product as  $\text{PbSO}_4$ . Very probably a galena surface can vary from a monolayer of oxygen all the way to several layers of lead sulphate. The results presented above on the interaction of galena with dithiophosphate seemed in accord with this picture, for they were strongly dependent upon the state of the galena surface. Variable uptake of dithiophosphate with weathering, the greater sorption of unleached over leached mineral, and the increase in uptake by stain on single crystals all point in this direction. It is most probable that an oxidized surface was always involved, even with leached galena or with freshly-cleaved single crystals, and that at no time was a clean, lead-sulphide surface at hand.

Table VIII. Estimated Limiting Concentrations

Galena	Pb Available (Assumed), Mol $\times 10^8$	Initial Carbonate, Mol $\times 10^8$	Estimated Limiting Conc., Mol per Ce $\times 10^8$	Lowest Conc. Investigated, Mol per Ce $\times 10^8$
Single crystals	25	0	0.8	0.1
Galena No. 2	100	400	5	1.5
Galena No. 4	450	400	0.5	
Galena No. 5	300	400	0.7	

As mentioned previously, however, similarity in form of the isotherms for all the galena preparations, regardless of the differences in the amounts taken up, suggests that the heterogeneity was one of degree rather than of kind. Finally, the general shape of the isotherm may be interpreted as arising from interaction with a number of surfaces, each of slightly different properties and each present in varying amounts from preparation to preparation. In view of this surface heterogeneity, it is evidently very unlikely that the results can be satisfactorily represented by any one theory. The considerations made in the following are confined to the rapid initial interaction completed in 10 to 12 min.

The observed uptake of dithiophosphate by galena was not a physical adsorption on the galena surface resulting from van der Waal's forces or like causes. Such physical adsorption is in general characterized by very rapid rates of sorption and desorption, and by practically instantaneous exchange between the adsorbed and the solution phases. None of these characteristics were observed here. On the contrary, the rate of uptake, although fairly rapid, was finite and measurable, the desorption was slow and incomplete, and the exchange required a time comparable to the sorption time. Accordingly the interaction was either some kind of a chemisorption or chemical reaction at the surface.

Formal distinction between chemisorption and surface chemical reaction is difficult since the interaction forces of chemisorption are of the same type as are concerned in chemical reaction. Both require activation energies and proceed at finite rates. Chemisorption is usually considered to involve no more than a monolayer, and the term will be used in this sense here. On the other hand, a sorptive process will be defined as one of chemical reaction when an isotherm of the form required by the phase rule is obtained, namely no sorption occurring below that limiting concentration fixed by the normal solubility product of the compound formed.

Assuming that lead dithiophosphate was the only product formed as a result of a possible chemical reaction between dithiophosphate and galena, an attempt was made to apply solubility product considerations.\* The difficulty in such an application

\* R. K. Madison of the Laboratories, using radiolead as an indicator, found the solubility of lead sec.-butyl dithiophosphate was  $0.37 \times 10^{-5}$  mol per liter of water, pH 6 at 25°C. The solubility determination using radioactive dithiophosphate as indicator gave a figure of  $0.53 \times 10^{-5}$  mol per liter, in satisfactory agreement. The solubility product calculated therefrom is  $0.37 \times 10^{-15}$  (mol per liter)<sup>3</sup>. In carbonate solution,  $20 \times 10^{-5}$  mol per liter, a concentration of dithiophosphate ion of  $7 \times 10^{-5}$  mol per liter was observed. This is almost exactly the value calculated from the observed solubility of lead dithiophosphate in water and the known solubility product of lead carbonate,  $0.16 \times 10^{-10}$  (mol per liter)<sup>8,10</sup>.

here lay in the fact that the amount of lead available for reaction was uncertain. Table II suggests that since the total amount of dithiophosphate taken up by the various galenas was 50 to  $800 \times 10^{-8}$  mol per g, 25 to  $400 \times 10^{-8}$  mols of lead was available, in 20 cc of solution. Using these figures as representative of the maximum possible lead-ion concentration, the limiting dithiophosphate-ion concentration fixed by the solubility product was calculated; the assump-



tion was made on the one hand that no carbonate formed and on the other that lead carbonate and lead dithiophosphate co-precipitated. The results are shown in table VIII. Unfortunately, only with galena No. 2 and with single crystals were measurements made at low enough concentrations to permit comparison between calculation and experiment. Although definite conclusions cannot be drawn in view of the uncertainties of the calculations, the fact that uptake was observed below the estimated limiting concentration suggests that the formation of lead dithiophosphate was at least not the sole process. This is supported by the fact that the sorption isotherms were not of the form defined for simple reaction. The flotation curve of fig. 1 supplies additional evidence by showing that measurable flotation, and therefore sorption, took place at concentrations as low as  $0.1 \times 10^{-8}$  mol per cc. From these considerations, it does not appear that only lead dithiophosphate was produced by the interaction of dithiophosphate with galena during the initial 12-min period.

Chemical reaction, as defined, cannot be entirely excluded, however, for the following shows that chemisorption cannot be the sole process occurring. Using  $70 \text{ \AA}^2$  as the projected area of a di-sec-butyl dithiophosphate ion, as indicated by Fischer-Hirschfelder models, a monolayer would be equivalent to  $0.025 \times 10^{-8}$  mol per  $\text{cm}^2$  or  $25 \times 10^{-8}$  mol per g for a surface of  $1000 \text{ cm}^2$  per g. Table II indicates that amounts much larger than required for a monolayer were taken up, and consequently positive support is lacking for a theory that a monolayer formed. Further, the desorption data suggested that a product had formed which, in  $20 \times 10^{-8} \text{ Na}_2\text{CO}_3$ , was in equilibrium with a dithiophosphate-ion concentration of 4 to  $16 \times 10^{-8}$  mol per cc. The equilibrated value for lead dithiophosphate was  $7 \times 10^{-8}$  mol per cc in carbonate solution. This may indicate that lead dithiophosphate existed after 20 min, but the data cannot establish its formation as the sole primary product in the interaction.

It was not possible from the exchange rates to clarify further the nature of interaction. If exchange had occurred with the sorbed phase on the mineral and not with the lead dithiophosphate, then the obvious conclusion could be drawn that the interaction did not form the lead dithiophosphate. Since appreciable exchange occurred in both cases no clear conclusion was possible.

From the foregoing discussion, the interaction of dithiophosphate with galena appears to have characteristics of both chemisorption and chemical reaction. Its complexity probably lies in the heterogeneous nature of the galena surface. This may be visualized by supposing that the overall interaction is a composite of separate interactions of dithiophosphate with lead sulphide and all its oxidized forms. Each interaction must be controlled by a definite activation energy for a process having a definite, free-energy change. The surface at the end of the rapid initial sorption process is thus also complex and heterogeneous. Therefore, no single mechanism will account for the observed overall interaction. In this system, at least, these findings appear to be at variance with the generalized theory of chemical reaction<sup>10</sup> or for that of adsorption<sup>7</sup> for the interaction in mineral-collector systems.

The relation of the interaction to the flotation results is of interest. The speed of the sorption process appeared to be fast enough so that, in flotation times normally employed, a significant amount of sorption

could occur. The process was quite irreversible. It was also brought out that flotation was achieved at relatively low concentrations of dithiophosphate and relatively small amounts taken up. These latter were lower than estimated for a monolayer on the nominal galena surface. Evidently, therefore, if the particles of galena were sufficiently small, only a partial covering was necessary to impart to them a sufficiently hydrophobic character.<sup>12</sup> The fact that these low concentrations were operative indicated that the interaction products adhered to the galena surface. Even though part of the weathered coating could be leached, apparently this did not result in a loss of the dithiophosphate for flotation.

### Summary

Radiotracers were demonstrated to be of considerable value in the study of a typical mineral-collector system, dithiophosphate-galena. Since they provide a rapid and sensitive analytical method, they are especially applicable to the conditions of flotation.

For the chosen system, measurements were made of the rate of uptake, the isotherms, the desorption, and the exchange of dithiophosphate on both ground mineral and single crystals.

It was concluded that no single mechanism could account for the observed overall interaction. The process was complex because of the heterogeneous nature of the galena surface. The interaction was not one of physical adsorption, but appeared to have characteristics of both chemisorption and surface chemical reaction.

It was shown that sorption proceeded fast enough so that significant uptake of dithiophosphate occurred within times normally employed in flotation. The process was quite irreversible. Flotation was achieved at relatively low concentrations of agent and at relatively small amounts sorbed.

### Acknowledgment

We wish to express our appreciation for encouragement and support from S. J. Swainson and R. B. Barnes, for helpful discussion with R. B. Booth, S. A. Falconer, J. E. Carpenter, and for technical assistance from various other members of these Laboratories during this investigation.

### References

- <sup>1</sup> G. T. Seaborg: *Chem. Rev.* (1940) **27**, 1.
- <sup>2</sup> A. Gaudin, P. de Bruyn, F. Blocher, and C. Chang: *Min. and Met.* (1948) **29**, 432.
- <sup>3</sup> F. Paneth: *Radioelements as Indicators*. 1928, New York. McGraw-Hill Book Co. F. Henriques and C. Marguetti: *Ind. and Eng. Chem. Anal. Ed.* (1946) **18**, 476.
- <sup>4</sup> R. B. Barnes and D. J. Salley: *Ind. and Eng. Chem. Anal. Ed.* (1943) **15**, 4.
- <sup>5</sup> H. Herd and W. Ure: *Jnl. Phys. Chem.* (1941) **45**, 93.
- <sup>6</sup> S. Brunauer: *Adsorption of Gases and Vapors*. p. 54, 82. 1943, Princeton Univ. Press.
- <sup>7</sup> F. Paneth: *A. Elektrochem.* (1927) **28**, 113; L. Imre: *Physikal. Chem.* (1931) **135A**, 262; *Trans. Faraday Soc.* (1940) **35**, 758; *Kolloid Zeit.* (1944) **106**, 39.
- <sup>8</sup> I. Wark: *Principles of Flotation*. 1938. Australian Inst. of Min. and Met., Inc., Melbourne, Australia.
- <sup>9</sup> A. F. Taggart, T. C. Taylor, and A. F. Knoll: *Trans. AIME* (1930) **87**, 217. E. Plante and K. Sutherland: *Trans. AIME* (1949) **183**, 160; *Min. Tech.* (Jan. 1948) TP 2297.
- <sup>10</sup> A. Seidell: *Solubilities*. 3rd Ed. 1940, p. 1382. D. Van Nostrand Co.
- <sup>11</sup> A. F. Taggart and M. D. Hassialis: *Trans. AIME* (1946) **169**, 259; also A. F. Taggart, T. C. Taylor and A. F. Knoll, ref. 9.
- <sup>12</sup> A. M. Gaudin and G. S. Preller: *Trans. AIME* (1946) **169**, 255.



# Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units

Although determining and controlling specific gravity of operating medium in a heavy-media plant manually presents no problem, there are advantages to automatic recording and control. The two installations discussed employ the usual method of measurement. Pneumatic control with band control and reset features is used to regulate dilution water and hence specific gravity. Difficulties have been mechanical in nature and associated with obtaining a proper sample and maintaining tight air connections in bubble-tube piping.

by James J. Bean

THE task of measuring the specific gravity of the operating medium in a heavy-media separation system has never presented a particularly difficult problem because the medium is fairly stable and the overflow of the separatory vessel, as well as its underflow, can be sampled easily and accurately and the specific gravity of the suspension determined easily by weighing a known volume. However, while this method is simple and accurate it does require the operator to take the sample by hand and to weigh it and there is considerable temptation to avoid the periodic sampling if everything seems to be going well, or if something is occupying the attention of the operator. Furthermore all operators do not sample in exactly the same manner and considerable practice is required for two operators to be able to "check" each other to the last few hundredths, particularly if the sample is cut underneath the drainage screen where location of the point of sampling and load on the screens tends to influence the determination.

While none of the above presents much of a problem, we have all recognized that some mechanical method of continuous measurement and recording would be advantageous since the operator would merely have to glance at the meter to check the gravity and to have an indication of the trend of any changes. Also if the instrument were of the recording type, a permanent record would be available for the guidance of the superintendent.

The Eagle-Picher Mining and Smelting Co. was the first heavy-media user to actually install such a recording meter. In 1946 they installed in their Central Mill at Cardin, Okla., a specific gravity recorder manufactured by the Bristol Co. of Waterbury, Conn. R. A. Barnes, of the Bristol Co., working with E. H. Crabtree, Jr. and Elmer Isern, of Eagle-Picher, made the application and worked out the problems of sampling and measuring. Their attempts to measure the specific gravity of the medium in the cone itself were not entirely successful and they resorted to an outside sample tube for actually making the determination. Because of the particular flowsheet used, it was possible to tap off from the medium return pipeline a stream of medium and divert it into the sampling tube, which was provided with a constant level overflow and a spigot

---

JAMES J. BEAN, Member AIME, is Mineral Dressing Engineer, American Cyanamid Co., Stamford, Conn. AIME Columbus Meeting, September 1949.

TP 2831 B. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received July 1, 1949.

underflow, and into which the bubbler tubes dipped.

The Eagle-Picher installation was successful and its possibilities were recognized by the Mineral Dressing Laboratory of the American Cyanamid Co. It was decided to install a similar unit in the heavy-media pilot plant to investigate further its possibilities. Chief among these was the continuous record which it was felt would be proof of the steadiness of the gravity in a heavy-media cone, something which is not always appreciated by potential users.

Because the heavy-media pilot plant is required to operate at a wide range of specific gravities, it was realized that the unit would have to record all gravities from 1.25 to 3.50, and do it to the nearest 0.01. It would not be necessary to record all of this wide range on a single chart and the method selected was to have 4 bands, each band range overlapping the other a small amount and calibrated so that with standard charts one division would represent 0.01 sp gr. A shift from one band to another could be arranged without alteration of the instrument itself, being accomplished by a simple change in the bubble-tube lengths, as described later. Accordingly, a recording type instrument was purchased and installed.

Because there were some advantages in doing so, the first installation attempted to measure the gravity of the cone proper by placing the bubble tubes in the cone. This was not at all satisfactory and the second scheme utilized a fixed vertical screen at the surface of the cone, and an external sample-tube arrangement. We were particularly anxious to make this work as we felt it would be advantageous to measure the top level of medium where the separation was actually being made, but we were doomed to disappointment because it was impossible to keep the screen clean of float.

Since the top gravity of the cone is the most convenient place to sample for control, a launder about 2 in. wide was installed longitudinally beneath the

center of the float drainage screen. This sloping launder took a continuous sample of the float medium draining through the float screen, and diverted it into the sample tube. Following its installation no further difficulty was experienced in securing continuously a representative sample of float drainage medium.

When the problem of sampling was overcome, we immediately started to get recordings on the meter, but because of a variety of causes, it was some time before they were accurate and dependable. These difficulties are described later in this paper.

When it appeared that recording the density was going to be satisfactory even though there were some "bugs," the possibility of not only recording the density but actually controlling it was discussed. It was thought that such control was possible if the specific gravity of the medium could be adjusted by controlling one element such as the water or the densified medium.

Because the densified medium returning to the circuit is always at higher gravity than the circulating medium, the controller could regulate either the amount of densified medium returned to the circuit, or could regulate the amount of water being added to the circulating medium. The latter method was chosen because of its simplicity and furthermore because experience had indicated that this was the best method for manual control.

### Description of Control Method

The principle involved in mechanically measuring specific gravity is an old one, and is employed by most, if not all, of the manufacturers of specific-gravity recorders. It utilizes two pipes of different lengths which dip into the medium or liquid to be measured. These pipes are supplied with a small amount of air through sight bubblers. The pressure required to overcome the head of medium or liquid is of course different for each pipe, and builds up until the air bubbles out of the bubble pipe. The air lines supplying the bubble pipes are connected, one to each side of a differential manometer, and thus, the differential in air pressure in the two bubble pipes is measured. This differential is recorded as specific gravity since obviously any change in spe-

cific gravity of the medium will be reflected immediately as a corresponding change in the air pressure. The two tubes of different lengths are employed to eliminate the need of an absolutely uniform overflow level in the sample tube. Such a uniform overflow level is virtually impossible to obtain, although it has been found necessary to maintain it as closely as is practical for close indication.

As mentioned earlier, the instrument installed in the heavy-media pilot plant had to be able to record a wide range of specific gravities. The change in range is easily accomplished by changing the difference in orifice submergence between the two bubble pipes, the difference being less for the higher gravities. Actually one set of bubble pipes is used with the long one provided with tapped holes at the proper points, which are normally closed with plugs. By unplugging the proper hole, the range of the instrument is easily set for the particular medium gravities that are to be measured.

The actual specific-gravity ranges, together with the corresponding difference in orifice submergence (dimension *H*, fig. 1) of the bubble pipes, are as follows:

"H" in In.	Specific-gravity Range
20	1.25 - 1.75
16 2/3	1.50 - 2.10
12 1/2	2.00 - 2.80
10	2.50 - 3.50

The arrangement of the equipment is shown in fig. 1, in which (1) is the sample launder under the float drainage screen. (2) is the sample tube with its constant level overflow, (3), discharging back into the medium drainage sump (4), and its under-flow spigot (5) discharging into a box and thence back into the medium drainage sump. (6) are the bubble pipes with a difference in submergence *H*

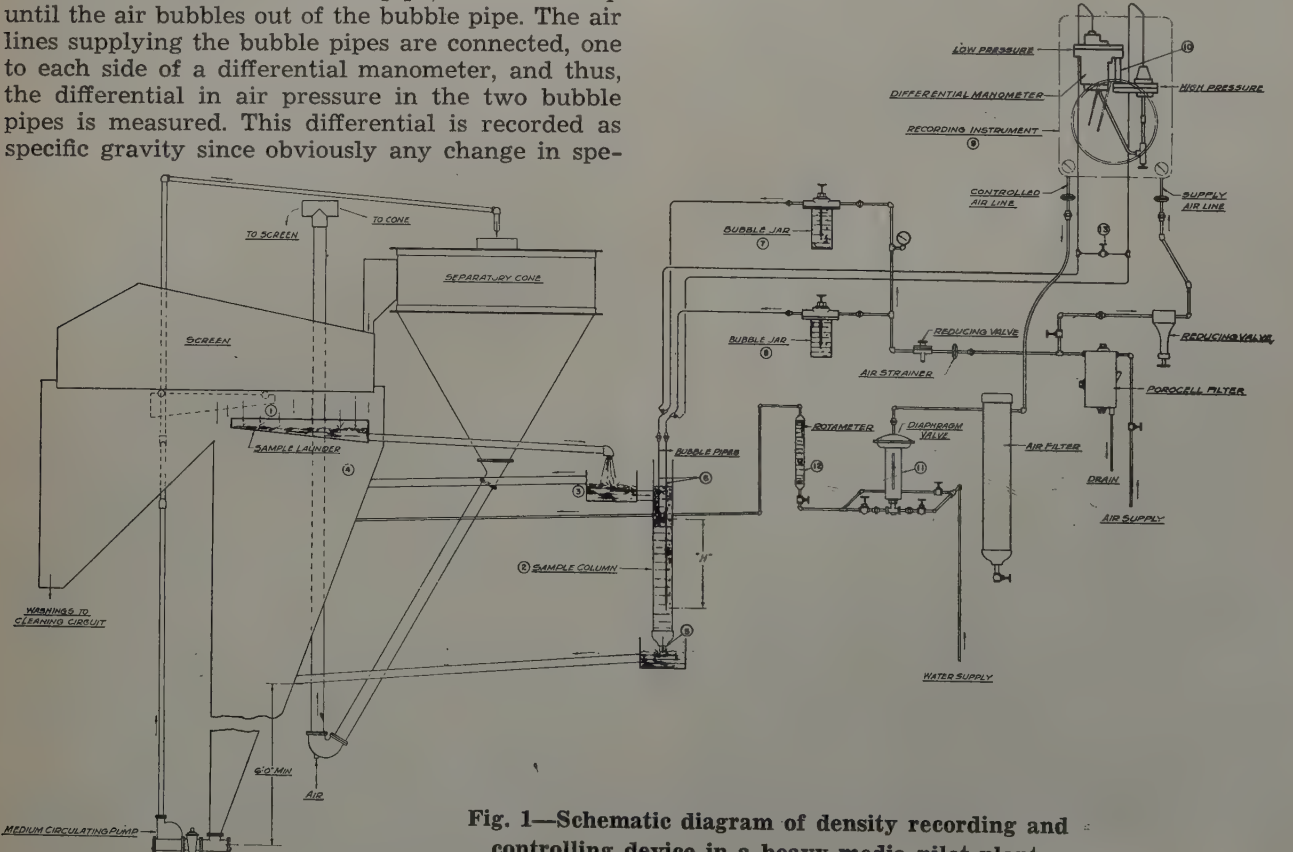


Fig. 1—Schematic diagram of density recording and controlling device in a heavy-media pilot plant.



which as explained above is adjustable for different gravity ranges. Air is supplied to the bubble pipes through the bubbler jars (7) and (8) which have needle valves for adjustment. The actual measurement and record is made by the differential manometer (9).

The differential manometer has an internal construction as shown in fig. 2. From this drawing it is apparent that the instrument is a U tube filled with mercury. The low-pressure side is provided with a float which operates the recording pen through a suitable packing gland. The high-pressure side is simply a reservoir of mercury. Both chambers are provided with check valves to prevent the loss of mercury. At the bottom of the U tube a needle valve is provided to damp the pen movement if necessary, but our experience has been that it is not necessary to damp the pen. Because it is desirable to have a relatively short range of specific gravity over which the meter measures, a counterbalance is built into the meter body. This counterbalance suppresses the zero point, that is, it is built in such a way that the two mercury columns of the U tube are not level until the minimum reading is reached. By this arrangement the full pen movement can be confined to an 0.5 difference in specific gravity between low and high gravity. The counterbalance feature is provided by raising the low-pressure side of the meter above the high-pressure side by means of a spacer (10), fig. 1, and a longer U tube. By raising this chamber it can be seen that considerable pressure must be placed on the high-pressure chamber before mercury is forced into the low-pressure chamber and the float moved.

Once the gravity has been measured, controlling equipment can provide a variable air pressure proportional to the displacement of the pen from a given set point. This controlled air supply can in turn operate a diaphragm valve, or some positioning device. Thus, if the gravity of the medium in the cone circuit can be controlled by varying one element, the instrument can control the gravity of the medium. As it happens the heavy-media separation system is generally so operated, that sufficient medium at a high enough gravity is returned from the densifier to the medium drainage sump to make necessary the continuous addition of a small amount of water in order to maintain the circulating medium at the proper gravity. Under manual control this water supply is varied by the operator and thus it was only natural that when the gravity recording meter started to function, the possibility of controlling this water by the meter was considered.

An instrument was purchased that was equipped with the necessary additional auxiliary units to control the water through a diaphragm valve. The control unit is of the "Free Vane" type and is provided with "Reset" and "Proportional Band" features. The description of the method whereby these two features are obtained would be more involved than the scope of this paper provides, and we shall confine ourselves to a short description of what "Reset" and "Proportional Band" features do.

The proportional-band feature simply narrows the range through which the pen must move in order for the control air pressure to vary from 0 to 17 lb, which is the full pressure required to operate the diaphragm valve. Perhaps this is more clearly stated by saying that a 100 pct proportional band would be the full width of the chart, that is to say, the pen would have to move from the lowest point

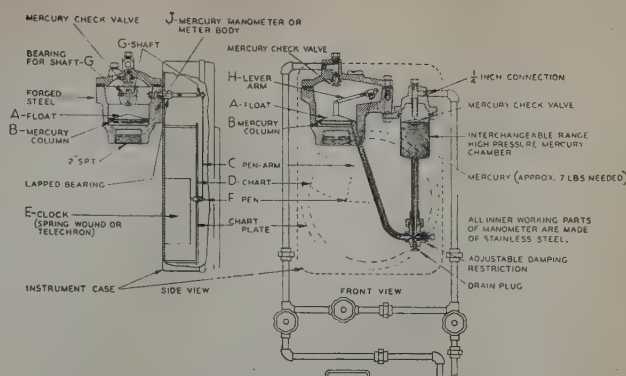


Fig. 2—Construction of differential manometer.

to the highest point of the chart in order to open or close completely the diaphragm valve on the water supply. Through proper linkage this percentage can be varied from 0 to 100. At 0 the controller would be an "on" and "off" affair since any change of the pen away from the set point would fully open or close the valve.

The reset feature functions only when the pen is off the set point when it operates in such a way as to tend to restore the pen to its set point. It moves at intervals which are adjustable through a needle valve, and by proper adjustment tends to eliminate the "offset" or variation between the set point and the actual control point, the latter being the point at which the system is stable for any particular valve setting due to the proportional-band control.

By properly adjusting both controls so that one supplements the other, it has been possible to control the water to the medium drainage sump so accurately that the pen recording the specific gravity virtually draws a circle on the chart, fig. 3.

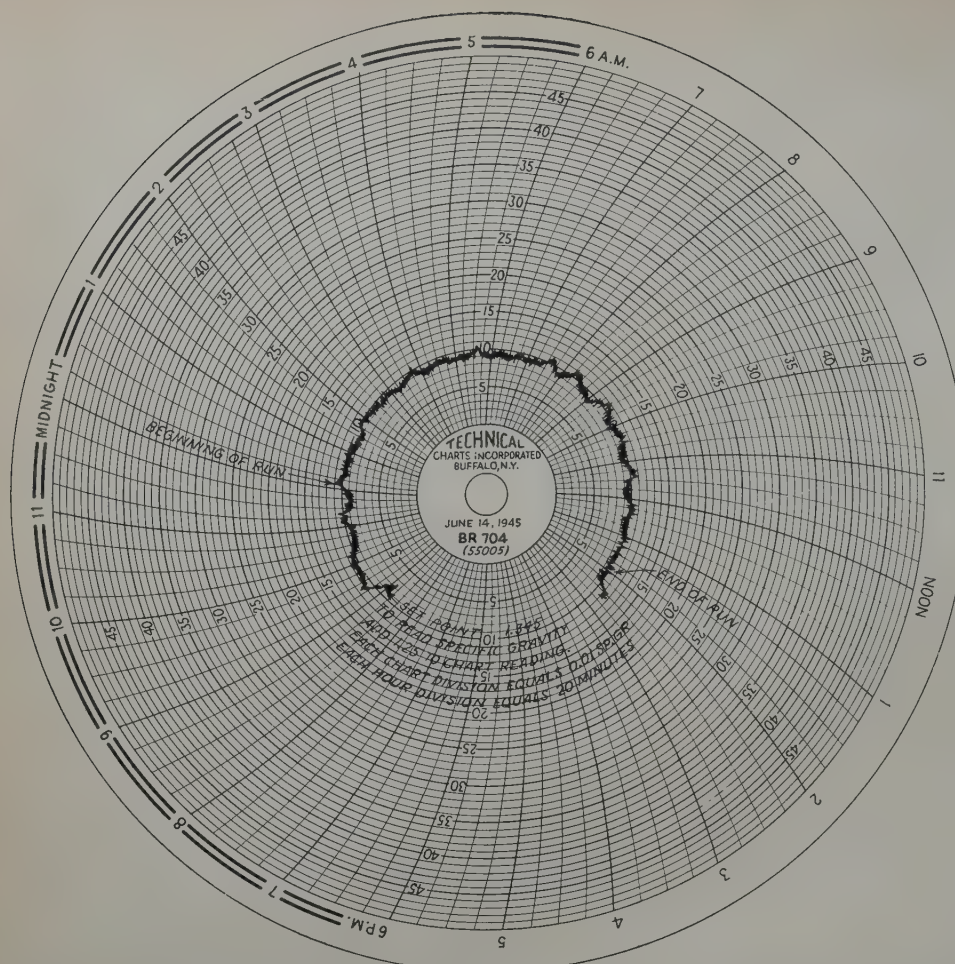
The controller operates in such a way that the controlled air pressure increases as the specific gravity decreases, and an increase in control air pressure tends to shut the diaphragm valve, (11) fig. 1, which supplies water to the medium drainage sump and thus acts to increase the specific gravity of the circulating medium. In the pilot plant, the actual amount of water being supplied is measured with a rotameter (12) but this refinement would not be necessary in an operating plant. Also because we erred in estimating the flow through the diaphragm valve, it is somewhat too small to supply the full water requirement and an auxiliary supply, manually controlled, has been provided. There is considerable debate as to the merits of this arrangement over a single diaphragm valve, and we intend to install a large valve with adequate capacity to settle this point.

Although the units we have installed are standard units, adapted to the particular job to be done, due to the fact that the continuous measurement of specific gravity is relatively new, particularly in the case of suspensions, there are some features which are novel to this installation.

As mentioned earlier, the matter of securing a representative sample for measurement and control purposes has been something of a problem, and while it is believed that the sample launder arrangement is entirely adequate, it does have the disadvantage of requiring headroom where the latter may be at a premium. Some other arrangement may ultimately be devised, which would still allow the measurement of the gravity of the overflow medium, and require less headroom.

We suspect that the bubble pipes may accumulate





**Fig. 3—Chart of specific gravity control in heavy-media pilot plant.**

Coal plant treating coal using magnetite medium. Overflow gravity = 1.345.

medium on themselves, which might cause the opening to close after a period of operation. A preventive measure would seem to be a fixed schedule of wash down to remove such accumulations. Other measures suggest themselves such as feeding water down the bubble tube, but because of the short runs made at the pilot plant, we do not get the accumulations and so cannot work out the answer.

The greatest difficulty we have experienced has been in getting absolutely air tight connections in the bubble-pipe system. Originally  $\frac{1}{8}$ -in. iron pipe was used, but the system leaked so badly that it was removed after only a few days of trial. The next attempt was made with  $\frac{1}{4}$ -in. copper tubing having compression fittings but this also leaked and was replaced with sweat-solder fittings.

As originally installed, each line from the bubble pipes to the manometer was equipped with a valve for test purposes. These valves leaked and were therefore removed. Only the balancing valve, (13) fig. 1, has been retained, and in this position gate valves, globe valves, and needle valves were investigated, and at time of writing a diaphragm sealed type of globe valve was being tried. This diaphragm sealed valve has a packing gland and seems to be air tight.

The difficulty of making pipe connections air-tight to small flows of air at about 10 psi has been incredible to us. Of course with a bubble rate of about 1 per sec, the loss of one or two bubbles a minute is serious, and the pipe joints must be absolutely leak-proof. Joints that will not show a leak to soap suds will still leak too much for accurate recording and the only way to detect them is to isolate a section of the system and use the bubbler itself as the detector. If it stops bubbling eventually

with air pressure on its needle valve, the system is tight enough, though sometimes three quarters of an hour is necessary to bring the system up to pressure and to stop the bubbler.

However, copper tubing with sweated joints is relatively easy to install air tight, and if it is used there should be no difficulty in getting good air tight joints.

When the controller was installed some difficulty was experienced in getting the right setting for the proportional band and the reset, largely because neither ourselves nor the manufacturers had had any experience with such an installation. This inexperience coupled with leaking tubes made things interesting for some time, but now we believe we can diagnose most of the usual faults in the system and correct them without too much difficulty.

Although we have a Bristol instrument in the pilot plant, a number of manufacturers are able to supply similar equipment and have done so. Among these manufacturers are Moore Products Co., the Brown Instrument Co., and the Foxboro Co. No doubt others can supply similar equipment because it is essentially the differential manometer used in flow meters of the orifice type, and the usual control equipment used with such meters.

#### Acknowledgment

Grateful acknowledgment is made to Messrs. Hamilton Bristol, G. M. Heslin, and W. C. Joram, of the Bristol Co., for their determined efforts to make the installation a success, to Mr. Pickens, of the heavy-media pilot plant, who had to live with the unit after it was installed and to the officials of the American Cyanamid Co. for permission to publish this paper.



# Mineral Particles in Electrostatic Separation

by Shiou-Chuan Sun, J. D. Morgan, Jr., and R. F. Wesner

A new apparatus termed a "distribution analyzer" is introduced to ascertain the optimum conditions of electrostatic separation of minerals. It was found that with similar surface condition, testing environment and a constant feed rate of not higher than one layer of particles fed on the grounded roll of the separator, the behavior of each component mineral in a synthetic mixture or ore is roughly similar to that of the same mineral when tested alone.

FOR successful electrostatic separation, the behavior of the component minerals of the tested ore under various conditions should be known. In the course of investigation, it was found that these are difficult to measure accurately by the presently available methods. The method used by Johnson<sup>1</sup> is handicapped by the fact that particles in the extreme front and rear end of the falling stream are difficult to control or observe accurately. Furthermore, this method is not applicable when the tested sample consists of more than one mineral. The deflection method, as reported by Fraas,<sup>2</sup> is inadequate to show the differential amounts of mineral particles deposited at the various horizontal positions under the grounded roll of the separator. This weakness is further noted when the tested sample contains more than one mineral.

SHIOU-CHUAN SUN, Member AIME, is Assistant Professor of Mineral Preparation, Pennsylvania State College, State College, Pa.; J. D. MORGAN, JR., is Assistant for Materials and Stockpile Policies, National Securities Resources Board, Washington, D. C.; and R. F. WESNER, Junior Member AIME, is Engineer, McNally Pittsburg Manufacturing Corp., Pittsburg, Kans.

AIME Columbus Meeting, September 1949.

TP 2829 B. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Dec. 9, 1948; revision received Oct. 24, 1949.

The purpose of this paper is to introduce a new method capable of measuring the behavior of mineral particles under various conditions, and of defining the optimum condition for separation. The main factors which influence the behavior of mineral particles in electrostatic separation are also described.

**Experimental:** A new apparatus termed, "distribution analyzer," was used to replace both the

dividing gauge and the two attached collecting chutes of a Johnson separator.<sup>3</sup> This distribution analyzer consists of sixty or more 1.0-cm wide cells as shown in part in fig. 1; it has been fully described in a previous article.<sup>4</sup>

Except where otherwise stated, the testing procedure used in this paper is as follows: (a) calculate the weight of feed for a 1-min run at a constant feed rate of one layer of closely packed particles falling on the grounded roll; (b) weigh and dry the feed at 100°C for 12 hr; (c) place the

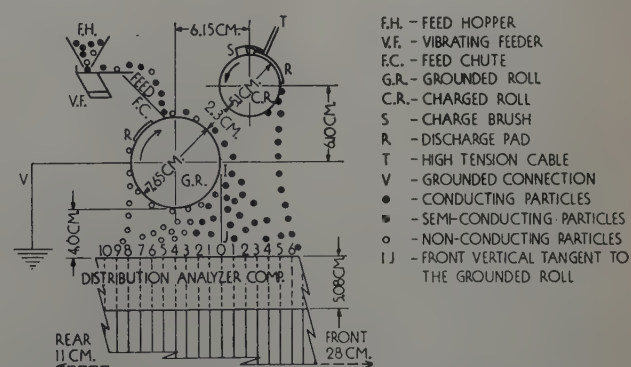


Fig. 1—Schematic diagram showing the operation of the electrostatic separator in combination with a distribution analyzer.

top of the distribution analyzer level 4 cm below the bottom of the grounded roll, so that the cells are longitudinally parallel to the grounded roll and the zero edge of a suitable cell is directly under the front vertical tangent to the grounded roll (see fig. 1); (d) with the charged roll at a negative potential of 15.5 kv, the dry hot sample is introduced into the feed chute at a constant feed rate; and (e)

finally the deposited mineral particles are removed from each cell and weighed and analyzed when the tested sample contains more than one mineral. Weighing alone is sufficient if the tested sample contains one mineral.

The weight of feed was calculated according to the following equation:

$$Q = dwvcgn \quad [1]$$

in which  $Q$  is the weight of feed of sized mineral particles in grams for a minute run;  $d$  is the di-

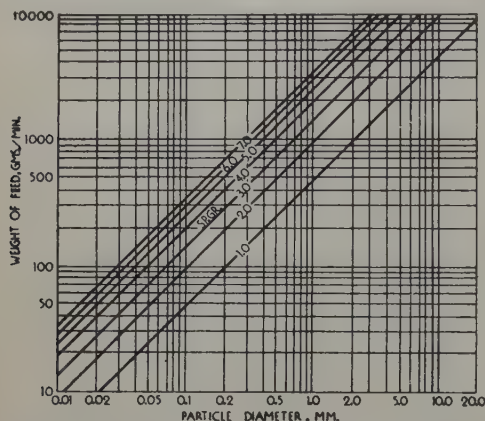


Fig. 2—Calculated weight of feed in grams for one minute runs at a feed rate of one layer of closely packed mineral particles fed on the grounded roll with assumed specific gravities from 1 to 7.

ameter of particle in centimeters;  $w$  is the effective width of grounded roll in centimeters, 11.35 cm for this separator;  $v$  is the linear velocity of grounded roll in centimeters per minute, 766.0 cm per min for this separator;  $g$  is the specific gravity of the tested mineral;  $c$  is the packing voids factor, equal to 0.524; and  $n$  is the predetermined number of layers of closely packed particles fed on the grounded roll. From Eq 1, the weight of feed in grams for minerals with assumed specific gravities from 1 to 7 are calculated and plotted in fig. 2. The curves in fig. 2 can be expressed mathematically as follows:

$$Q = kd^m$$

$$\log Q = \log k + m \log d \quad [2]$$

where  $m$  is the slope and  $k$  is the intercept on the ordinate of 1.0 abscissa. To use the curves of fig. 2 for any roll-type separator, multiply the observed quantity from fig. 2 by a factor  $wv/3694$ . Symbols  $w$  and  $v$  are the effective width and linear velocity, respectively, of the grounded roll of the subject separator.

Mineral samples were prepared from selected lumps of high purity by crushing, grinding, and screening to the desired sizes. All the sized samples except coals were cleaned by magnetic separation and tabling, and finally washed with very dilute hydrochloric acid and distilled water. The cleaned samples were air dried and stored in glass jars.

**Weight Per Cent Distribution:** The weight per cent distribution curves of figs. 3 to 10 were obtained in the following manner: the weight per cent for a given cell is plotted at the midpoint of that cell and these points have been connected with solid and broken lines as shown. Where the points are connected by a solid straight line the following mathematical equation applies:

$$\log W = k + mx \quad [3]$$

in which  $W$  is the weight per cent of particles deposited in any cell of the distribution analyzer,  $x$  is the horizontal distance between the central point of a cell and the front vertical tangent to the grounded roll (the zero distance on the abscissa),  $k$  is the ordinate intercept at zero abscissa, and  $m$  is the slope with minus and plus sign for the front and rear, respectively.

Eq 3 is not applicable to the broken line portions of the curves, which are believed to be caused by the combination of two interrupting forces; one, a mechanical rubbing of the attached particles on the rolls by the discharge pads, and, the other, an excessive attraction between mineral particles and charged roll. The former force may play a larger role than the latter, when nonconducting particles of medium size (48 to 150 mesh) are tested under moderate potentials (8 to 15 kv). The latter force may become the more dominant factor, when conducting particles of fine size are tested under high potentials.

**Optimum Condition of Separation:** There are two methods to ascertain the optimum condition of electrostatic separation of minerals. One is to test the ore sample directly under various conditions and is termed "ore-method." This can be exemplified by the tested data of a synthetic mixture of 82 pct anthracite coal, 14 pct quartz and 4 pct pyrite by weight, as shown in fig. 3. From curves 2, 4, and 6 of fig. 3, it can be seen that quartz and pyrite are partially removed from anthracite coal by

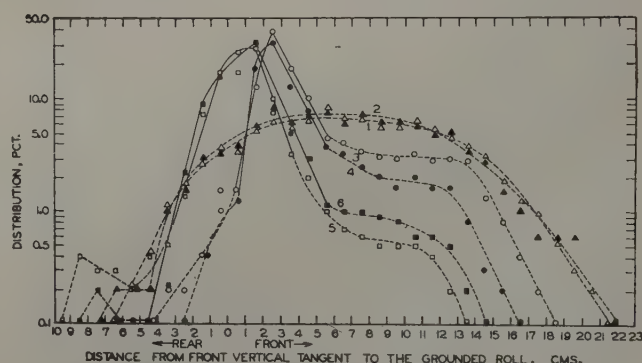


Fig. 3—The electrostatic behavior of 48x65 mesh mineral particles when tested alone and in a synthetic mixture.

Curves: 1, anthracite coal; 2, anthracite coal in mixture; 3, pyrite preconditioned slightly with coal dust; 4, pyrite in mixture; 5, quartz preconditioned slightly with coal dust; and 6, quartz in mixture.

placing the dividing gauge at 4 or 5 cm in front of the front vertical tangent of the grounded roll. The weight of the component minerals deposited in each cell of the analyzer was ascertained by means of sink-float determinations and by microscopic assay of the three fractions.

The other method is to test the individual component minerals instead of the ore itself under various conditions, and is termed "mineral-method." The results of the individual component minerals tested under the same environment are plotted in one figure and the optimum condition of separation can be determined by comparing the different figures. This can be visualized by comparing curves 2, 4, and 6, respectively, with curves 1, 3, and 5 of fig. 3. This method is based on the discovery that, under the same test condition and at a constant feed rate of not higher than one layer of particles fed on the grounded roll, the behavior of each component



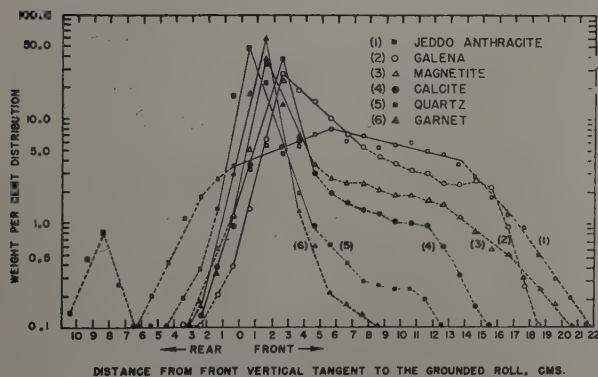
**Table I. Compiled Data Show the Relationship between Electrostatic Interfacial Resistivity and Electrical Resistivity, and also Dielectric Constant**

Minerals	Chemical Composition	Electrostatic Interfacial Resistivity <sup>1</sup>	Electrical Resistivity <sup>12,13,15</sup>	Dielectric Constant, <sup>12,14,15</sup>	
		Relative Volt. (A)	Ohm-Cm (B)	e.s.u. (C)	Frequency Cycles
Graphite	C	1.0-1.28	$8 \times 10^{-4}-0.5$		
Anthracite coal	$C_{70}H_{41}O_6N$ (a)	1.28	$70.0-5 \times 10^{-6}$	5.6-6.3	
Bitu. coal	$C_{135}H_{97}O_9NS$ (a)	1.45-2.23	$(+) 10^8-10^8$		
Mica schist	$(H,K)_2(Mg,Fe)_2Al_2(SiO_3)_3$	1.06-1.73	$16-6 \times 10^4$	16.0-17.0	
Halite	NaCl	1.45	$(+) 3 \times 10^8-10^7$	5.6-6.3	$4 \times 10^8$
Bornite	$Cu_5FeS_4$	1.67	$6 \times 10^{-2}-0.5$		
Chalcocopyrite	$CuFeS_2$	1.67	0.1-10.0		
Fluorite	$CaF_2$	1.84	$(+) 7.9 \times 10^7$	6.8	
Marcasite	$FeS_2$	1.95	$1.4-3 \times 10^2$		
Barite	$BaSO_4$	2.06		7.6-12.2	$4 \times 10^8$
Serpentine	$H_2Mg_3Si_2O_9$	2.17	$2 \times 10^{-4}-10^6$		
Hematite	$Fe_2O_3$	2.23	$3.5 \times 10^{-1.2} \times 10^3$	25.0	
Talc	$H_2Mg_3(SiO_3)_4$	2.34		<102	
Chalcocite	$Cu_2S$	2.34	$3.1-2.3 \times 10^8$		
Barium-strontium-titanate <sup>(18)</sup>		2.34(±)	$(+) 10^{10}-10^{13}$	$6.5 \times 10^8$	
Pyrrhotite	FeS	2.34	$5 \times 10^{-4}-7.0$		
Galena	PbS	2.45	$3 \times 10^{-3}-5.0$		
Stibnite	$Sb_2S_3$	2.45	$(+) 2 \times 10^8-10^{14}$		
Molybdenite	$MoS_2$	2.51	$7.9 \times 10^{-1}-1.6 \times 10^4$		
Ilmenite	$FeTiO_3$	2.51	$2 \times 10^2-5 \times 10^3$		
Siderite	$FeCO_3$	2.56	$7 \times 10^8$	6.9-7.9	$4 \times 10^8$
Tourmaline	$H_2Al_3(B.OH)_2Si_4O_{10}$	2.56		6.5-7.2	
Gypsum	$CaSO_4 \cdot 2H_2O$	2.73		5.0-9.9	$4 \times 10^8$
Magnetite	$FeOFe_2O_3$	2.78	$* 8 \times 10^{-3}-5 \times 10^{-1}$	31-78.5	
Rutile	$TiO_2$	2.62-3.18			
Pyrite	$FeS_2$	2.78	$* 2.3 \times 10^{-3}-9.2$		
Dolomite	$CaMg(CO_3)_2$	2.95		7.3	
Sphalerite	ZnS	3.06	$10^8-10^8$	7.8-8.3	
Quartz	$SiO_2$	3.17-5.3	$10^7-10^{14}$	4.6-5.1	$4 \times 10^8$
Quartz, fused	$SiO_2$	3.17-5.3		3.5-3.6	
Sulphur	S	3.9	$10^{14}-10^{17}$	3.6-4.7	$4 \times 10^8$
Calcite	$CaCO_3$	3.9	$5.5 \times 10^{14}$	7.5-8.5	$4 \times 10^8$
Zircon	$ZrSiO_4$	3.96		11.0-13.0	
Wulfenite	$PbMoO_4$	4.18		23.8	
Apatite	$(CaF)Ca_4(PO_4)_3$	4.18	$> 5 \times 10^7$	8.5	
Smithsonite	$ZnCO_3$	4.45		9.3-9.4	
Topaz	$(AlF)_2SiO_4$	4.45		6.2-7.4	
Corundum	$Al_2O_3$	4.9		11.4-13.2	
Aragonite	$CaCO_3$	5.29	$1.4 \times 10^{14}$	9.1	

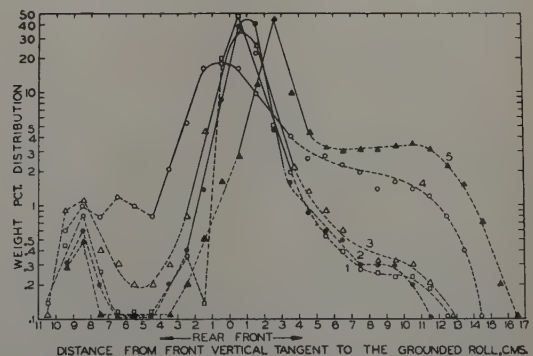
(+) Indicates that the value is apparently too high.

\* Indicates that the value is apparently too low.

(a) Indicates that the formulas are approximate and given by ref. 20.



**Fig. 4—The electrostatic behavior of some representative mineral particles of 48x65 mesh size.**



**Fig. 5—Effect of chemical treatment on the electrostatic behavior of 48x65 mesh quartz particles.**

mineral in a synthetic mixture or ore is roughly similar to that of the same mineral when tested alone.

It should be noted that the surface condition of the component minerals used for the mineral-method requires special attention. For example, it was found in the course of investigation that the conductivity of lower conducting minerals was somewhat increased by contamination with higher conducting minerals, such as coal, graphite, and galena. It should be noted that along this line the pyrite and quartz sample used, respectively, for curves 3 and 5 of fig. 3 were preconditioned slightly with a small amount of coal dust.

In comparison of the ore-method with the mineral-method, the former is more direct and accurate but requires analysis of the mineral particles deposited in each cell of the analyzer. The latter is indirect and less accurate but requires no analytical

work; it is most convenient when the distribution curves of component minerals are available.

**Surface Conductivity:** The experimental data of fig. 4 show that, under similar test conditions, some minerals are repelled farther by the separator than others. This is due to the fact that minerals with a high-surface conductivity or low-surface resistivity are more easily polarized<sup>5,6</sup> to the same polarity of the grounded roll than those of low-surface conductivity. The effect of the sign and magnitude of electrical charge on the behavior of mineral particles in electrostatic separation is given by Columb's law.

$$F = \frac{Q_1 Q_2}{kr} \quad [3]$$

where  $F$  is the magnitude of the attractive or repulsive force in dyne between two charged bodies;  $Q_1$  and  $Q_2$  are, respectively, the charge of rolls and

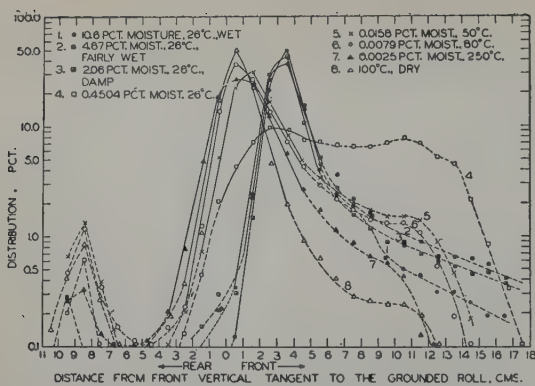


Fig. 6—Effect of moisture content on the behavior of 48x65 mesh quartz particles.

mineral particles in electrostatic units;  $k$  is the dielectric constant of the surrounding medium; and  $r$  is the distance in centimeters between the two charged bodies.

The behavior of mineral particles in an electrostatic field is a surface phenomenon and is governed chiefly by the surface atoms and ions instead of the internal composition of mineral particles. For example, the electrostatic behavior of quartz can be

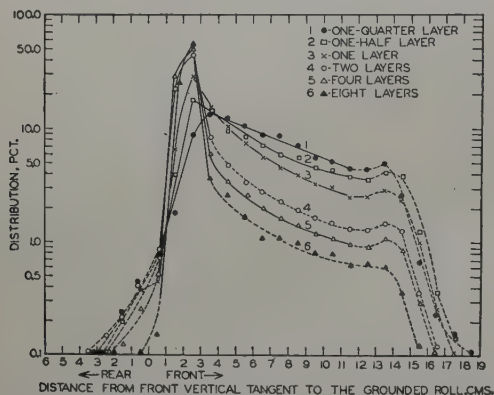


Fig. 7—Effect of feed rate on the behavior of 48x65 mesh galena particles.

altered by changing its surface condition with chemicals and moisture, as shown in figs. 5 and 6, respectively. The data of table I indicate that it is difficult to establish a relationship between electrostatic behavior (col A) and dielectric constant (col C) of minerals. This may be due to the fact that the dielectric constant<sup>7-10</sup> of minerals varies within narrow limits and represents more or less the electric conductance of the bulk body instead of the surface of the mineral particles. On the other hand, there is a general relationship between the established data of electrical resistivity (col B)<sup>9, 10, 11</sup> and the electrostatic behavior (col A) of minerals.

**Chemical Treatment:** The data of fig. 5 indicate generally that the electrostatic behavior of quartz particles is altered by chemical treatment, changing in the decreased order, lauryl amine hydrochloride, copper sulphate, hydrofluoric acid vapor, and barium chloride. This behavior depends chiefly on their surface conditions, which are in turn governed by the surface atoms and ions, as previously described. The surface condition and consequently the surface conductivity of quartz is altered by the absorption of aminium ions<sup>12, 13</sup> from lauryl amine hydrochloride solution, of copper ions<sup>13</sup> from copper sulphate solution, and of barium ions<sup>14</sup> from barium

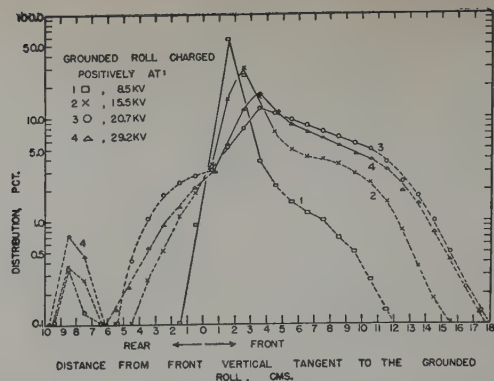


Fig. 8—Effect of the magnitude of applied voltage on the behavior of 48x65 mesh calcite particles.

chloride solution. The effect of hydrofluoric acid vapor on the behavior of quartz may be caused by the formation and evaporation of silicon tetrafluoride vapor on quartz surface.

The curves of fig. 5 are: 1, no chemical treatment; 2, conditioned 5 min with 10 mg per liter of lauryl amine hydrochloride at 1.5 pulp dilution; 3, conditioned 10 min with 50 mg per liter of  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$  at 3.0 pulp dilution; 4, treated 15 min with hydrofluoric acid vapor; and 5, boiled 15 min with 50 mg per liter of barium chloride at 1.5 pulp dilution.

**Moisture Content:** From curves 4 to 8 of fig. 6 it can be seen that to a certain limit, the magnitude of repelling of quartz particles by the separator is increased with the increase of moisture content. This phenomena can be explained by the established data<sup>15, 16</sup> that the presence of water film on quartz surface gives higher surface conductivity. Curves 1, 2, and 3 of the same figure indicate that when moisture content is about 2 pct or higher, the quartz particles form large aggregates. These aggregates are subjected to greater centrifugal force by the revolving roll but are too heavy to be widely spread as compared with small particles.

From the above data and discussion it can be concluded that, with other factors being constant, quartz particles will be repelled farthest when their moisture content is sufficient to form thin water-films on quartz particles but not excessive to form aggregates. This condition is achieved when 48x65 mesh quartz particles contain 0.45 pct moisture, as shown in curve 4 of fig. 6.

**Feed Rate:** The tested data of fig. 7 indicate that, to a certain limit, galena particles are repelled farther with the decrease of feed rate. On account of the limited surface area of the grounded roll, at high feed rate some of the particles may not have the opportunity of making direct contact with the grounded roll. Furthermore, at high feed rates the charged particles must pass through a thick layer of uncharged particles resulting in interference and nullification. Thus, with the reduction of feed rate, these detrimental effects are partially eliminated.

**Potential and Polarity:** In regard to the effect of the magnitude of applied voltage on the behavior of mineral particles, fig. 8 shows that to a certain limit calcite particles are repelled farther with the increase of applied voltage. When applied voltage is increased the field intensity<sup>17</sup> is also increased and consequently the electric induction of mineral particles is facilitated. A mathematical deduction of the force on particles in terms of field intensity has been given by Hatfield<sup>18</sup> and advanced by Fraas and Ralston.<sup>19</sup>



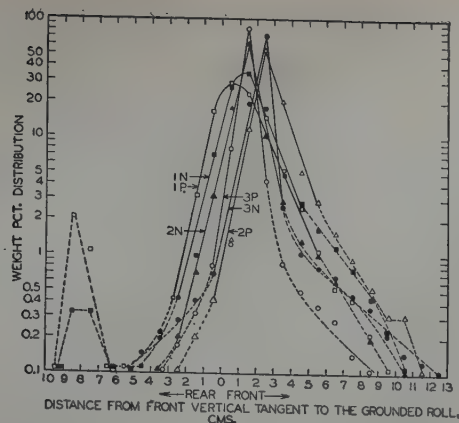


Fig. 9—Effect of the polarity of applied voltage on the behavior of garnet, fluorite, and quartz particles.

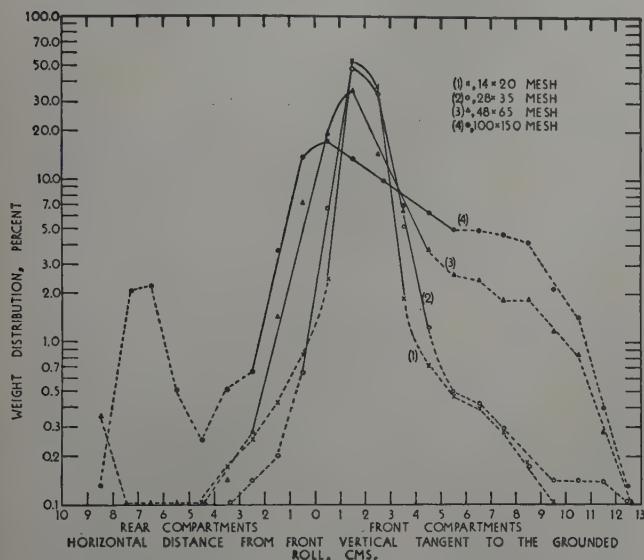


Fig. 10—Effect of particle size on the behavior of halite particles.

With respect to the effect of the polarity of applied voltage on the behavior of mineral particles, fig. 9 shows that garnet and fluorite, which were previously reported as nonreversible, are found to be respectively reversible negative and reversible positive. The contradictory results may be partially caused by the fact that mineral samples used in this paper differ from that used in the previous investigation. The other possibility may be that the distribution analyzer is more accurate to detect reversibility than the method used before.

In fig. 9, the three groups of curves 1P, 1N; 2P, 2N; and 3P, 3N; are, respectively positive and negative values for 100x150 mesh garnet, 48x65 mesh fluorite, and 28x35 mesh quartz particles tested with grounded roll charged at 15.5 kv.

**Particle Size:** From the tested data of fig. 10 it can be seen that, to a certain limit, the dispersion or spread of the weight per cent distribution curve of halite is increased with the decrease of particle size. This is due to the fact that fine particles with larger specific surface area can attain higher specific surface density of electrical charge than coarse particles. Furthermore, fine particles are less subjected to gravitational force than coarse particles, because of their difference in weight.

**Summary:** 1. A new apparatus termed "distribution analyzer" is used to determine the optimum condition of electrostatic separation and the

electrostatic behavior of minerals under various conditions.

2. It is found that with similar surface condition, test environment and a constant feed rate of not higher than one layer of particles fed on the grounded roll, the behavior of each component mineral in a synthetic mixture or ore is roughly similar to that of the same mineral when tested alone.

3. The main factors which influence the behavior of mineral particles in electrostatic separation are: surface conductivity, of mineral particles, chemical treatment, moisture content, feed rate, potential and polarity of applied voltage, and particle size.

4. With other factors being kept constant, quartz particles are repelled farthest when their moisture content is sufficient to form thin water-film on quartz surfaces and not excessive to form aggregates.

**Acknowledgment:** The writers wish to express their thanks to Professor David R. Mitchell, Head of the Department of Mineral Engineering, for his administrative guidance and heartening encouragement.

### References

- <sup>1</sup> H. B. Johnson: Selective Electrostatic Separation. *Trans. AIME* (1939) **134**, 409.
- <sup>2</sup> F. Fraas: The Conductance Electrostatic Separator. *Trans. AIME* (1943) **153**, 576.
- <sup>3</sup> The Johnson Electrostatic Separator, manufactured by the Ritter Products Corp., Rochester, N. Y.
- <sup>4</sup> Shiou-Chuan Sun: Analyzer Aids Electrostatic Research. *Eng. and Min. Jnl.* (1949) **150** (5) 90.
- <sup>5</sup> C. P. Smyth: Dipole Moments. *Ind. and Eng. Chem.* (1931) **23** (11) 1224.
- <sup>6</sup> C. E. Bennett: New Outline of First Year College Physics. 73-81 (1947) New York. Barnes and Noble, Inc.
- <sup>7</sup> C. P. Smyth: Dielectric Constant and Molecular Structure. 45 (1931) New York. The Chemical Catalog Co., Inc.
- <sup>8</sup> C. A. Heiland: Geophysical Exploration. 656-667 (1940) New York. Prentice-Hall, Inc.
- <sup>9</sup> L. Hartshorn, C. C. Harris, J. A. Hey, T. I. Jones, and A. G. Milligan: Dielectric Constant. International Critical Tables, **6**, 99 (1929) New York. McGraw-Hill Book Co.
- <sup>10</sup> F. Birch, J. F. Schairer, and H. C. Spicer: Handbook of Physical Constants. Geol. Soc. of Amer., *Spec. Paper No. 36*, 304-318 (Jan. 31, 1942).
- <sup>11</sup> G. M. J. Mackay: Electrical Resistivity. International Critical Tables, **6**, 153 (1929) New York. McGraw-Hill Book Co.
- <sup>12</sup> Shiou-Chuan Sun: The Mechanism of Slime Coating. *Trans. AIME* (1943) **153**, 479.
- <sup>13</sup> A. M. Gaudin and S. C. Sun: Correlation Between Mineral Behavior in Cataphoresis and in Flotation. *Trans. AIME* (1946) **169**, 347; *Min. Tech.*
- <sup>14</sup> A. M. Gaudin and A. Rizo-Patron: The Mechanism of Activation in Flotation. *Trans. AIME* (1943) **153**, 462.
- <sup>15</sup> H. L. Curtis: Insulating Properties of Solid Dielectrics. *Bull. Bur. of Standards* (1915) **11**, 359.
- <sup>16</sup> R. F. Field: The Formation of Ionized Water Films on Dielectrics Under Conditions of High Humidity. *Jnl. of Applied Physics* (1946) **17**, (5) 318.
- <sup>17</sup> C. I. Dawes: A Course in Electrical Engineering. 3rd Ed. **1**, 315-351 (1937) New York. McGraw-Hill Book Co.
- <sup>18</sup> H. S. Hatfield: Dielectric Separation: A New Method for the Treatment of Ores. *Trans. Inst. of Min. and Met. London* (1923-1924) **33**, 335.
- <sup>19</sup> F. Fraas and O. C. Ralston: Dielectric Constant in Air-Ambient Electrostatic Separation. U. S. Bur. of Mines. *R.I.* 4278 (1948).
- <sup>20</sup> W. Fuchs and A. G. Sandhoff: Theory of Coal Pyrolysis. *Ind. and Eng. Chem.* (1942) **34** (5) 567.

# Organizing and Financing Cooperative Research

by Elmer R. Kaiser

Cooperative research is an established and important activity of modern business by which whole industries can advance on a broad front. The economy of jointly sponsored research as well as the limited research talent available direct attention to the advisability of projects in which a number of companies participate.

After outlining the growth of industrial research, the paper discusses one successful method for organizing and financing group research programs. A systematic procedure is advocated by which engineers and scientists can bring to management practical plans for cooperative research and development programs.

**Industry Adopts Research:** Seventy-five years ago Thomas A. Edison began in a modest way and with limited funds to gather about him men of various talents to form the first industrial research laboratory. His laboratory at Menlo Park, N. J., became famous for many discoveries and inventions, but possibly the most important of all may yet be the industrial research laboratory itself.

ELMER R. KAISER, Member AIME, is Assistant Director of Research, Bituminous Coal Research, Inc., Columbus, Ohio.

Joint AIME-ASME Meeting, White Sulphur Springs, Nov. 1948.

TP 2833 FK. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Dec. 7, 1948.

Today, private industry operates 2500 research laboratories in which 135,000 persons<sup>1</sup> are directly and indirectly creating new processes and products and improving old ones, as well as developing new information to help keep the fountain of knowledge from running dry. In this change-making function, the scientific personnel comprises 41 pct; the technical personnel 26 pct; and the balance, 33 pct, non-technical.

The term "industrial research" has become so broad as to include almost everything beginning with the search for and discovery of the laws governing the properties of matter and energy to the development and testing of processes, methods, and equipment. These classes of investigation commonly merge so that no sharp boundary can be traced between them. The National Research Council therefore defines industrial research as "the endeavor to learn how to apply scientific facts to the service of mankind."<sup>1</sup>

Currently, expenditures for research by industry are variously estimated at between \$450,000,000 to

\$700,000,000 a year,<sup>2</sup> or about twice the expenditure for 1940. The President's Scientific Research Board<sup>3</sup> estimated that the national research and development for 1947 (excluding atomic energy) was as follows:

Industry .....	\$ 450,000,000
Government .....	625,000,000
War and Navy Depts.....	\$500,000,000
Other departments .....	125,000,000
University .....	45,000,000
Other .....	40,000,000
Total.....	\$1,160,000,000

A few companies spend as much as 5 pct of their gross receipts for research, but 1½ to 2 pct is considered a good proportion by many.<sup>4</sup>

**Shortages:** Commendable as is the record of growth and accomplishment of science in industry, it is only a beginning. America and the rest of the world have merely sampled the fruits of industrial research; most industries have yet to make research the key to their future.

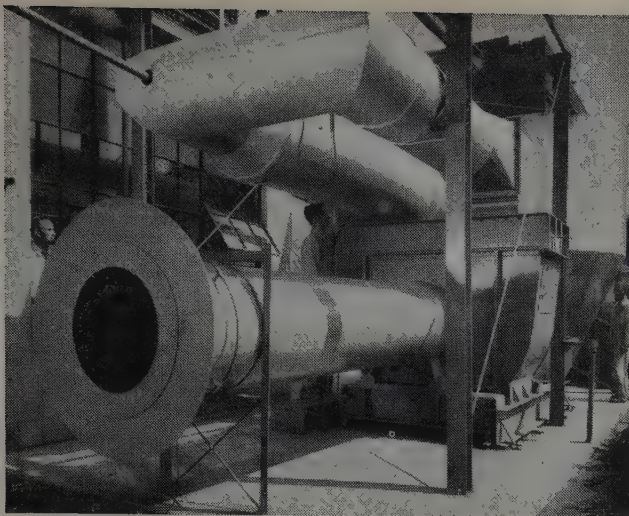
A top research consultant, T. A. Boyd, of General Motors, states "... perhaps 90 per cent of all companies still have no research laboratories."<sup>5</sup> For example, of more than 17,000 manufacturing concerns whose gross sales exceed \$500,000 a year, less than 2500 have research laboratories. A still smaller proportion of the 184,230 manufacturing establishments in 161 industries<sup>6</sup> have or could afford their own research facilities. Yet, the small and medium-sized companies with less than 250 employees each, produce one half of the dollar volume of manufactured products.<sup>7</sup>

How are these companies to participate in and share the benefits of research?

Boyd gives two reasons for the inadequate number of laboratories in industry:

Many people still do not appreciate the importance of research, nor understand its tremendous potential.





**Fig. 1—Bituminous Coal Research, Incorporated, is one of the largest cooperative sponsors of research at Battelle Memorial Institute.**

(This apparatus is an experimental combustor used in a project aimed at the development of a coal-driven gas-turbine locomotive. The work is conducted under the auspices of BCR's Locomotive Development Committee and financed by railroad and coal interests.)

ties. Another important reason lies in the belief that research is too costly and too complicated for any but large organizations to undertake.

Still another factor, equally vital in any attempt to increase the total research facilities rapidly, is the shortage of technically trained and experienced men and women to organize and conduct research. The demand for scientists and engineers still exceeds the supply, and will continue to do so for some time, despite the increasing numbers being recruited from the graduating classes of the colleges and universities.

Thus the combined limitations of management, money, and man power are forcing industry to look about for additional facilities, talent, and suggestions for growing research requirements.

**Cooperative Research an Answer:** Individual companies have a choice today whether to (a) found their own laboratories, (b) sponsor private projects at established research institutes or at independent research and development laboratories, (c) sponsor or conduct research cooperatively with competitors

or others, and (d) combinations of (a), (b), and (c). Of course, there is always the option of not participating in research at all, which is fast becoming recognized as another form of gambling in business with the odds against success.

The first two options have been discussed in most of the many books and articles on industrial research. Less known are the potentialities of cooperatively sponsored programs and how they are organized and financed. Once such programs are established, the technical work itself proceeds much as in company laboratories.

Under joint sponsorship, research on such subjects as "The Behavior of Steels in High-Temperature Steam Piping Service," can be undertaken by a single team of researchers for a group of companies at only a fraction of the cost and man power per company that would be required if each company made the study privately. Results obtained by a group are more widely and more readily accepted, and more experience can be applied through the committee of sponsors which administers the program. This is an example of the type of research that individual companies would be glad to help support, but cannot justify doing alone for the benefit of an industry.

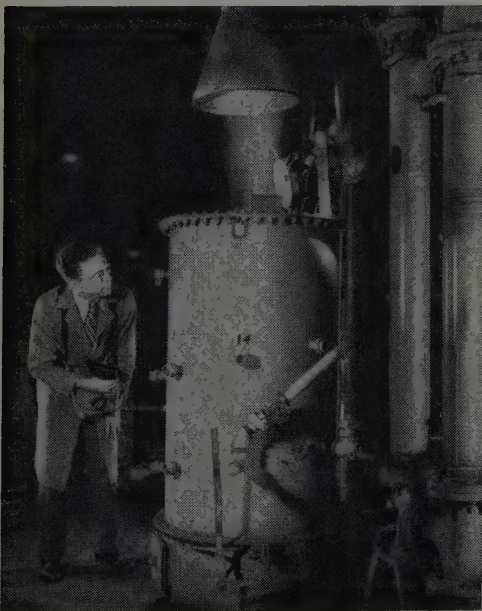
It is the purpose of this paper to discuss some of the factors and experience in cooperative research, including the organization of such undertakings, possible combinations of sponsorships, and dividing the cost among a sizable number of sponsoring companies. Many opportunities exist for the organization of sponsoring groups by those who will give time and effort to the work. It is a vital activity for our industrial progress and national strength in which technically trained men must take a leading role.

**Research by Associations:** Companies with common business problems are often members of trade associations, of which there are 11,000 in the United States. Of this number, 2600 are national and interstate organizations and 1900 are associations of business competitors in one industry.<sup>8</sup> "Industrial research as to production or products" is one of the principal activities of 629 associations, while 752 associations include "commercial research as to markets or marketing."

Trade associations form a logical medium for organizing and sponsoring research programs for the industries they represent. Businessmen discuss openly in meetings of such cooperative organizations many industrial matters which would be considered trade secrets in some countries. Here research is a business matter and an important factor in the ability of an industry to meet competition.

Outstanding examples of 30 associations in industry which have their own laboratories are American Gas Association, Anthracite Institute, National Lumber Manufacturers Association, Portland Cement Association, and Underwriters Laboratories.<sup>8</sup> Such laboratories are supported by dues from members and from additional contributions of funds and payments for special services.

Over 35 associations, including a few which operate their own laboratories, sponsor projects at universities and institutes.<sup>8</sup> Among the industrial laboratories established to conduct research for industry on a not-for-profit basis are Armour Research Foundation, Battelle Memorial Institute, Mellon Institute, Midwest Research Institute, and Southern Research Institute. Several hundred independent research and



**Fig. 2—As part of its program of research aimed at improvements in manufacture of gas, the American Gas Association is sponsoring at Battelle a study of fundamentals of water-gas reaction.**

(This small-scale generator for production of water gas is used in research to study reactions that occur in generating bed.)



development laboratories are also available for carrying out cooperative and private programs for large and small businesses.

Among government agencies, the National Bureau of Standards, Agricultural Department, and Bureau of Mines are three of many units<sup>8</sup> in which industries have working arrangements for research, sponsored by industry in government laboratories. For example, a study of acid mine drainage is being sponsored at the Bureau of Mines in behalf of the coal producers by Bituminous Coal Research, Inc. The basic studies of coal carbonization of American coals is another example of joint research with support over a period of years from the American Gas Association and now continued by the Bureau.

Many professional technical societies are also active in research. The American Society of Mechanical Engineers, for example, serves as a forum and coordinating agency. Seventeen special research committees are active, and support to date of \$586,000, largely from industry, reflects the confidence industry has in such work.

The American Institute of Mining and Metallurgical Engineers sponsors research through The Engineering Foundation, but industry contributes

(3) cutting production costs, (4) sales volume and net profit on new processes and products, and (5) serve production through development of new and improved processes.

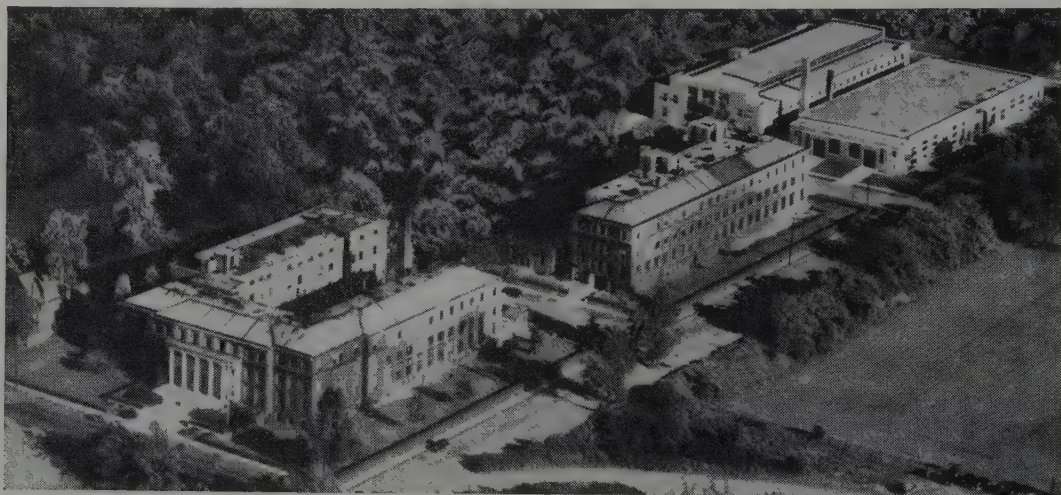
The popularity of these expectations is confirmed by a survey on the "Research Requirements of American Industry" by the Evans Research and Development Corporation.<sup>2</sup> The survey revealed that 47.6 pct of industry invests funds in research to improve present products and processes, 42.3 pct to develop new products and processes in their own fields, and 14.7 pct to develop products and processes in other fields.

The types of projects appropriate for groups to sponsor are not necessarily the same as for individual companies. In some cases they depend upon the degree of initiative in the competitive race that members wish to retain. The following are generally suitable:<sup>8</sup> (a) studies of raw materials, (b) extension of the commercial uses of products, (c) standardization of analytical procedures and improvement in methods and instruments of measurement, (d) improvement in safety and sanitation, and (e) expansion of fundamental science in industry's field.

When members agree, the list can be extended

**Fig. 3—An aerial view of the main buildings of Battelle Memorial Institute, Columbus, Ohio.**

(Approximately 30 pct of the Institute's industrial research is for cooperative sponsorships.)



directly to projects requiring additional funds. The Committee on Research is made up largely of representatives from industry.

The medium of an existing organization is not necessary for organizing a jointly sponsored project. Two or more individual companies with a common technical problem but no facilities to investigate it, today can obtain advice from competent sources and assistance in planning a research program which they may find economical to sponsor. Producers and consumers, government and industry, universities and technical societies, trade associations and committees, manufacturers and suppliers, farmers and processors, can organize to undertake joint investigations and development work. The number of cases is increasing as scientists, engineers, and businessmen become aware of the opportunities.

**Typical Topics for Cooperative Research:** When the National Research Council investigated what management in nearly 100 companies expects of research,<sup>9</sup> the five most common replies listed in their order of rating were as follows: (1) new products, (2) maintenance of competitive technical position,

with the following: (f) development of new products, and (g) development of new processes and improvement of old ones.

Manufacturers usually agree to research which raises the general quality level of an industry's products.

It is not advisable to undertake or conduct projects that will: (a) interfere with the competitive activities of members (b) have limited applicability, or (c) be too far ahead of industry's thinking.

The ideal situation is (a) the knowledge it is planned to provide is needed, (b) that need is widely felt, and (c) the results obtained will be accepted immediately and used.

**Organizing a Program: Phase 1. Concept of a Project or Program:** The background necessary to evaluate opportunities in research may be obtained through years of experience in an industry or through a formal survey. Usually an acute or chronic technical problem, with its adverse economic effects, is apparent in an industry, the solution of which at reasonable cost would spell profits. An investigation usually will disclose several such problems or needs,



some of which are important enough to warrant a program. For example, the coal industry has urgent need for a machine that will mine coal continuously at low cost, an industrial coal burner that will automatically stoke itself and place the ashes into containers, a less expensive method of producing synthesis gas directly from coal, and a coal-burning locomotive of higher efficiency and greater service availability.

In metallurgy there are always problems in producing sound castings, in reducing corrosion, and in producing tougher metals and alloys for service at elevated temperatures. Every progressive industry has technical problems and opportunities; only dead industries do not.

Engineers and businessmen know the problems in their own industry, but may be too close to them by daily contact to see them in true perspective. An investigator from outside an industry, qualified by experience or association with research programs and business, can often see opportunities not apparent to those closely associated with daily operating matters. A fresh point of view and a knowledge of what other industries have done in similar circumstances are assets in conceiving new programs, because processes in one industry sometimes can be applied in another. Consulting research specialists are also available to suggest lines of investigation which are likely to yield valuable results. Many such specialists have laboratories and technical staffs ready to conduct all or parts of the investigations being considered.

In this initial phase, it is not necessary to make a thorough study of each opportunity or problem or how the research could be conducted, or by whom. It is better to become more thoroughly familiar with the industry under consideration, its economic justification, the companies in it, their products, volume of sales, competitive position, and the evidence of cooperation between the companies and the key individuals. Some companies are notably sympathetic toward joint efforts to improve the industry, while others prefer independent action.

Out of this preliminary study should come definite and encouraging indications of worth-while opportunities and necessary research. Clear ideas of the possibilities of a plan of research must be formulated and expressed in a written statement of the factors just discussed. The statement may constitute the report of an investigator hired for the purpose, or it may be retained by the originator of the plan for his own information and guidance. The statement may raise questions and suggest more thorough surveys. It would be premature at this stage to present a cut-and-dried outline of specific projects to be undertaken.

**Phase 2. Preliminary Discussions:** The product of Phase 1 is hardly more than a basis for discussion. A period of incubation is necessary to develop a plan which can be presented to industry and warrant an investment. Usually, this can best be done by the originator in personal interviews with engineers and businessmen.

Entree must usually be arranged beforehand, and it is of considerable value to be representing an organization known for good works, as a member of a committee of a technical society or trade association, employee of a research institute, independent laboratory, university, or member company in an industry.

With a knowledge of an industry's needs and problems, the originator now sets out to interview the top executives and engineers who are most likely to lend a sympathetic ear and give sincere comments and suggestions. When businessmen learn that an engineer has come to seek their advice and suggestions on some ideas he has gathered for improving (or increasing the market for) the industry's products, for reducing costs, or for developing new processes, he can be sure of a welcome.

The trade and professional associations in the industry are vitally interested in any consideration given to the industry's welfare. The directors and officers are usually glad to assist in furnishing information and in suggesting avenues of approach to key individuals. Editors of the leading journals in the industry are also well informed and interested in the industry's future.

Valuable experience and information not known outside the company or industry, all bearing directly or indirectly on the particular subject under discussion, are often made available. Reminiscence sheds light on old ideas and helps to evaluate new ones. Informal discussion, in which it is wiser to listen than talk, is the best meeting of the minds. In companies which have research facilities, the director of research should be interviewed, as he will have ideas and advice to contribute.

At this stage, it is best not to let any portion of the plan "jell" or "freeze," and one must not be too definite as it leads to fixed positions and may develop early opposition. New ideas, and that includes good ones, are usually criticized and said to be unsound and impractical until they can be organized and expressed properly. The important criterion is not whether someone believes a program can be organized or will not be supported, but why? Constructive comments and advice are obtained from only a few, but they are all important. However, it is best to listen to everyone.

In this second phase, it is usually better to have individual interviews than joint meetings as they provide more freedom of thought and expression and permit confidential information to be given to the interviewer. During the preliminary discussions, it is well to determine who is who in the industry, which executives are held in highest regard and are well known for high integrity. Interest should be developed in attending an organization meeting should the preliminary discussions disclose a sincere desire on the part of a number to proceed with planning. The auspices under which the meeting could be called must be well established so as to prevent any feeling of factionalism. It is advisable to ask who would be interested in attending an organization meeting, and which men in each company would represent (a) management and (b) engineering or research.

Many will readily see the advantage of cooperative enterprise to develop the ideas, as few companies have the financial resources available to develop a project that will benefit an entire industry as much as the developer company.

Enough executives should be interviewed to obtain comments from at least the representative units of the industry or until the pattern of comments repeats itself often enough to show definite interest in proceeding to the next step.

Preliminary discussions may be held in small technical committee meetings, which can be a great saving of time, if those present are all interested in



formulating a plan and have the necessary background and freedom of expression. This is suggested only where the originator is well known and has the full confidence of those invited. This procedure is usually followed in technical societies, except that only one project is usually considered at a time rather than a broad program.

*Phase 3. Organization Meetings and Committees:* When and if enough interest has been shown in the several major ideas that have been discussed, a meeting of a selected group of men can be called under the proper auspices to develop the ideas further and to take some coordinated action. Invitations should be extended to all companies, groups, and individuals who are believed to be of a cooperative frame of mind and likely to be helpful in organizing a program. A central location should be selected for the meeting and notice given at least two weeks in advance. A list of names of those invited, an agenda, and any memoranda that might well be considered beforehand should be sent with the invitations so as to keep everyone as fully informed as possible.

In a trade association the president is a logical one to call the meeting and to serve as chairman, but where no such association exists, one of the leading engineers or executives should be selected. A secretary should be designated, and minutes recorded for preparation and distribution later. The minutes are to be sent to those who attended and to others who were also invited. As the meetings are to be working meetings, it is best to limit the attendance to 25 or 30, and usually to a smaller number. However, there is danger in having too small a group and excluding some whose interest will be needed later.

The first group meetings to consider a joint research program may be either on the business or technical level. If a business meeting is called first, action may be taken on the advisability of a program, the probable cost, who should be invited to joint sponsorship, and who is to have charge of arranging the plan and program. The technical details are left to a technical advisory committee and an executive committee.

If a technical meeting is held first, objectives of the program can be outlined, a number of subjects listed, together with an estimate of cost and a suggestion for personnel organization.

In one type of program the first meeting is usually best made a technical one with the sole objective of analyzing the problems that have been suggested, and agreeing if possible on a limited selection of problems. To give point to the meeting, a talk by the originator or investigator should be presented in which a digest of the problems and a general summary of the comments is given without identity as to source.

Each of those present should be free to put forth his best comments and suggestions without committing his company. Little or no regard should be given to who would sponsor the work, except that it would be assumed it would be a joint project sponsored by the members of the industry and all will be invited to join. Little or no attention need be given in the beginning to where the work may be done and what the cost may be. It is likely if attention is given to these details, conclusions may be reached which are entirely premature.

The presence of one or two top-management executives or prominent individuals usually has a

good effect in giving moral support to the meeting and encouragement to the technical men.

When more problems or projects are suggested than could possibly be undertaken, a selection must be made in a democratic manner. The topics may be listed in the order suggested during the meeting and lettered consecutively A, B, C, etc. Each of those in attendance then lists the topics in his own order of importance or preference, after which the ballots are collected and the results tabulated. In this manner twenty projects can be reduced to five or ten.

Too much caution cannot be exercised in maintaining democratic procedures and every evidence of fair consideration throughout the organization of a cooperative program. In highly competitive industries, particularly, bad blood is easy to generate as many individuals whose cooperation is highly desirable are sensitive to their prerogatives.

Individuals and companies may have dissimilar ideas and interests about what should be done, and how, but agreement can be reached directly on some matters, and a suitable compromise usually can be reached on others. Fortunately, there generally are so many noncontroversial problems to solve in an initial program that others can be laid aside.

The culmination of Phase 3 is a written statement of the projects proposed, together with a list of those who participated in the meetings or those who endorse the program. A short statement of each project, a justification for research on it, and an estimate of the annual expenditure recommended for it will be required. The estimate of annual cost can be made by research directors or consultants with reasonable accuracy. Predictions of the time that will be required to complete a research project are of limited value and usually unnecessary. If the work is progressing satisfactorily, sponsors often find it desirable to continue on at least a year-to-year basis.

The statement should be written for management as a clear recommendation of the group of engineers or executives who have participated in the discussions and meetings up to this time.

*Phase 4. Financing the Program:* Those who will have the most direct interest in a research program are the logical sponsors and administrators of it. The public is the ultimate beneficiary, of course, but the public expects to pay for research indirectly through the purchase of goods and services. Notable exceptions are the public contributions to research on cancer and infantile paralysis.

Many combinations of interest have financed research in the past. The sponsoring companies need not all be in the same industry. Two or more units in each of the following classifications or in two or more of these groups comprise most of the cases:

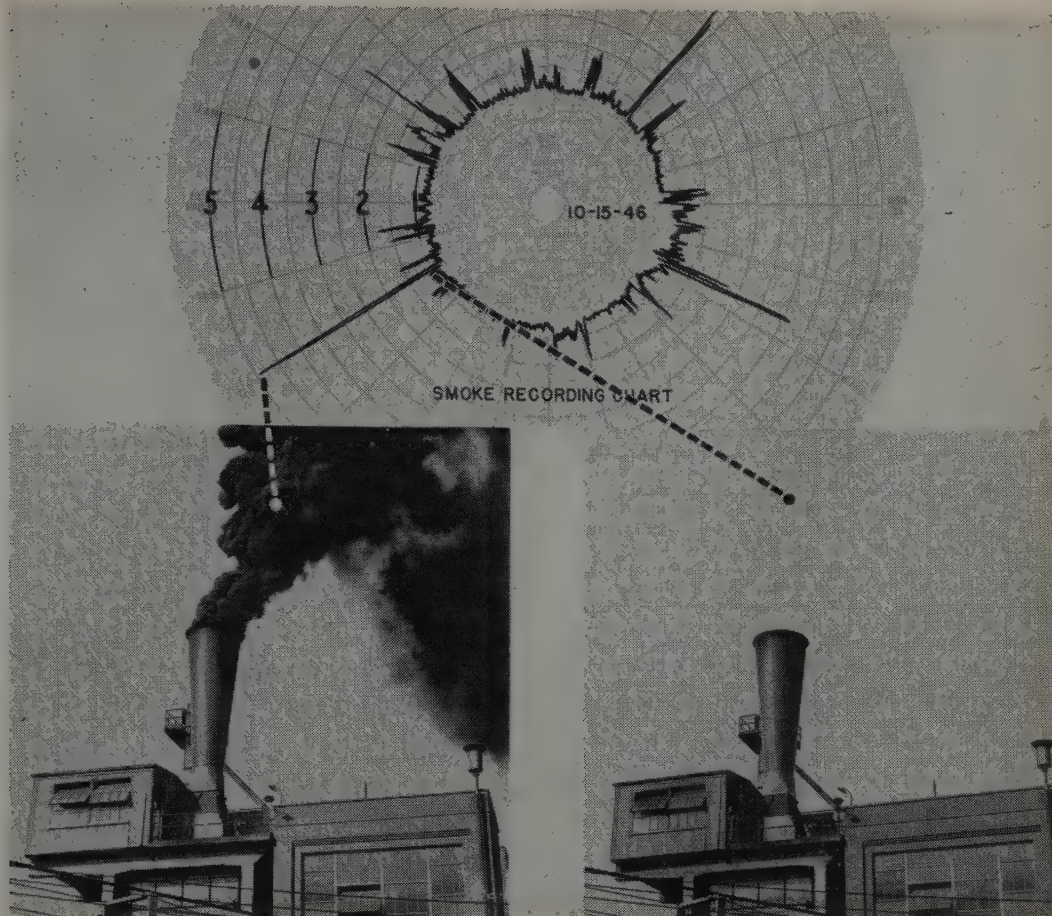
- |                          |                              |
|--------------------------|------------------------------|
| 1 Raw material producers | 8 Government departments     |
| 2 Manufacturers          | 9 Technical societies        |
| 3 Transporters           | 10 Research associations     |
| 4 Trade associations     | 11 Research institutions     |
| 5 Granges                | 12 Universities and colleges |
| 6 Consumers and users    | 13 Research foundations      |
| 7 Private individuals    | 14 Endowments                |

Numerous examples might be cited. For instance, Groups 1 and 3, comprising coal companies and railroads, are joint sponsors of a coal-mining-machine development under the Mining Development Committee of Bituminous Coal Research, Inc. Groups 2 and 10, comprising spreader-stoker manufacturers and the research agency of the bituminous-coal industry, are sponsoring a study of combustion in cooperation with one or two stoker users (Group 6),



**Fig. 4—Graphic proof of the effectiveness of modern overfire jets in eliminating smoke from boiler plants.**

(Left illustration shows smoke emission without jets in operation; at right, 45 sec after jets were turned on. The design and application of these modern overfire jets were effected under the sponsorship of Bituminous Coal Research, Inc., the national research agency of the bituminous-coal industry.)



and a research institution, (Group 11). Groups 2, 6, 9, and 11, consisting of manufacturers, users and the ASME, are sponsoring research on the metallurgy of high-temperature steam piping at an established contracting laboratory.

A decision on the selection of participants sets the stage for a membership plan, on the basis of which the solicitations are to be made. The simplest plan is for each sponsor to pay the research laboratory the sum agreed upon and to have the laboratory pay all expenses. No contracts between sponsors, nor a special corporation, are thus required. The payments and any incidental expenses of their representatives are charged off as current operating expenses by the sponsors.

Complexities, such as changing membership, necessity for paid employees, sponsorship at several laboratories, and commercialization of inventions, necessitates the formation of a more formal organization. A corporation not-for-profit is the usual type as there is rarely any intention to declare dividends on income that may be derived. The conduct of the program requires additional dues annually from members, and any income from royalties is used to supplement the dues.

Membership dues, income from various sources, investments, and in general, property of corporations not-for-profit, are free from taxation. The state and federal regulations governing such corporations should be consulted carefully as there are other restrictions besides nonpayment of dividends.

Of course, a corporation for profit can be organ-

ized and stock issued for funds invested. The general preference in industry is the nonprofit corporation because the contributions can be written off as operating expenses rather than continued on in the books as capital expenditures. Any indirect profits are to be made by the sponsor companies themselves, and they expect to use the research results without paying royalties.

In drawing up a charter and by-laws, copies of those in effect by similar corporations can be consulted and adapted by a lawyer. A typical object and purpose is:

To encourage, foster, and promote in any lawful manner the utilization of (products of industry) and of related equipment for and method of production, preparation, or utilization of (product of industry) by means of study, research, education, and other effort, or by developing equipment for and methods of production, preparation, or utilization of (product of industry) or any other lawful purpose.

All contributing companies may be classed as members, with voting power in proportion to total contributions to date. Powers and duties are vested in a board of directors, half of which are elected each year by the members. Officers elected each year by the board include a president, vice-presidents, secretary, and treasurer. In a small organization, the directors and officers are selected from the membership and serve without pay.

The research program may be outlined and administered by a committee which reports to the president and board. When the size of the program



warrants, a director of research and an administrative staff may be employed. As a first approximation, an administrative cost of 10 pct is reasonable.

**Basis of Participation:** How much money shall each member company contribute? Shall each pay equally or in proportion to size of company?

In theory, an equitable proportioning of the cost is determined by the opportunity each participant has to benefit. Improvements in a product would benefit a large manufacturer financially more than a smaller one because he manufactures and sells more of them. Payments prorated on the basis of sales volume are common. In this way companies ranging in size by as much as 100 to 1 can participate on a basis that is fair to all. In developing a new product, each participant would have a more nearly equal opportunity, in which case equal assessments might be made. This is also best when the companies are on a nearly equal footing in an industry, or where the total amount from each company is small. A compromise between proportionate and equal payments, in which a minimum and maximum fee are set, may satisfy the parties.

Quarterly or annual payments are convenient and are commonly made in advance.

Membership pledges or agreements can be simple signed pledges to contribute X-dollars a year in advance for Y-years for reports of research on subjects jointly agreed upon. More explicit legal documents can be executed if desired, binding the members and the research organization, but business firms prefer annual contributions without commitments to pay beyond one year at a time.

**The Time Factor:** Months and years of missionary work and the investment of thousands of dollars are required to organize the first research program in a sizable industry. Once the organization has been set up, and its value demonstrated, membership can be maintained and extended more easily. No rules can be given but a few examples will illustrate:

**Case 1:** A program administered by a Mining Development Committee of the bituminous-coal industry's research agency, and sponsored by 71 coal companies, railroads, and land companies. Objective: The designing of a machine for cutting and loading coal continuously in underground mining.

Phase 1	Date of conception .....	June, 1946
Phase 2	Preliminary discussions .....	7 months
Phase 3	Organization meetings .....	3 months
Phase 4	Financing .....	11½ months
Initial fund raised .....		\$258,000

**Case 2:** A program administered by a Spreader Stoker Research Committee and sponsored by seven stoker manufacturers and the bituminous-coal industry's research agency. Objective: To improve the utilization of coal on spreader-type stokers.

Phase 1	Date of conception .....	October, 1947
Phase 2	Preliminary discussions .....	4 months
Phase 3	Organization meetings .....	2 months*
Phase 4	Financing .....	4 months*
Initial fund raised .....		\$14,000

\* Partial overlapping.

**Case 3:** A program administered by a Locomotive Development Committee of the bituminous-coal industry's research agency, and sponsored by nine railroads and five coal companies. Objective: To develop a coal-burning gas-turbine railroad locomotive.

Phase 1	Date of conception .....	Summer of 1944
Phase 2	Preliminary discussions .....	6 months
Phase 3	Organization meetings .....	4 months
Phase 4	Financing .....	1 month
Initial fund raised .....		\$1,113,000
Total funds pledged .....		\$2,754,000

**Case 4:** A program administered by a joint committee and sponsored by 14 stove manufacturers and the bituminous coal industry's research agency. Objective: To develop principles for smokeless stoves, ranges, and water heaters.

Phase 1	Date of conception .....	June, 1940
Phase 2	Preliminary discussions .....	5 months
Phase 3	Organization meetings .....	4 months
Phase 4	Financing .....	4 months
Initial fund raised .....		\$70,600
Funds to date .....		\$180,700

**Conclusion:** Organizing and financing cooperatively sponsored research programs is an activity of management that engineers are well equipped to conduct. By increasing the number of participants in a program, the benefits of research are extended, and the research talent conserved. The procedures are systematic and do not include high-pressure sales tactics.

A few words of caution may be advisable. Short cuts in the steps outlined, such as expecting to organize a program in one meeting, are often doomed to failure. Personal visits and trips are far more effective than mimeographed letters and outlines, and are necessary for results.

Patience and diligence, optimism, and confidence are vital because the slowness in which decisions are received may be discouraging. The author has never found organization work or financing to proceed for long on the momentum of a few favorable decisions of prospective sponsors. Hard work is necessary at every step, but the rewards of accomplishment are indeed satisfying to the organizer and to the participants.

In the words of Prof. Warren K. Lewis:<sup>10</sup>

I feel that there is an unsolved problem as to how to handle the research needs of a small business, or at least one phase of those research needs, and that is the problem of competitive research in contradistinction to co-operative research. Co-operative research has been extraordinarily successful and has a large future ahead of it. By and large, there aren't many obvious rocks in the road.

## References

- <sup>1</sup>Industrial Research Laboratories of the United States. National Research Council, Washington, D. C., 8th Ed. 1946.
- <sup>2</sup>J. E. Hobson: Trends in Industrial Research. *Illinois Tech Engineer* (March 1948) 13, 10-12, and 30, 32, 34, and 36; (May 1948) 13, 14-16, 46, 48, and 50-54.
- <sup>3</sup>Science and Public Policy. Rept. of The President's Scientific Research Board (Aug. 27, 1947) 1, 12.
- <sup>4</sup>A Survey of Business Practices. *Bus. Record*. March 1947.
- <sup>5</sup>T. A. Boyd: Research—Everybody's Doing It Now. *ASTM Bull.* (May 1948) 53, 58.
- <sup>6</sup>Census of Manufacturers. U. S. Dept. of Commerce (1939) 1, 19.
- <sup>7</sup>Trade and Professional Associations of the United States. U. S. Dept. of Commerce (1942) 244.
- <sup>8</sup>Trade Association Industrial Research. U. S. Dept. of Commerce, Industrial Series no. 77 (1948).
- <sup>9</sup>Maurice Holland: The Place of Research in the Corporate Structure. Research Conference, Illinois Manufacturers Association, May 27, 1948.
- <sup>10</sup>The Future of Industrial Research. Standard Oil Dev. Co., New York, N. Y. (1945) 135-136.



# Operating Data for a Bird Centrifuge

by A. C. Richardson and Orville R. Lyons

Operating data is presented for a Bird centrifuge used to dewater coal treated at one preparation plant. The data include: (1) percentages of solids in centrifuge feed, cake, and effluent and the plant circulating water for a period of four consecutive operating shifts, and (2) screen-sizing data for a number of centrifuge feed, cake, effluent, and combined cake-effluent samples. An evaluation is included of the relationships between fine solids in the centrifuge feed and the total moisture content of the centrifuge cake.

THE Coal Division of Battelle Memorial Institute, during the course of an investigation conducted for a coal producer, carried out extensive sampling of the fine-coal section of a preparation plant. The information obtained during the testing period was used to evaluate the overall plant operation and provided the basis for recommended changes in the plant flowsheet. Included in the information were data for the results obtained when using a Bird centrifuge to dewater the particular coal being treated. At the completion of the investigation, the

A. C. RICHARDSON, Member AIME, is Supervisor, Minerals Processing Division, Battelle Memorial Institute, Columbus, Ohio, and ORVILLE R. LYONS, Member AIME, is Coal Preparation Engineer, Heyl and Patterson, Inc., Pittsburgh, Pa.

AIME New York Meeting, Feb. 1950.

TP 2818 F. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Oct. 14, 1949.

coal producer gave Battelle permission to publish the operating data for the Bird centrifuge in order to make this information available to the coal industry. The authors wish to take this opportunity to express their appreciation to this company and its officials for their willing cooperation in the matter of publication.

**Description of Preparation Plant:** The preparation plant on which the data were obtained is primarily a jig plant. The coal being washed is a low inherent moisture, friable, Pocahontus-type. The washed coal from the jig is dewatered and sized at  $\frac{1}{4}$  in. by a Parrish-type dewatering and sizing screen. The minus  $\frac{1}{4}$ -in. coal and the underflow water from the shaker screen flow to a sludge tank where, after the coal settles, the coal is conveyed by a flight conveyor into a pump sump. The pump, a 300-gpm sand pump, delivers a mixture of fine coal and water to a Bird centrifuge which produces a dewatered cake and an effluent. At the time when the investigation was made, the centrifuge effluent was returned to the sludge tank and the cake product was mixed with a coarser size of coal prior to loading into railroad cars. The semiclarified water from the sludge tank overflowed into a pump sump from which it was pumped to the jig.

As originally designed, the preparation plant was intended to operate with a closed water system but it was found, soon after operations started, that for various reasons this could not be done. In order to provide for removal of excess water from the system and to aid in maintaining the desired minimum percentage of solids in the circulating water, the plant was operated to provide from the sludge tank a continuous overflow of water and fine solids that was sent to waste.

The centrifuge feed represents approximately 35 tons per hour of minus  $\frac{1}{4}$ -in. coal in the form of a pulp averaging 38 pct solids.

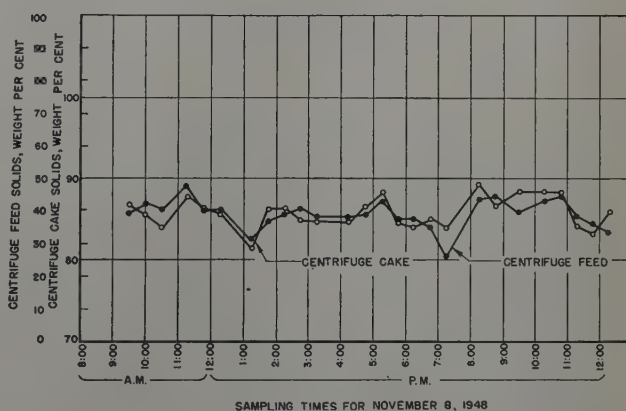


Fig. 1—Relation between solid contents of Bird centrifuge feed and cake.

**Sampling Procedure:** In order to obtain basic data for determining the relationships existing in the fine-coal section of the preparation plant, samples were obtained for: (1) the centrifuge feed, (2) the centrifuge cake, (3) the centrifuge effluent, and (4) the circulating water. The sampling was arranged to start on a Monday, in order to begin operations with a sludge tank full of fresh water, and was continued long enough to allow sufficient time for fine solids to build up within the water system or for a two-day, four-shift operating period. All samples were taken at half-hour intervals except when plant stoppages occurred. After a stoppage the plant was

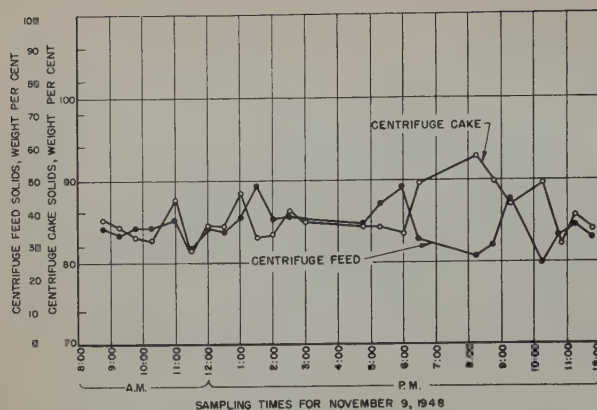


Fig. 2—Relation between solid contents of Bird centrifuge feed and cake.

allowed to operate for 15 min prior to the resumption of sampling.

The pulp samples, or the centrifuge feed, centrifuge effluent, and the circulating water, were obtained by momentarily diverting the full, pulp stream. The centrifuge-cake samples were obtained by cutting out, by means of a shovel, small increments of the cake from the total cake as it was being conveyed away from the centrifuge. These increments were taken over a period of 2 to 3 min and a sufficient number of increments were taken to provide a total individual sample weight of approximately 35 lb. The pulp samples were placed in glass jars and sealed. The centrifuge-cake samples were placed in 10-gal garbage cans which were weighed prior to and immediately after being filled with the sample.

The glass jars and garbage cans containing samples were labeled inside and out, were securely fastened to prevent spillage, and were shipped to Battelle.

**Centrifuge Performance:** Table I contains the data for percentages of solids and total moisture for all of the samples obtained. A study of table I reveals that: (1) the centrifuge-feed solids varied from 25.3 to 47.9 pct by weight with a normal value of approximately 38 pct, (2) the centrifuge-cake solids varied from 81.5 to 92.8 pct by weight with a normal value of approximately 85 pct, (3) the centrifuge-cake

total moisture varied from 7.2 to 18.5 pct with a normal value of approximately 15 pct, (4) the centrifuge-effluent solids varied from 1.5 to 5.5 pct by weight with a gradual increase in the percentage of solids throughout most of the sampling period, and (5) the circulating-water solids varied from 3.9 to 11.6 pct by weight with a gradual increase in the percentage of solids throughout most of the sampling period.

Figs. 1 and 2 show the relationship existing between the solid contents of the centrifuge feed and the centrifuge cake. A study of these figures indicates that the solid content of the centrifuge cake varied directly with the solid content of the centrifuge feed until approximately 5:00 p.m. on November 9, at which time the relationship changed and the solid content of the cake started to vary inversely with the solid content of the feed. The observations contained in table I show that at 5:00 p.m. the pump used to elevate fine coal from the sludge tank to the Bird centrifuge started to give trouble and that pumping operations were erratic from 5:00 p.m. until midnight. Fresh water was added to the sludge tank, in considerable quantity, almost continuously from 5:00 p.m. until midnight, supposedly to help the fine-coal solids flow into the sump, from

Table I. Plant-performance Data

Date	Sampling Time	Bird Centrifuge				Circulating-water Solids, Wt, Pct	Observations
		Feed Solids, Wt, Pct	Cake Solids, Wt, Pct	Cake Moisture, <sup>a</sup> Wt, Pct	Effluent Solids, Wt, Pct		
Nov. 8	9:30 a.m.	39.8	86.9	13.1	1.9	3.9	Preparation plant started at 8:00 a.m.
	10:00 a.m.	42.3	85.6	14.4	2.1	4.5	
	10:30 a.m.	40.8	84.5	15.5	2.6	5.2	
	11:15 a.m.	48.1	88.1	11.9	2.2	6.1	
	11:45 a.m.	40.3	86.6	13.4	1.7	6.5	
	12:15 p.m.	40.4	85.9	14.1	1.9	6.7	
	1:15 p.m.	30.6	81.6	18.4	3.0	6.5	
	1:45 p.m.	37.3	86.3	13.7	3.4	6.2	
	2:15 p.m.	39.2	86.3	13.7	3.1	6.2	
	2:45 p.m.	40.6	85.2	14.8	3.2	6.3	
	3:15 p.m.	37.9	84.9	15.1	3.1	6.5	Plant shut down between shifts from 3:30 p.m. until 4:00 p.m.
	4:15 p.m.	37.7	84.7	15.3	3.3	6.1	
	4:45 p.m.	39.1	86.7	13.3	3.2	6.7	
	5:15 p.m.	43.5	88.6	11.4	3.8	6.8	
	5:45 p.m.	37.1	84.7	15.3	3.4	7.0	
	6:15 p.m.	37.6	84.4	15.6	3.5	7.4	
	6:45 p.m.	35.2	85.0	15.0	3.8	7.8	
	7:15 p.m.	26.0	84.2	15.8	4.1	7.3	
	8:15 p.m.	43.0	89.4	10.6	3.4	7.9	
Nov. 9	8:45 p.m.	44.2	86.7	13.3	3.4	8.1	Plant waiting on coal 7:30 p.m. until 8:00 p.m.
	9:30 p.m.	39.4	88.5	11.5	3.3	8.5	
	10:15 p.m.	42.8	88.5	11.5	3.4	8.4	
	10:45 p.m.	44.4	88.2	11.8	3.4	9.2	
	11:15 p.m.	38.6	84.3	15.7	4.0	8.7	
	11:45 p.m.	35.9	83.7	16.3	4.1	8.4	
	12:15 a.m.	33.5	86.2	13.8	4.2	7.8	
	8:45 a.m.	35.0	84.8	15.2	4.6	9.2	
	9:15 a.m.	33.3	84.2	15.8	5.0	9.2	
	9:45 a.m.	34.9	83.1	16.9	5.3	9.0	
	10:15 a.m.	35.0	82.4	17.6	5.5	9.6	
	11:00 a.m.	38.1	87.5	12.5	5.0	10.3	
	11:30 a.m.	29.4	81.5	18.5	5.4	10.0	
	12:00 noon	35.3	84.0	16.0	5.4	11.3	
	12:30 p.m.	33.9	84.2	15.8	5.2	9.9	
	1:00 p.m.	38.6	88.3	11.7	5.0	10.0	
	1:30 p.m.	48.6	83.0	17.0	4.0	10.5	
	2:00 p.m.	38.2	83.3	16.7	5.5	11.2	
	2:30 p.m.	39.0	86.1	13.9	5.1	10.4	Plant shut down between shifts from 3:30 p.m. until 4:30 p.m.
	3:00 p.m.	38.1	85.1	14.9	5.1	10.8	
	4:45 p.m.	36.6	84.4	15.6	5.4	10.4	
	5:15 p.m.	42.7	84.4	15.6	4.7	10.8	Fire hose connected to 300 gpm sand pump and fresh water added to insure continuous flow.
	6:00 p.m.	47.9	83.3	16.7	5.3	11.0	
	6:30 p.m.	32.1	89.5	10.5	3.0	11.6	
	8:15 p.m.	26.9	92.8	7.2	1.4	10.4	Erratic centrifuge-feed conditions from 5:00 p.m. until 12:00 midnight.
	8:45 p.m.	30.5	89.5	10.5	1.5	9.8	
	9:15 p.m.	44.7	87.1	12.9	3.6	9.7	
	10:15 p.m.	25.3	89.8	10.2	1.7	9.7	
	10:45 p.m.	33.5	82.0	18.0	5.2	9.6	
	11:15 p.m.	36.5	85.6	14.4	3.6	10.0	
	11:45 p.m.	33.3	84.1	15.9	3.7	10.7	

<sup>a</sup> Seam moisture is approximately 1.5 pct.



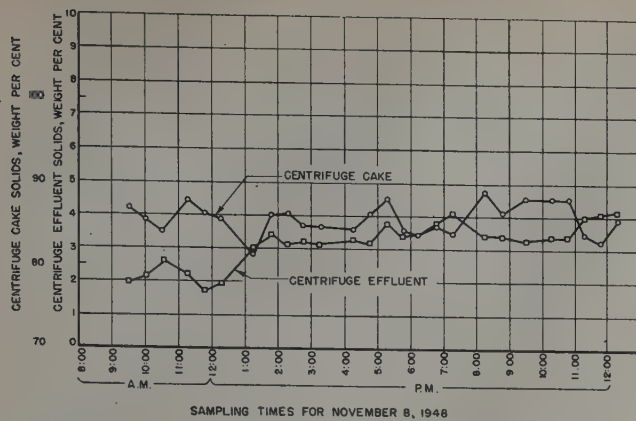


Fig. 3—Relation between solid contents of Bird centrifuge cake and effluent.

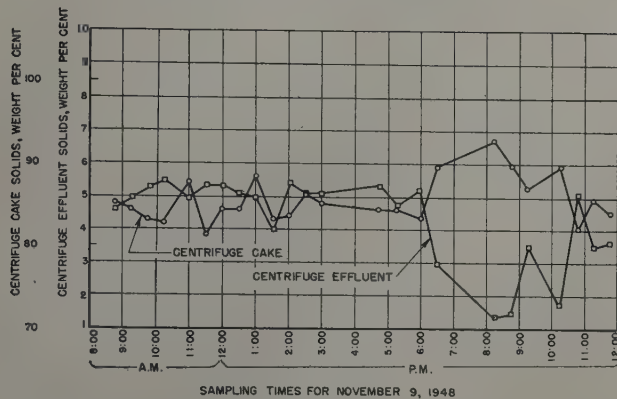


Fig. 4—Relation between solid contents of Bird centrifuge cake and effluent.

Table II. Screen-sizing Data for Bird Centrifuge Feed Samples Taken on November 9, 1948

Mesh	Weight, Per Cent					
	11:30 a.m.	12:00 Noon	1:30 p.m.	6:00 p.m.	8:15 p.m.	8:45 p.m.
—3 + 6	14.6	14.8	12.0	8.1	15.7	16.7
—6 + 10	22.6	19.5	27.9	23.8	40.3	26.3
—10 + 14	8.9	7.2	12.7	15.0	13.9	10.9
—14 + 20	10.8	9.5	13.4	14.7	6.2	11.4
—20 + 28	10.0	9.7	9.9	11.7	6.4	8.9
—28 + 35	5.7	7.2	5.9	6.7	3.3	5.7
—35 + 48	5.9	8.1	5.1	6.5	3.9	5.9
—48 + 65	4.3	6.1	3.4	3.6	2.7	4.1
—65 + 100	3.4	4.8	2.4	2.5	2.1	3.2
—100 + 150	2.3	3.0	1.3	1.5	1.4	1.8
—150 + 200	1.9	1.7	1.1	1.3	1.0	1.2
—200	9.6	8.4	4.9	4.6	3.1	3.9

which the centrifuge feed was withdrawn by means of the sand pump. Addition of fresh water to the sludge tank apparently diluted the feed pulp to some extent, as the 10 feed samples taken from 11:00 a.m. until 5:00 p.m. had an average of 37.6 pct solids, while the 10 feed samples taken from 5:00 p.m. until 11:45 p.m. had an average of 35.3 pct moisture, but this difference is not great enough to be responsible for the reversed relationship.

Table II contains screen-sizing data for samples of the centrifuge feed taken before and after 5:00 p.m. on November 9. A study of the data contained in table II reveals that the minus 200-mesh material in the centrifuge feed averaged 7.6 pct in the samples obtained prior to 5:00 p.m. and averaged only 3.9 pct in the samples obtained after 5:00 p.m. This marked reduction in the amount of minus 200-mesh solids in the centrifuge feed was evidently the result

of a classifying action created by overflowing the sludge tank and wasting excess water and entrained coal.

From the evidence available, therefore, the inverse relationship between the cake solids and the feed solids after 5:00 p.m. on November 9 appears to be the result of centrifuging a feed from which

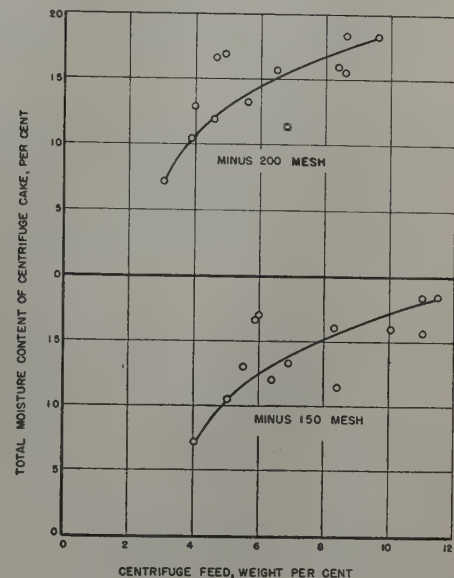


Fig. 5—Relation between fine solids in Bird centrifuge feed and moisture content of Bird centrifuge cake.

approximately 50 pct of the minus 200-mesh solids had been removed by classification. It seems indicated that the moisture content of the centrifuge cake obtained after 5:00 p.m. on November 9 was considerably lower than what it would have been if the additional minus 200-mesh material had been present in the centrifuge feed.

Figs. 3 and 4 show the relationship existing between the solid contents of the centrifuge cake and the centrifuge effluent. A study of the figures indicates, that, in general, the solid content of the centrifuge effluent varied inversely with the solid content of the centrifuge cake and that this inverse relationship was most pronounced from 5:00 p.m. until midnight on November 9. Evidently the centrifuge-cake moisture content and the effluent-solid content are proportional to the amount of fine solids in the centrifuge feed. The data presented in table III were used in the preparation of fig. 5 which shows the relationship existing between the minus

Table III. Data for Fine Solids in the Centrifuge Feed and for Moisture in Centrifuge Cake

Sample	Weight, Per Cent		Centrifuge Cake Moisture, Per Cent
	Minus 150-mesh	Minus 200-mesh	
Nov. 8, 9:30 a.m.	5.5	4.0	13.1
11:15 a.m.	6.4	4.6	11.9
1:15 p.m.	11.0	8.6	18.4
4:45 p.m.	6.9	5.6	13.3
7:15 p.m.	11.0	8.6	15.8
9:30 p.m.	8.4	6.8	11.5
Nov. 9, 11:30 a.m.	11.5	9.6	18.5
12:00 Noon	10.1	8.4	16.0
1:30 p.m.	6.0	4.9	17.0
6:00 p.m.	5.9	4.6	16.7
8:15 p.m.	4.1	3.1	7.2
8:45 p.m.	5.1	3.9	10.5
11:45 p.m.	8.3	6.5	15.9

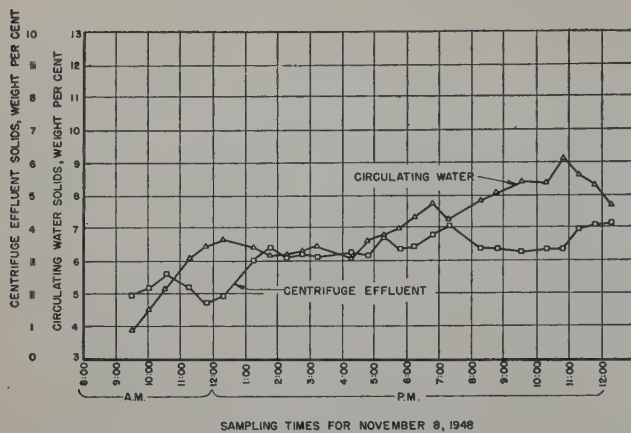


Fig. 6—Relation between solid contents of Bird centrifuge effluent and circulating water.

150-mesh and minus 200-mesh solids in the centrifuge feed, in per cent, and the total moisture content of the centrifuge cake, in per cent. The moisture content of the centrifuge cake varies with the percentage of fine solids in the centrifuge feed but the relationship is not exact enough to allow a more definite conclusion than that the cake moisture apparently can be reduced to less than 10 pct total moisture by the removal of fine solids from the feed material.

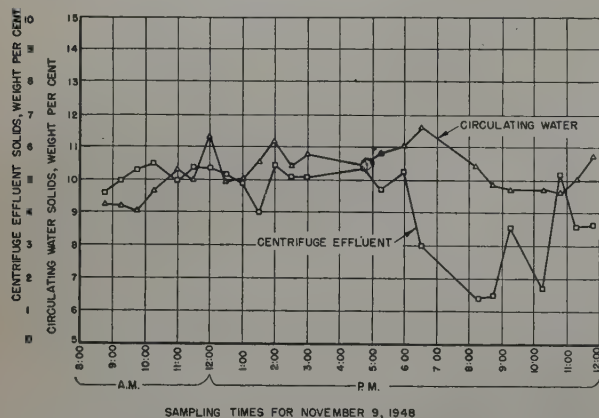


Fig. 7—Relation between solid contents of Bird centrifuge effluent and circulating water.

Table IV. Infrsizer Data for Centrifuge Effluent

Cone No.	Approximate Average Size of Solids, Microns	Effluent Solids, Wt, Pct
1		0.1
2		0.2
3		0.7
4	74	1.7
5	48	2.3
6	28	1.6
Bag	14	93.4
Total		100.0

Figs. 6 and 7 show the relationship existing between the solid contents of the circulating water and the centrifuge effluent. A study of figs. 6 and 7 indicates that, in general, the solid content of the circulating water varied with the solid content of the centrifuge effluent. The return of effluent solids to the sludge tank apparently was responsible for the buildup of solids in the circulating water.

In order to determine the size composition of the effluent solids, one of the samples was infrasized by means of a Haultain infrasizer. Table IV contains the results of the infrasizer test. It is evident that the effluent solids are extremely fine and that their removal from the circulating-water system, which appears necessary on the basis of the buildup condition determined, presents a difficult problem.

Table V. Comparative Screen-sizing Data for Bird Centrifuge Feed and Product Samples

Mesh	9:30 A.M. Nov. 8			6:00 P.M. Nov. 9		
	Feed Wt, Pct	Cake Wt, Pct	Composite Cake Plus Effluent Wt, Pct	Feed Wt, Pct	Cake Wt, Pct	Composite Cake Plus Effluent Wt, Pct
-3 + 6	21.7	24.0	23.3	8.1	11.8	10.9
-6 + 10	26.2	20.7	20.1	23.8	17.2	15.9
-10 + 14	9.0	14.3	13.9	15.0	13.9	12.9
-14 + 20	8.4	7.0	6.8	14.7	9.5	8.8
-20 + 28	7.7	6.9	6.7	11.7	8.6	7.9
-28 + 35	5.3	5.5	5.3	6.7	6.7	6.2
-35 + 48	6.2	3.7	3.6	6.5	6.4	5.9
-48 + 65	4.4	4.4	4.3	3.6	5.8	5.4
-65 + 100	3.6	3.7	3.6	2.5	5.4	5.0
-100 + 150	2.0	2.1	2.0	1.5	3.0	2.8
-150 + 200	1.5	1.9	1.9	1.3	4.6	4.2
-200	4.0	5.8	8.5	4.6	7.1	14.1

A study of this nature would be incomplete without some discussion of the amount of degradation that occurs when dewatering fine coal by means of a Bird centrifuge. Table V contains comparative screen-sizing data for samples of the centrifuge feed and cake and for composite samples of the cake plus the effluent solids. It is evident that considerable degradation does occur. The sample for 9:30 a.m. November 8 shows an increase in the minus 200-mesh fraction from 4.0 pct in the feed to 8.5 pct in the combined cake-effluent product, while the sample for 6:00 p.m. November 9 shows an increase from 4.6 pct in the feed to 14.1 pct in the combined cake-effluent product. There are some inconsistencies in the screen analyses, in that in both cases the minus 3-plus 6 mesh fraction of the combined product represents a larger percentage than that shown possible by the screen analyses of the feed. However, these inconsistencies are the result of a time lag in the flow of material through the centrifuge when taking instantaneous samples and the general conclusion that considerable degradation occurs is still valid.

**Summary:** This paper presents operating data for a Bird centrifuge working on the coal treated at one preparation plant. For this particular installation, when treating a minus ¼-in. feed in the form of a pulp that had a normal value of approximately 38 pct solids, the machine had the following characteristics: (1) the centrifuge-cake total moisture had a normal value of approximately 15 pct, (2) the centrifuge-effluent solids varied from 1.5 to 5.5 pct by weight with a gradual increase in the percentage of solids throughout most of the sampling period, and (3) the moisture content of the centrifuge cake varied to a considerable extent with the percentage of minus 150-mesh and minus 200-mesh solids in the centrifuge feed.

It would be helpful to the coal industry if similar data could be published for other feed conditions or for other types of equipment. It is the authors' hope that operators possessing such information will make it available to the industry in the near future.



# Production of Graded Glass Sand by Grinding and Classification

In a laboratory study of grinding and classification of silica sand, a satisfactory means of producing the medium-fine specification sand desired by producers of flint-glass containers was developed.

The best procedure consisted of ball-mill grinding and double classification, which gave a properly sized sand containing less iron than the feed.

by M. M. Fine

THE problem of producing a uniform, medium-fine sand for glass-furnace feed has been of interest to the glass-container industry for many years. In the present investigation of the problem, conducted by the Bureau of Mines in cooperation with the Owens-Illinois Glass Co., Alton, Ill., a satisfactory method of producing such sand was developed. The investigation involved continuous-grinding studies of glass sand with flint and porcelain pebbles, steel balls and rods, and a simultaneous study of classification.

M. M. FINE, Member AIME, is Metallurgist, Rolla Branch, Metallurgical Division, Bureau of Mines, Rolla, Mo.

Laboratory work for this report was completed April 1949. This paper presents the results of work done by the Rolla, Mo., Branch, Metallurgical Division, of the Bureau of Mines.

AIME New York Meeting, Feb. 1950.

TP 2814 H. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Oct. 5, 1949.

The most satisfactory procedure consisted of grinding in a pebble or ball mill followed by a double classification: hydraulic classification to separate the oversize for return to the grinding unit and mechanical classification to remove fines and impurities from the ground sand. The proper application of this flowsheet usually yielded a graded sand containing less iron than the feed even when grinding was done in a steel-lined ball mill.

The amount of sand ground into the so-called "dusting" sizes was, in all cases, less than 10 pct. Experiments were made which showed that this very fine sand could be beneficiated to a usable material.

## Background

The advantages in utilizing a uniform, medium-fine sand as a feed for glass furnaces has motivated a considerable amount of research on the problem of producing such sand by glass producers and others. For the glass producer, the faster melting rate of such a graded sand offers the possibilities of reduced fuel costs and increased furnace capacity. In addition, there is the likelihood of improving the quality of the melt and thereby reducing the number of seeds (air bubbles) and stones (unmelted grains) in the finished ware.

Although the melting rate of a glass batch depends on other factors in addition to the grain size of the sand, there is experimental evidence to show that grain size influences the melting rate considerably. Potts, Brookover, and Burch<sup>1</sup> found that when the raw materials (sand, soda ash, and limestone) were always matched in size, there was a great decrease in the melting time as the grain size of the materials was decreased. They found, too, that when the grain

sizes of the soda ash and limestone were held constant at 40 to 60 mesh, the minimum melting time occurred when 40 to 60 mesh sand was used. The increase in melting time with decrease in grain size beyond 40 to 60 mesh was due, they postulated, to accelerated demixing or separation of batch during melting as the grain size of the sand decreased in relation to the grain size of the other materials.

An earlier investigator<sup>2</sup> found that the highest rate of melting and purifying ordinary glass was obtained when medium-size sand (0.385 to 0.120 mm, correspondingly roughly to 40 to 120 mesh) was used. Although batches containing smaller-size sand were found to fuse more rapidly, the time required to remove air bubbles from such melts actually increased the total duration of the process. Coarse-grained sand, it was stated, increased greatly the times of both melting and purifying.

Although the laboratory data are difficult to confirm on a plant scale, observations by the Owens-Illinois Glass Co. indicate a higher furnace capacity of better-quality ware when a medium-fine sand is used in the batch.

During the recent war, the glass producer was inclined to overlook a considerable variation in raw materials in the interest of higher production. For instance, table I shows the wide range in the amounts of various screen fractions in sand supplied by one producer to the Terre Haute plant of Owens-Illinois. The sand, from the northern Illinois St. Peter sandstone, represented material used by that plant over a period of a few years during the war.

Since the war, the glass industry has insisted on

Table I. Variation in Amounts of Sand Screen Fractions during War

Size, Mesh	Range in Amount of Size, Per Cent
Plus 20 .....	0.0 - 0.3
20 to 30 .....	1.2 - 7.2
30 to 40 .....	16.4 - 50.6
40 to 50 .....	15.4 - 43.5
50 to 60 .....	5.0 - 16.6
60 to 80 .....	3.0 - 25.2
80 to 100 .....	0.8 - 13.4
100 to 140 .....	0.6 - 13.8
Minus 140 .....	0.2 - 11.9

Table II. Postwar Sand Shipments from Two Producers, Screen Analyses

Size, Mesh	Range in Amount of Size, Per Cent	
	Sand A	Sand B
Plus 20 .....	0.0 - trace	0.0 - trace
20 to 30 .....	1.9 - 3.4	0.4 - 1.7
30 to 40 .....	40.7 - 59.3	2.8 - 6.9
40 to 50 .....	30.5 - 46.4	13.6 - 22.4
50 to 60 .....	5.6 - 7.3	12.0 - 16.3
60 to 80 .....	2.8 - 7.6	29.4 - 37.4
80 to 100 .....	0.4 - 1.8	12.0 - 13.9
100 to 140 .....	trace - 0.5	8.5 - 10.3
Minus 140 .....	trace - 0.1	4.4 - 6.3



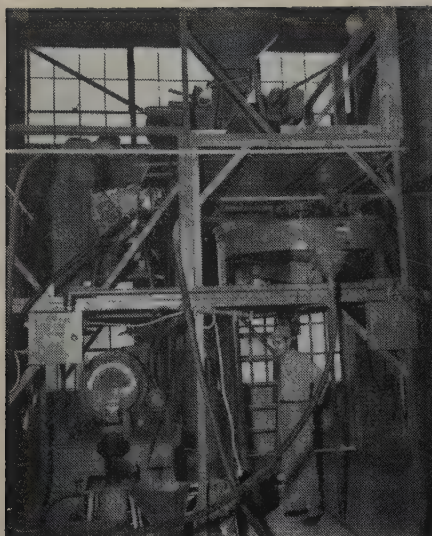


Fig. 1—Grinding and classification unit.



Fig. 2—First hydraulic classifier.

greater uniformity in glass sand, and sand A, table II, represents postwar shipments from another sand producer in the Illinois field. Wide ranges in the amounts of each screen fraction have been considerably reduced, but the sand is still regarded by the Owens-Illinois Co. as too coarse for an ideal glass sand.

A sand nearer the preferred screen analysis is sand B in table II. The sand (shipments from a deposit in eastern Missouri) corresponds to what is indicated by experience and experimental data to be desirable. Unfortunately, the operation is a small one and can supply only the sand requirements of the Alton plant.

#### Sand-preparation Practice

Although the American Ceramic Society and the National Bureau of Standards have established a recommended grading for glass sand, manufacturers have not adhered to the recommendation.<sup>3</sup> The practice in glass-sand preparation is directed, for the most part, to recovering the sand grains at the natural size in which they occur in the sandstone. Although the flowsheet may vary in different plants, the process is essentially one of liberating the individual grains in the sandstone, washing to remove excess fines and impurities and screening on about 20 mesh to separate any oversize. This is particularly true of most of the operations in the loosely cemented St. Peter sandstone of northern Illinois<sup>4</sup> and Missouri.<sup>5</sup>

In other areas, where more firmly consolidated sandstones are found, more rigorous treatment is required to liberate the individual sand grains. The operations in the Pennsylvania and West Virginia fields<sup>6</sup> have standardized on the chaser mill or wetpan grinder as a fine crusher before washing. In a very few installations, a tube or ball mill has been used to supplement the action of jaw crushers and washers in disintegrating compact sandstones.<sup>7, 8</sup> In none of these installations, however, is the grinding unit used to prepare a specification sand for the glass industry. Whatever comminution takes place is incidental to the objective of freeing the individual grains in the sandstone.

A plant in California that was recently completed and placed in operation<sup>9</sup> does prepare a fine-glass sand by ball-mill grinding. This plant, the Pacific Coast Division of Owens-Illinois Glass Co., near Corona, Calif., is producing glass sand from an

arkose sand in clay. The sand is scrubbed in various devices and then separated at 40 mesh in revolving screens. The primary undersize is suitable for colored glass only and is treated further in a separate circuit. The oversize is reduced to minus 40 mesh in a ball mill in closed circuit with a mechanical classifier and another revolving screen. The latter product, after washing, drying, and magnetic separation, is used in the manufacture of flint containers.<sup>10</sup>

This plant, of course, was built after the present investigation was well under way and represents, to some extent, an independent investigation of the same problem.

#### Statement of Problem and Previous Work

If the glass manufacturers should insist on a medium-fine specification sand, the sand producers will find it necessary to choose one of several alternatives. These are: (1) mine only those deposits containing finer sands, (2) screen out and reject the unacceptable coarse fractions of other deposits, or (3) grind the coarse sands to glassmaker's specifications. To adopt either of the first two alternatives would leave a considerable amount of sand in the ground or in producers' stockpiles and would waste natural resources. An investigation of the third possibility led to the present study. There was no doubt that the coarse sand could be reduced to any size desired with any one of a variety of grinding machines. The problem was to do so without introducing materials, such as iron, that are deleterious to glass sand and with production of minimum amounts of extreme fines.

The problem has been under study in the laboratories of the Owens-Illinois Glass Co. for a number of years. Their efforts were on a small, batch scale, and the results were not conclusive. Grinding studies of glass sand were also conducted by other industrial laboratories but no acceptable products were made. The ground sands were either too fine, badly contaminated with iron, or both.<sup>10</sup>

#### Present Investigation

**Description of Sample:** The present investigation was made on a 35-ton sample of minus 20 mesh

Table III. Screen Analyses of Feed and of Ideal Glass Sand

Size, Mesh	Weight, Per Cent	
	Feed Sand	Ideal Glass Sand
Plus 35 .....	47.8	1 - 2
35 to 48 .....	28.9	25 - 30
48 to 80 .....	18.4	30 - 40
80 to 100 .....	2.8	
100 to 200 .....	1.9	36 <sup>a</sup>
Minus 200 .....	0.2	
Total .....	100.0	100.0

<sup>a</sup> Minus 80-mesh material with not more than 1 to 2 pct minus 200-mesh material. The minus 80-mesh material should not exceed 40 pct. It was further stated that if it were possible to conform to the other size requirements and keep the amount of minus 80-mesh material below 30 pct, it would be desirable to do so.



washed silica sand from the Ottawa Silica Co., Ottawa, Ill. The sample was essentially silica with small amounts of pyrite, rutile, tourmaline, iron oxides, and other heavy minerals. Some of the silica grains contained inclusions of the accessory minerals; others had surface stains of iron oxide. Chemical analysis showed 0.024 to 0.033 pct total iron reported as ferric oxide. The average sieve analysis of more than 25 samples of the feed sand collected during this investigation is shown in table III.

For purposes of the present study, the Owens-Illinois Co. set up a screen analysis of an "ideal glass sand" toward which the work was directed. It was not expected that the test products would match that analysis in all respects, but it was an excellent guide. The screen analysis of the "ideal glass sand" is also shown in table III.

**Grinding with Pebble Mill:** To avoid contamination of the sand by iron, the first series of experiments was made with a pebble-mill grinding circuit. The mill, 15 by 36 in., was lined with silex blocks and charged with 145 lb of 1 to 2½-in. porcelain pebbles. The ends were lined with rubber belting, and the mill had a 6-in. discharge. Other items in the experimental setup included a single-spigot, hydraulic classifier, a 3-ft bowl classifier, and necessary sand pumps. In addition, an 8-in. spiral was utilized to dewater the underflow from the hydraulic classifier and convey it back to the pebble mill. A photograph of the entire unit is shown in fig. 1.

The hydraulic classifier was the subject of considerable experimentation before a satisfactorily operating device was constructed. It was patterned after an open-bottom hydraulic classifier which was developed in another Bureau of Mines laboratory and described in another publication.<sup>11</sup> The fact that the present problem was concerned with the classification of a single mineral permitted some simplification. It was constructed of a piece of 4-in. pipe 26-in. long fitted with a conical bottom. Feed was introduced at a point 6 in. from the top, and the hydraulic water entered 9 in. from the bottom. Fig. 2 is a photograph of the classifier. The machine was fitted with a periodic pincers which has been used on other materials to give a semicontinuous discharge of spigot product. In this case, however, it was unsatisfactory; repeated opening and closing of the pincers caused such surging of the sand pulp that classification was impossible. A few tests soon showed the most satisfactory continuous spigot discharge was through a soft rubber hose constricted by a screw clamp.

Several factors in operating a revolving mill can be varied to influence the grain size of the product, among them being mill speed, size and amount of grinding mediums, pulp dilution, rate of feed, and circulating load. Actually, in the present investigation of pebble-mill grinding, it was practical to vary only the last two. To do otherwise would have complicated the investigation unduly.

The mill speed was held constant at 44 rpm or about 65 pct of the critical speed. The volume of the pebble charge, including voids, was somewhat less than half that of the mill. The pulp in the mill was held between 55 and 60 pct solids. At less than 55 pct solids the pulp became too dilute, and at much over 60, some of the smaller pebbles floated out the mill through the feed scoop.

The test procedure was as follows: crude sand

was fed to the hydraulic classifier, which was set to overflow a minus 35-mesh product. The coarse-sand underflow was pumped to the 8-in. spiral and conveyed to the pebble mill for grinding. The mill discharge was laundered to the same hydraulic classifier to close the circuit. The hydraulic-classifier overflow was pumped to the bowl classifier, where most of the extreme fines overflowed and the rake product represented the finished sand.

The circuit was controlled by taking frequent timed samples of the hydraulic-classifier overflow and wet-screening on a 35-mesh screen. Visual comparison of the quantity of sand retained on the screen with the total hydraulic-classifier overflow collected during an equal interval gave a rapid check on the operation. The hydraulic water was regulated to give an overflow containing 1 to 3 pct of plus 35-mesh sand. Amounts less than that caused excessive loss of finished fine sizes in the spigot underflow.

The results of a group of preliminary tests, showing the effect of increases in the circulating load upon the screen analysis of the finished product, are given in table IV. The circuit was first determined to have a capacity of about 2.0 lb per min at a circulating load of 90 pct. Increases in circulating load were then effected by a temporary rapid increase in feed rate until the desired sand load was attained. The feed rate was then reduced to an amount somewhat higher than the original value and the circuit balanced at the higher circulating load.

**Table IV. Variation in Grain Size with Circulating Load**

Test No. ....	1 90	2 170	3 230
Circulating Load, Per Cent .....			
Size, Mesh	Weight, Per Cent		
Plus 35.....	1.6	1.0	2.8
35 to 48.....	24.0	32.0	43.7
48 to 80.....	41.1	42.3	36.6
80 to 100.....	8.4	7.6	5.8
100 to 200.....	16.4	12.4	8.5
Minus 200.....	8.5	4.7	2.6
Total.....	100.0	100.0	100.0

The increase in the mean grain size with increase in circulating load was very evident, particularly the rapid increase in the quantity of 35 to 48-mesh sand and the decrease in the minus 80-mesh fraction. This same phenomenon was true, of course, with variations in the rate of new feed; a decrease in the feed rate would result in a corresponding decrease in the mean grain size of the product. These two factors, rate of new feed and circulating load, were the principal means of controlling the grain size and screen analysis of the finished sand.

Before proceeding with the next group of tests, the porcelain pebbles were replaced with a charge of flint pebbles. The wear on the porcelain spheres during the preliminary tests had been severe, and it was thought that flint would be more representative of commercial practice. The new grinding charge consisted of 155 lb of ¾ to 2-in. pebbles containing approximately 35 pct each of the 2 to 1½-in. and 1½ to 1-in. size and 30 pct of the 1 to ¾-in. size.

The flint pebbles had a higher specific gravity than the porcelain grinding medium and, as expected, their use increased the capacity of the pebble mill. The first few tests with the flint pebbles soon established the fact that higher feed rates and



circulating loads than with the porcelain were necessary to produce ground sand with a satisfactory screen analysis. Table V shows the screen analyses of all products from a test in which a reasonably satisfactory ground sand was produced.

Under the conditions noted above, the power consumption and the pebble and liner wear were determined in a prolonged test during which a total of 2900 lb of sand was ground. The silex-liner loss based on per ton of sand feed amounted to 2.1 lb, the pebble consumption was 4.1 lb and the power consumed was 9.6 kw-hr.

**Table V. Results of Flint Pebble-mill Grinding**

**a. Screen Analyses of Products**

Size, Mesh	Weight, Per Cent				
	Feed	Pebble Mill Discharge	Hydraulic Classifier Underflow	Bowl Sands	Bowl Overflow
Plus 35.....	47.9	31.9	50.8	1.0	
35 to 48.....	29.1	42.0	42.6	31.4	
48 to 80.....	18.8	18.0	6.4	43.2	
80 to 100.....	2.6	2.8	0.2	9.9	
100 to 200.....	1.6	3.1	0.0	12.1	7.8
Minus 200.....	0.0	2.2	0.0	2.4	92.2
Totals.....	100.0	100.0	100.0	100.0	100.0

**b. Operating Data and Iron Analyses of Products**

Product	Amount, Lb per Min	Weight, Per Cent	Analysis, Per Cent Fe <sub>2</sub> O <sub>3</sub>
Feed.....	2.29		0.033
Pebble mill discharge.....	5.94		0.026
Hydraulic classifier underflow.....	5.90		0.027
Bowl sands.....	2.11	95.5	0.026
Bowl overflow.....	0.10	4.5	0.100

The quantities of products given in table V, part b, column 2, were the average of several observed values taken while the test was in progress. The circuit was easily operated and, as shown, a weight balance of the products was readily made. The simple hydraulic classifier operated with surprising efficiency. Very little sand finer than 48 mesh was returned to the pebble mill for regrinding.

The finished product (the bowl sands) compared very favorably with the screen analysis of the hypothetical, ideal glass sands. It is also of interest to note that the process was one of benefaction as well as size reduction. The iron content of the bowl sand was materially lower than that of the feed. Some of the iron oxide in the sand was liberated by the grinding and reported in the fine overflow from the bowl classifier.

The recovery of ground sand was 95.5 pct. In other words, less than 5 pct of the feed was overground and rendered useless for glass-making purposes. However, it is possible that this fine bowl overflow, containing 0.10 pct iron oxide and more than 92 pct minus 200 mesh, could be recovered for use as ground silica.

In addition to the tests outlined above, others were made in which a vibrating screen, fitted with a 35-mesh brass cloth and equipped with sprays, was used as the sizing device. This circuit gave a good product but the screen cloth had a relatively short life before it became blinded. Frequent brushing of the screen was necessary to maintain an effective separation. It is probable that other types of screen cloth, such as slotted stainless steel, would have a longer effective life but neither the time nor equipment was available for such a study.



**Fig. 3—Pebble-mill and rod-ball mill.**

**Grinding with Rod Mill:** The next phase of the investigation was a brief study of grinding the sand in a rod mill. The rod-mill shell had the same external dimensions as the pebble mill so that the two could be used interchangeably on the same driving mechanism. The rod-mill shell was lined with Ni-Hard liners, and the ends were lined with ½-in. rubber belting so that the internal dimensions of the mill were 16½ by 35 in. The mill had a 12-in. discharge to facilitate rapid exit of the ground pulp and was charged with 300 lb of rods. The rods were ¾ and 1-in. SAE 4150 steel,\* which had been hard-

\* Contains 0.48 to 0.53 pct C, 0.75 to 1.00 pct Mn, 0.80 to 1.10 pct Cr, and 0.15 to 0.25 pct Mo, according to catalogue data.

ened by heat treating and oil quenching to give a surface hardness of 45 Rockwell C. Fig. 3 is a close-up showing both the rod and pebble mills.

The rod mill, of course, had a greater capacity than the pebble mill, so much more that the efficiency of the hydraulic classifier was considerably impaired. It then became necessary to conduct a series of tests during which various changes in the design of the classifier were investigated. After some experimentation the classifier shown in fig. 4 was developed and it gave good service in the subsequent work with rod and ball mills.

The new classifier utilized the basic 4-in.-diam classifying column of the other, but the top section (6-in. long) was 5 in. in diameter and the feed pulp was introduced at the bottom of the 5-in. section. The enlarged top section very effectively reduced the velocity of the entering pulp to a proper level, decreased surging, and contributed to smooth operation of the unit. The lower half of the classifier now consisted of an open-bottom, 4-in., classifying column enclosed in a 7-in. pipe with a conical bottom. The hydraulic water was introduced in the annular space between the two diameters. The function of the 7-in. pipe was to provide additional sand-storage capacity in the classifier.

Two changes were made in the operation of the circuit. The dry sand feed was introduced directly into the rod mill to improve the size distribution of the product, and the pulp consistency in the rod mill was increased to 64 to 65 pct solids. Otherwise, the flow of the pulp was the same as previously described.

To minimize oxidation in the mill and reduce the amount of iron picked up by the sand, the pulp was kept alkaline by introducing suitable reagents. The use of the alkaline pulp was not the whole solution to the problem of iron contamination, but there is no doubt it helped clean the surface of the sand grains. No attempt was made to determine the optimum quantity of reagents beyond ensuring that the pulp pH was between 9.0 and 9.6. In the rod-



**Table VI. Results of Rod-mill Grinding**

a. Screen Analyses of Products

Size, Mesh	Weight, Per Cent				
	Feed	Rod Mill Discharge	Hydraulic Classifier Underflow	Bowl Sands	Bowl Overflow
Plus 35 .....	45.5	32.2	54.5	4.0	
35 to 48 .....	29.2	38.0	36.4	33.2	
48 to 80 .....	19.9	20.2	8.5	41.2	
80 to 100 .....	3.4	3.6	0.5	9.6	1.5
100 to 200 .....	2.0	4.0	0.1	11.0	14.7
Minus 200 .....	0.0	2.0	0.0	1.0	83.8
Totals .....	100.0	100.0	100.0	100.0	100.0

b. Operating Data and Iron Analyses of Products

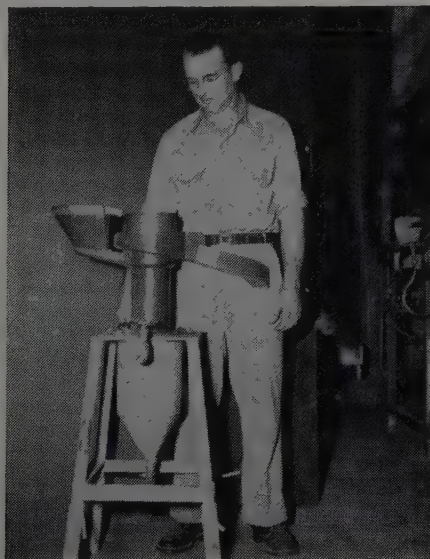
Product	Amount, Lb per Min	Weight, Per Cent	Analysis, Per Cent Fe <sub>2</sub> O <sub>3</sub>
Feed .....	4.93		0.030
Rod-mill discharge .....	14.56		0.050
Hydraulic-classifier underflow .....	10.16		0.034
Bowl sands .....	4.84	93.4	0.037
Bowl overflow .....	0.34	6.6	0.67

mill test, reported below, the reagents were sodium hydroxide and sodium silicate, 0.4 lb of the former and 1.0 lb of the latter per ton of sand feed. The results of a typical rod-mill test are given in table VI.

As shown in table VI, the new sand feed to the rod mill was 4.93 lb per min as compared to 2.29 for the pebble mill. The recovery of ground sand amounted to 93.4 pct, and 6.6 pct was discarded in the overflow from the bowl classifier. The new hydraulic classifier functioned satisfactorily, and, as before, very little minus 48-mesh sand was circulated.

The disposition of the iron was of interest as it affected the grade and value of the product. Most of the iron abraded from the grinding medium was fine enough to remain in suspension long enough to overflow the bowl classifier. The remainder increased the iron oxide content of the bowl sands to 0.037 pct as compared to the feed value 0.030 pct. Although this is not a large increase, it might be enough to render such sand unfit for certain types of flint-glass containers.

Much of the iron in the product was magnetic and could be removed magnetically. However, the addition of magnetic separation would add considerably to the cost of the process.



**Fig. 4—  
Second  
hydraulic  
classifier.**

In addition to a higher iron content, the product of the rod-mill circuit was somewhat coarse. Although it would have been possible to change the operation to produce finer sand, such changes (to increase the retention time in the mill) would tend to increase the iron content of the product also.

**Grinding with Ball Mill:** The final phase of the work was a study of sand grinding in a ball mill. The same flowsheet, including the grinding unit, was used, except that the rods were replaced with a charge of Moly-Cop forged-steel balls consisting of 50 lb of 2-in., 175 lb of 1½-in., and 75 lb of 1-in. balls. The same amounts of alkaline reagents were used in these tests to reduce the iron pick-up.

A few preliminary runs were made to become familiar with the action of the balls and to make necessary changes in the operating conditions. The results of a satisfactory test with the ball mill are detailed in table VII.

As shown in table VII, substitution of an equal weight of balls for rods permitted an increase in the rate of feed to 7.15 lb per min. The recovery of ground bowl sand was 90.8 pct, and 9.2 pct of the weight was in the fine bowl overflow.

The screen analysis of the bowl sands compared very favorably with that of the "ideal glass sand" and was one of the best produced during the investigation. Equally important was the fact that the iron content of this product was actually less than that of the feed. Chemical analyses for iron were made by Bureau of Mines and Owens-Illinois analysts and are shown in table VII. Although the analyses differ to some extent (due, probably, to differences in the methods of analysis), it is evident that the final bowl sand contained less iron than the feed.

**Table VII. Results of Ball-mill Grinding**

a. Screen Analyses of Products

Size, Mesh	Weight, Per Cent				
	Feed	Ball-mill Discharge	Hydraulic Classifier Underflow	Bowl Sands	Bowl Overflow
Plus 35 .....	48.7	26.2	47.2	2.6	
35 to 48 .....	28.2	33.6	36.8	24.7	
48 to 80 .....	17.9	24.5	12.9	45.2	
80 to 100 .....	3.5	5.7	1.9	11.4	0.5
100 to 200 .....	1.6	6.1	1.0	13.8	12.8
Minus 200 .....	0.1	3.9	0.2	2.3	86.7
Totals .....	100.0	100.0	100.0	100.0	100.0

b. Operating Data and Iron Analyses of Products

Product	Amount, Lb per Min	Weight, Per Cent	Fed. Bur. of Mines Analysis, Per Cent Fe <sub>2</sub> O <sub>3</sub>	Owens-Illinois Analysis, Per Cent Fe <sub>2</sub> O <sub>3</sub>
Feed .....	7.15		0.031	0.025
Ball-mill discharge .....	18.76		0.037	0.028
Hydraulic-classifier underflow .....	11.81		0.026	0.015
Bowl sands .....	6.38	90.8	0.026	0.020
Bowl overflow .....	0.65	9.2	0.40	0.42

Just why the ball mill gave a product much lower in iron than the rod mill is a matter for conjecture, but there were several contributing factors. The larger sand load carried by the ball mill probably cushioned the grinding medium, resulting in more iron-to-sand and fewer iron-to-iron contacts. The ball mill gave a finer grind; and, in so doing, the abraded iron was ground fine enough so that almost all of it overflowed the bowl classifier. Another consideration was the difference in composition and physical properties of the two iron-grinding medi-

ums. The balls were harder and may have been more abrasion resistant as well.

After determining the conditions under which a properly sized sand could be produced, a test during which 5 tons of sand were ground was made to note the wear on the balls and liners. The ball consumption was 1.1 lb per ton, and the liner wear amounted to 0.3 lb per ton of sand feed. These figures are not to be regarded as absolute determinations, the test was too short for that, but as an estimate of the iron consumption in the laboratory setup.

The power consumed during the same test was 4.2 kw-hr per ton of sand. This, of course, is an experimental value but it is likely that it could be reduced. The ball mill was not operated at maximum capacity for several reasons, but had it been possible to do so, the power per ton would have been less.

**Beneficiation Tests on Fine Overflow:** There are several factors, such as greater capacity per unit of mill volume and less power per ton of sand, which give the ball mill a definite advantage, in this case, over the other grinding units. The only objection which could be raised to the use of a ball mill for grinding glass sand is the amount of fines produced. In the test reported, 9.2 pct of the feed was overground and removed as the overflow in the bowl classifier. This is twice as much fines as was produced in the pebble-mill grind. There is, of course, the possibility of recovering and utilizing this fine-sand overflow, provided the iron content can be materially reduced.

To investigate this possibility, a substantial sample of the fine overflow from the ball-mill test was collected for experimental work. It was thickened, but was not permitted to dry, to keep oxidation of the contained iron at a minimum. A combination wet-screen analysis and sedimentation test was made on the material to note the size and iron distribution. The results of the test are tabulated in table VIII.

**Table VIII. Screen Analysis of Bowl Overflow**

Size, Mesh	Weight, Per Cent	Analysis, Per Cent Fe <sub>2</sub> O <sub>3</sub>	Per Cent of Total Fe <sub>2</sub> O <sub>3</sub>
Plus 100 .....	0.9	0.114	0.2
100 to 200 .....	13.4	0.033	1.0
200 to 270 .....	17.8	0.034	1.4
270 to 325 .....	9.7	0.061	1.4
325 to 400 .....	11.4	0.090	2.3
Minus 400 sand .....	11.5	0.180	4.7
Minus 400 slime .....	35.3	1.110	89.0
Total .....	100.0	0.44	100.0

Aside from the very small amount of plus 100-mesh sand, there was a marked increase in iron content with decrease in grain size. As a result of these data, several sedimentation and classification tests were made to recover a low-iron sand product, but the results were unsatisfactory. A combination of classification and low-intensity wet magnetic separation did recover a fairly good product, but the loss of sand was high. This scheme yielded a sand product containing 0.039 pct iron oxide, with a weight recovery of 59.5 pct.

Best results were obtained by separating the entire overflow in a Ferrofilter—a high-intensity, grid-type, wet magnetic separator. By the use of this machine a 90.6 pct weight recovery was made in a fine-sand product containing 0.037 pct iron. This product should be acceptable for many applications where ground silica is specified. Among these are

foundry use as mold wash, in production of glazes, and in the manufacture of such products as metal polishes, abrasive soaps, composition flooring, and paint fillers.

## Summary and Conclusions

The data collected during the investigation and presented in this paper show that a graded-glass sand acceptable to the glass-container industry can be readily produced. An acceptable sand, of medium fineness, was made by grinding and double classification of a washed silica sand. The amount of overgrinding was not excessive and the finished sand was free of iron contamination.

Of those grinding units investigated, the ball mill was favored for producing a most satisfactory product at highest capacity and lowest unit power. The ball mill did produce a higher percentage of extreme fines but this fine fraction was recovered as a bowl overflow and could be beneficiated to make a usable byproduct.

This process offers sand producers a simple means, should it become necessary, of producing the medium-fine sand which the glass container industry desires. A commercial unit should be able to make further simplifications not possible in the laboratory. For instance, a multispigot hydraulic classifier could possibly be used to separate both the too-coarse and too-fine fractions from the finished sand in a single operation. In the laboratory, both hydraulic and mechanical classification were utilized to make a properly sized sand.

## Acknowledgments

The author is indebted to E. W. Summers, Owens-Illinois Glass Co., Alton, Ill., who was largely responsible for initiating the investigation and who supplied much of the information in the introductory portions of this paper. The help of W. W. Slade of the Toledo, Ohio, laboratory of the same company, who assisted in some of the test work, was very welcome.

The advice and guidance of R. G. Knickerbocker, Chief, Rolla Branch, Metallurgical Division, Federal Bureau of Mines, Rolla, Mo., are gratefully acknowledged.

## References

- <sup>1</sup> J. C. Potts, George Brookover, and O. C. Burch: Melting Rate of Soda-lime Glasses as Influenced by Grain Sizes of Raw Materials and Additions of Cullet. *Jnl. Amer. Ceram. Soc.* (1944) **27**, (8) 225-231.
- <sup>2</sup> S. Ya. Raf.: *Keram. i. Steklo.* (1937) **13**, (12) 13-17 [*C.A.* 32, 2697<sup>g</sup>, 1938].
- <sup>3</sup> H. Ries: *Special Sands. Ind. Min. and Rocks.* p. 757. 1937. New York. AIME.
- <sup>4</sup> J. E. Lamar: *Geology and Economic Resources of the St. Peter Sandstone of Illinois.* Ill. State Geol. Survey *Bull.* 53, 1928.
- <sup>5</sup> H. L. Sheakley and D. J. Coolidge: Use of Silica Sand in the Glass Industry in Missouri. *Trans. AIME* (1947) **173**, 601; *Min. Tech.* (Nov. 1942) TP 1538.
- <sup>6</sup> W. R. Scholes: *Glass Sands. The Glass Industry* (1937) **18**, (9) 295.
- <sup>7</sup> John B. Hutt: *A Glass Sand Enterprise on the Pacific Coast. Eng. and Min. Jnl.* (1937) **138**, (12) 29.
- <sup>8</sup> Edmund Shaw: *Mining and Milling Methods and Costs at the Glass-Sand Plant of P. J. Weisel, Inc., Corona, Calif. Bur. of Mines. I. C.* 6937, 1937.
- <sup>9</sup> *Eng. and Min. Jnl.* (1949) **150** (1) 110.
- <sup>10</sup> E. W. Summers: Owens-Illinois Glass Co., Alton, Ill., private communication.
- <sup>11</sup> Will H. Coghill, G. D. Coe, and I. L. Feld: Performance of a Hydraulic Classifier Designed to Incorporate Four Hitherto Neglected Principles. *Bur. of Mines. R.I.* 3844, Dec. 1945.



# Production of Ammonium Sulphate and Manganese Oxides

Manganese Products, Inc. has developed a chemical process for the recovery of high-grade manganese oxides from low-grade manganese ores. Ammonium sulphate is also produced. Manganese ore is leached with sulphurous-sulphuric acids. The impurities are removed with ammonia, heat, and air. Manganese oxides are produced by addition of more ammonia and air under pressure. Ammonium sulphate is recovered from the remaining solution.

by Norman Ketzlach

"**M**ANGANESE today overshadows all strategic materials in our planning for national security," according to a report issued recently by the American Manganese Producers Association. By far the greater part of the manganese ore used in the United States is imported.

**NORMAN KETZLACH** is Chief Chemist of Manganese Products Inc., Seattle, Wash.

Third Annual Northwest Industrial Minerals Conference sponsored by the Columbia Section in cooperation with the Oregon and the North Pacific Sections, AIME, May 1949.

TP 2835 H. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received May 26, 1949.

"We have enough manganese ore in the United States to last from one hundred to three hundred years and longer, but most of the deposits are undeveloped. As in the case of copper, lead, and zinc, manganese ore, both foreign and domestic are low grade. Manganese ores have to be concentrated and processed. After being processed, domestic ores are higher grade than foreign." Most of the manganese ore found in this country, however, is not amenable to concentration by ore-dressing methods.

Manganese Products, Inc. has developed a sulphurous-sulphuric acid leaching process for the recovery of manganese from low-grade, intermediate manganese ores. By the term "intermediate manganese ores" is meant those ores in which the manganese in the form of manganese oxides is found to have two different valences, usually two and four, as is probable in  $Mn_2O_3$ , or the ore may be a mixture of manganous and manganic oxides. "Intermediate" manganese siliceous ores are ores in which the silica is a constituent of the ore and is loosely linked to the manganese by water of crystallization.

This process has been carried out with the type of manganese ore found in the Olympic Peninsula of western Washington; Clough, N. C.; and Lava Hot Springs, Idaho, as well as just recently with manganese ore from Las Vegas, Nev.

Table I is a typical analysis of each of these ores.

**Table I. Manganese Ore Analysis, Per Cent**

Constituent	Wash.	N. C.	Idaho	
			Lava Hot Springs	Pocatello
Mn (total)	25.9	17.9	27.9	31.5
Mn (tetravalent)	10.9	13.5	23.7	30.3
Mn (divalent)	15.0	4.4	4.2	1.2
SiO <sub>2</sub>	25.0		23.3	18.1
Fe <sub>2</sub> O <sub>3</sub>	4.95		10.5	10.0
Al <sub>2</sub> O <sub>3</sub>			2.0	0.5
BaO			8.1	
CaO	5.8	Trace	2.2	2.0
MgO	0.1	Trace	0.1	0.3
Ignition loss	5.8	11.1	11.2	

As far as the manganese leaching process is concerned, these ores differ primarily in the form in which the manganese oxides occur. The tetravalent manganese, the manganese existing in the dioxide form, is soluble in a sulphurous acid solution. To dissolve the divalent manganese, sulphuric acid is required.

To dissolve the type of ore found in Washington and North Carolina, a mixture of both sulphurous and sulphuric acid is used. Whereas to dissolve the type of manganese ore found in Idaho and also in Montana, sulphurous acid alone is sufficient. The Nevada manganese ore is of the same type as that found in Idaho as far as the leaching process is concerned.

It has long been known that manganese ores, with the exception of rhodonite, which is a true manganese silicate ore, can be dissolved readily in sulphurous and sulphuric acids.<sup>1</sup> The removal of the impurities such as silica, iron, and alumina from the resulting solution is the step that has been difficult.

The process worked out by Manganese Products has been carried through pilot-plant equipment. This process has also been extended to the production of ammonium sulphate.

**Leaching of Ore:** The manganese ore, ground to —65 mesh is leached with sulphurous acid and sul-

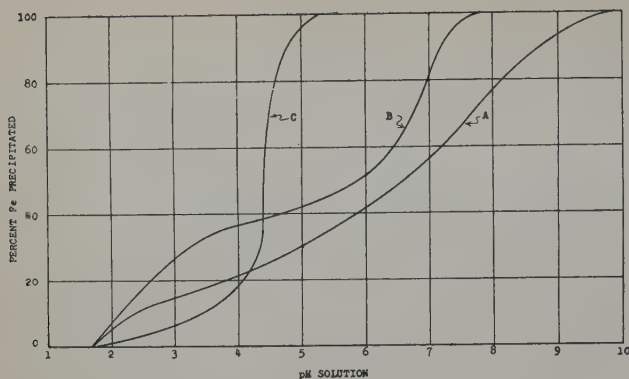
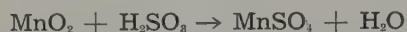


Fig. 1—Precipitation of iron vs. pH.

phuric acid at a temperature of 60°C. The sulphurous acid dissolves the quadrivalent manganese in the ore as is shown by the following equation:



Manganese sulphate formed according to this reaction acts as a catalyst to convert some sulphurous acid to sulphuric acid in the presence of air:



This sulphuric acid is capable of dissolving some of the divalent manganese:



Consequently, in the treatment of ores such as found in Idaho, Montana, and Nevada, in which most of the manganese is in the dioxide form, no sulphuric acid is added to the leach.

The formation of dithionates by this leaching step must be prevented:



Oxygen in the leaching step of the process tends to repress the formation of dithionates. By maintaining an oxygen, sulphur dioxide molal ratio of about four to one, no trouble is encountered. By this leaching operation 90 to 95 pct of the manganese in the ore is recovered.

Besides dissolving the manganese, some iron, alumina, and silica are also dissolved. The method of separation of these impurities from the manganese sulphate solution has been worked out by carefully controlling the pH, temperature, and aeration.<sup>2</sup>

**Removal of Impurities:** The precipitation of iron is shown graphically in fig. 1. Curve A, representing the run in which low temperature was used without the addition of air, indicates that under these con-

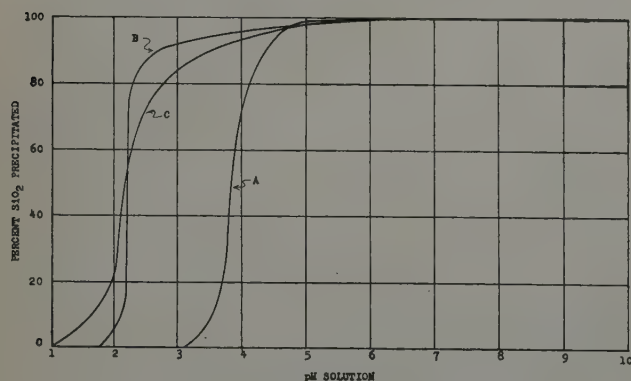


Fig. 2—Precipitation of silica vs. pH.

ditions a pH of over 9.5 is necessary for complete precipitation. When the solution is heated, iron is completely removed at a pH of 7.6 (curve B), and when both heat and air are employed, the desired result is obtained at a pH of 5.2 (curve C). Precipitation seems to be somewhat greater in the lower pH range without aeration, but above pH 4, the air rapidly increases the degree of completion. It is apparent that heating and aeration aid in the more rapid removal of iron from the solution. It is of interest to note that all of the curves are displaced to the right with respect to theoretical curves. This is probably due to the excess  $\text{SO}_2$  and other constituents in solution that tend to act as a buffer to repress the precipitation.

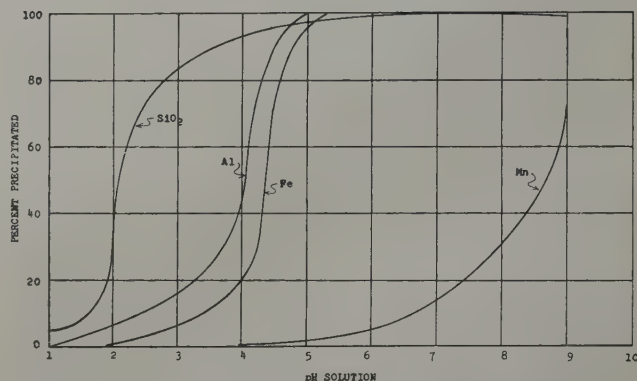


Fig. 3—Percentage of precipitation of  $\text{Fe}_2\text{O}_3$ ,  $\text{Al}_2\text{O}_3$ ,  $\text{SiO}_2$ , and manganese vs. pH.

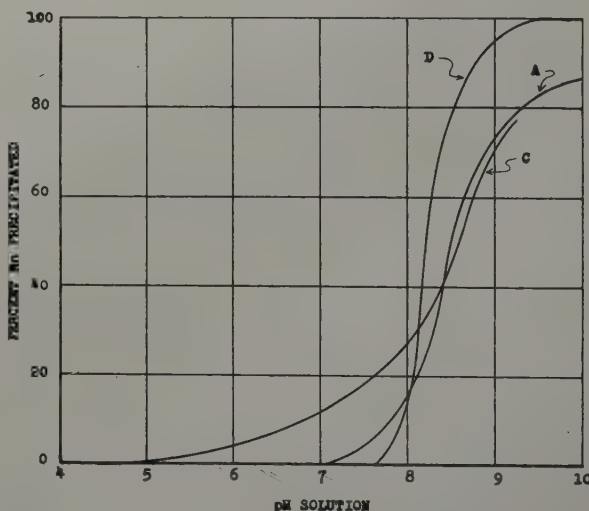


Fig. 4—Precipitation of manganese with ammonia at pH 4 to 10.

Fig. 2 illustrates the precipitation of silica. In the heated runs (B and C), 80 pct of the silica is removed at a pH of about 2.5, and in the room temperature run (A) at a pH of about 4. However, the curves reach 98 pct precipitation at pH of 5.0, and in none of them is precipitation quantitative until about pH of 7.0.

Since heating and aeration are essential, primary consideration should be given to the results which show the effect of these factors. Fig. 3 gives a plot of the percentage of  $\text{Fe}_2\text{O}_3$ ,  $\text{Al}_2\text{O}_3$ , and  $\text{SiO}_2$  precipitated, and manganese versus pH for run C (heat and aeration). On increasing the pH,  $\text{SiO}_2$  precipitates first, followed by alumina, iron, and finally at a much higher pH, manganese. The action of silica



and alumina are both surprising in that they precipitate before the iron. The results show that essentially all of the impurities (iron, alumina, silica) are removed from solution at a pH of 5.2, while approximately 2 pct of the manganese is lost.

**Precipitation of Manganese Oxides:** The precipitation of manganese with ammonia at various pH's is shown in fig. 4. Run A was made at room temperature and without air oxidation. Run B, made with the addition of heat alone, gave essentially the same result indicated by curve A. Run C shows that aeration causes Mn to precipitate at a lower pH.

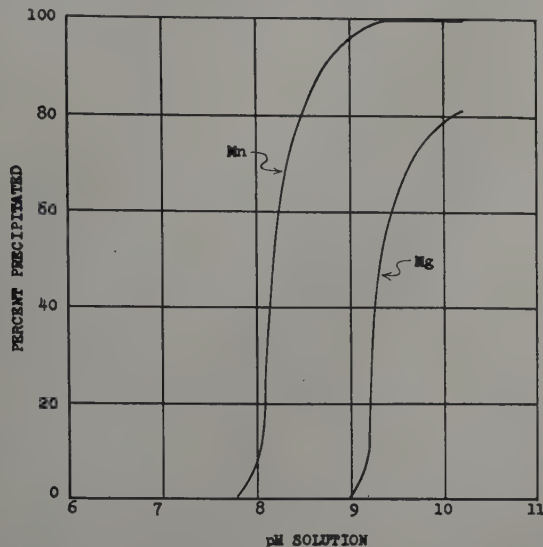


Fig. 5—Precipitation of manganese and magnesium vs. pH.

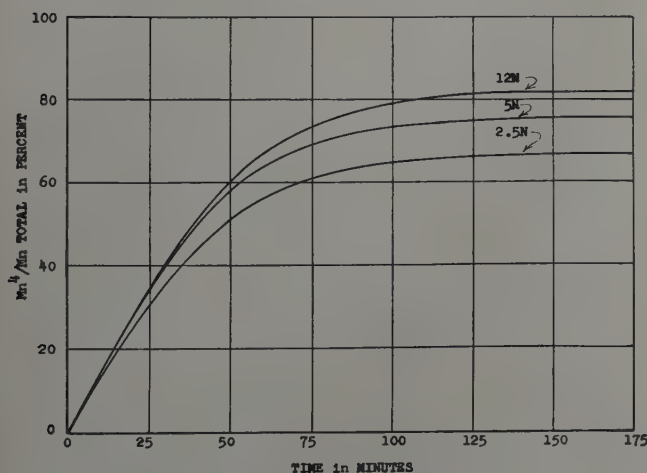


Fig. 6—Effect of ammonia concentration on oxidation of manganese.

These curves at atmospheric pressure are similar at the higher pH values. Complete precipitation of Mn without pressure requires a pH above 10.

Run D, at 40 psi air pressure, indicates that complete precipitation of Mn can be attained at pH 9.4, and 95 pct at pH 8.98. This run also indicates that below a pH of 8.98, no magnesium is precipitated (fig. 5). When the pH of the solution reached 9.4, 59 pct of the magnesium was removed. It is evident that precipitation conditions must be maintained so that the pH of the solution does not rise above 9.0, at which point 95 pct of the manganese is precipitated without any magnesium contamination. This care must be exercised only when manganese ore high in soluble magnesium is treated. For the pur-

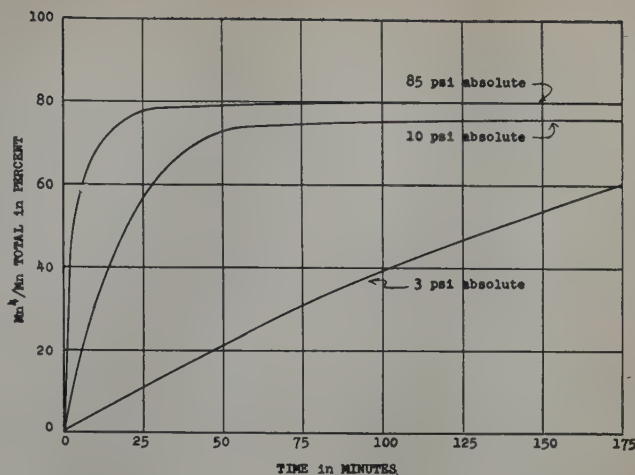


Fig. 7—Effect of oxygen pressure on oxidation of manganese.

pose of these precipitation tests magnesium sulphate was added to the solution at the rate of 2 g of magnesium per liter in order to complete the information regarding the removal of impurities from the solution. However, all the ores tested were low enough in magnesium so that such careful control of magnesium contamination was not necessary.

After removal of impurities, the solution is essentially manganese sulphate with some ammonium sulphate (formed from ammonia precipitation of impurities). Under proper control a manganese oxide product, in which 70 to 80 pct of the manganese is in the dioxide form,<sup>8</sup> will precipitate from the manganese sulphate solution.

Fig. 6 illustrates the effect of ammonia concentration on the precipitation. The data show that for a given temperature and pressure, the degree of oxidation increases with an increase in ammonia concentration.

Fig. 7 shows that, for a given temperature and ammonia concentration, the degree of oxidation is a function of oxygen pressure, increasing with an increase in pressure.

Fig. 8 demonstrates that for a given oxygen pressure and ammonia concentration, the degree of oxidation varies inversely as the temperature.

In all of these tests an optimum of 80 pct oxidation is reached. There is evidently an equilibrium established between the tetravalent and divalent manganese. All runs were continued until complete

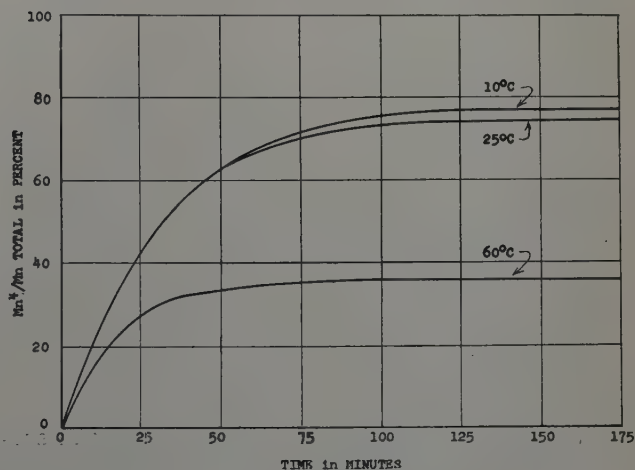
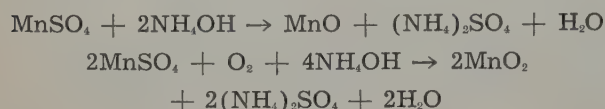


Fig. 8—Effect of temperature on oxidation of manganese.

precipitation of the manganese in solution was reached. From these curves it can be concluded that optimum conditions are reached when: (1) the ammonia concentration is above 3N, (2) the partial pressure of oxygen is above 10 psi absolute, and (3) the temperature is 25°C, or less.

The following reactions take place in this step of the process:



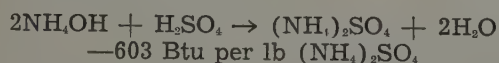
The equations show that an excess of ammonia must be used to insure complete precipitation of the manganese. The following analysis is typical for the product of this process:

Component	Per Cent
MnO <sub>2</sub>	77.9
MnO	11.5
SiO <sub>2</sub>	0.2
R <sub>2</sub> O <sub>3</sub>	0.1
BaO	0.0
CaO	0.01
MgO	0.1
SO <sub>3</sub>	0.0
Water of hydration	10.2 (approx.)

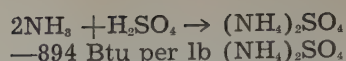
The original intention was to recover the excess ammonia by steam stripping the solution after removal of the manganese precipitate, and recovering the combined ammonia from the (NH<sub>4</sub>)<sub>2</sub>SO<sub>4</sub> by the addition of lime, liberating ammonia and leaving byproduct gypsum.

**Ammonium Sulphate Production:** The present plan, however, is to recover the ammonium sulphate. When the manganese ore is leached to a concentration in solution of 100 g Mn per liter, the equivalent ammonium sulphate formed is 24.7 lb per 100 lb of water. This is neglecting the ammonium sulphate formed from the precipitation of the impurities of iron, alumina, and silica. Ammonium sulphate can be recovered several ways: (1) addition of outside heat to evaporate the water after the excess ammonia has been stripped from the solution, (2) neutralization of the excess ammonia with sulphuric acid and addition of outside heat to evaporate the water, or (3) addition of enough sulphuric acid and ammonia and taking advantage of their heat of reaction to affect crystallization of ammonium sulphate.

**Thermodynamic Considerations:** In determining which method of ammonium sulphate production should be employed, markets and availability of raw materials as well as heat of reaction between ammonia and sulphuric acid must be considered. After the manganese oxides have been removed from solution, the remaining ammonium sulphate solution contains an excess of ammonium hydroxide. The addition of enough sulphuric acid to neutralize this excess ammonium hydroxide will liberate 603 Btu per lb of ammonium sulphate formed as represented by the following equation:



The addition of more anhydrous ammonia and sulphuric acid will liberate 894 Btu per lb of ammonium sulphate produced according to the following equation:



Using 1000 Btu as the theoretical heat required to evaporate 1 lb of water and an evaporating efficiency of 80 pct, the material balance on the ammonium sulphate produced if the original solution is taken to dryness is:

Ammonium Sulphate Balance	
Ammonium sulphate formed from	Pct of Total
Mn precipitation	14.3
Neutralizing excess NH <sub>3</sub> in solution	13.5
	27.8
Additional NH <sub>3</sub> and H <sub>2</sub> SO <sub>4</sub> added	72.2
	100.0

From the above it is seen that 27.8 pct of the ammonium sulphate is formed from the Mn precipitation. Other factors being equal, this percentage is also the approximate economic advantage of this process over that required in the direct production of ammonium sulphate from ammonia and sulphuric acid.

Taking a basis of 100 tons of 30 pct Mn ore per day and assuming 90 pct recovery of Mn, 451.8 tons is the theoretical ammonium sulphate production if all the water is evaporated.

Primarily because of the shortage of sulphuric acid at the beginning of operations, the excess ammonia will be stripped from the solution, and the water will be evaporated to crystallize ammonium sulphate.

**Production Costs:** In any process in which two products are produced, that portion of the costs which can not be placed definitely on one product can be divided in a number of ways. The present estimates are based on the treatment of 90,000 tons of Mn ore a year (27 pct Mn) and 90 pct recovery. This will give an annual production of 37,700 tons of manganese oxides containing 58 pct Mn and 50,700 tons of ammonium sulphate.

With the cost breakdown used, the manganese oxide production cost is \$0.358 per long ton unit of Mn and the ammonium sulphate is \$43.57 per ton. This manganese oxide cost does not include mining or freight on the ore. However, both costs do include amortization of an \$800,000 investment in four years. The raw material costs used are based on \$23.75 per ton for sulphur and \$110.00 per ton for anhydrous ammonia. Also the ore for treatment is the type found in Idaho, Nevada, and Montana which does not require sulphuric acid. The ammonium sulphate cost is based on single-effect evaporation which is about 18 pct of the ammonium sulphate cost. Needless to say another saving would be made by multiple-effect evaporation.

Thus starting with low-grade, intermediate, hydrated siliceous manganese ore, found in great abundance in the United States, an economical ammonium sulphate as well as a high-grade manganese oxide can be produced.

## References

- <sup>1</sup> C. W. Davis: Dissolution of Various Manganese Minerals. U. S. Bur. Mines R. I. 3024 (July 1930) p. 7.
- <sup>2</sup> John R. Allen: Process of Treating Intermediate Manganese Siliceous Ores. U. S. Pat. 2,451,647. (Oct. 19, 1948).
- <sup>3</sup> R. W. Moulton: Preparation of Oxide of Manganese. U. S. Pat. 2,459,714. (Jan. 18, 1949).



## Concrete Aggregates for Large Structures

by Elliot P. Rexford

The quality of aggregate materials is of major importance in governing durability and permanence of concrete structures. The problem of selecting suitable aggregate materials is two-fold. Geological factors include the type and character of the aggregate source, presence of known or suspected reactive constituents, and presence of thermally incompatible materials. Laboratory investigations include petrographic studies, soundness and reactivity tests, and tests of thermal properties. The importance of each factor in the selection of satisfactory aggregate material is considered in detail.

IN the early days of concrete construction, the question of quality of sand or gravel to be used as aggregate in concrete was given little or no consideration. If the construction engineer had sufficient quantity for his structure within a reasonable distance, he was contented and gave no thought to the possibility that the material might have adverse physical, chemical, or thermal properties which could cause future deterioration of his concrete and possible failure of his structure.

ELLIOT P. REXFORD, deceased, was Chief, Petrography Section, South Pacific Division Testing Laboratory, Corps of Engineers, Department of Army, Los Angeles, Calif.

AIME Los Angeles Meeting, October 1948.

TP 2837 H. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Dec. 7, 1948.

Today a very different situation exists. With the building of larger and larger concrete structures, with a longer and longer life expectancy, the quality of the aggregate material has become of major importance. Even in small concrete structures aggregate quality cannot be passed off lightly and the quality of the aggregate must be such as to give the anticipated life without costly maintenance.

The first question to arise in the selection of an aggregate is what makes a sand or gravel suitable or unsuitable for use in concrete, and how can its quality be judged. No one rock type can be depended upon to furnish sound durable material in all instances. Granitic rocks, for example, make very satisfactory aggregates in some cases, but they are among the most unsuitable of materials when they are fractured, weathered, or highly micaceous. Limestones make excellent concrete aggregates on the one hand, and on the other they may be unsound and lack durability if they contain appreciable clay, they may have a harmful chemical reaction with the cement if they contain amorphous silica, or they may have thermal characteristics which make them highly unsuitable for use in concrete. Each rock, gravel, or sand must be proved

on its own merits for the particular job under consideration.

The selection of suitable aggregate materials may be divided into two major phases, the geological phase which is carried on largely in the field, and the testing phase which of course is a laboratory problem; both are important. Very often the geological phase alone is sufficient for the selection of satisfactory aggregates for minor structures but for major structures both phases must be thoroughly investigated.

### Geological Phase

In the southwest part of the country nearly every major rock type, igneous, metamorphic, and sedimentary, may be found with all imaginable intermediate phases and in all imaginable stages of alteration. Frequently very different rock types, different as to geological age, composition, and condition occur very close together. Alluvium and stream-bed sands and gravels will consist of many different rock types and even ledge rocks may vary radically as to texture and composition within the area of a quarry site. Some of the rocks make suitable aggregate materials whereas others are highly deleterious in concrete. Too often the good and the bad occur together and more than ordinary caution is required in aggregate selection.

**Aggregate Sources:** Concrete aggregates are usually obtained from stream-bed sands and gravels or from ledge rocks. Each has certain favorable and unfavorable characteristics and it is sometimes necessary to investigate both types to determine the most suitable source for a specific structure.

**Stream Beds:** The predominant sources of aggregate materials in southern California and Arizona are stream beds. Vast quantities of sands and gravels are brought from the mountain ranges by the rivers during times of flood but most of these streams are dry a large part of the time and hence these deposits make easily accessible sources of aggregate materials. From many standpoints stream-bed materials make very acceptable aggregates. The gravel pebbles and the sand grains are

frequently rounded and of satisfactory shape for a good, workable concrete mix. Often coarse and fine aggregates can be obtained together with no more complicated processing than washing and screening. "Quarrying" or excavation operations are relatively simple and the aggregates often can be obtained close to the construction site. On the other hand, stream-bed materials are derived from the various rocks of the entire drainage area of the stream in which the deposit is located. Danger of contamination by unsound and chemically reactive rocks is very high if such occur anywhere within the drainage basin. Stream-bed deposits may change in character with depth. Sometimes rock types, not observed on the surface, occur below the surface, and vice versa. Deposits having comparatively sound and durable material on the surface may contain an excessive amount of severely decomposed material only a few feet beneath it. Therefore, constant guard must be maintained to detect indications of those deleterious features in considering a stream bed as a source of concrete aggregates.

**Ledge Rocks:** If suitable stream-bed deposits are not available, it may be necessary to consider quarrying and processing the available ledge rocks in the vicinity of a proposed structure. Ledge rocks usually are comparatively homogeneous throughout. There is not the danger encountered in stream-bed deposits of unexpected contamination by reactive constituents or unsound weathered particles, once the character of the ledge rock is established. Ledge rocks, however, have the disadvantage of requiring relatively expensive quarrying and processing operations since crushed, unprocessed ledge rocks usually make comparatively harsh aggregate materials.

**Soundness:** The degree of soundness of an aggregate material is a measure of its potential durability. Unsound rock fragments, gravel pebbles, or sand grains will cause ultimate deterioration of the concrete, sometimes within a very short time. Aggregates with soft weathered surfaces or coated with soft clayey substances will have poor bonding properties in concrete and will contribute to low strength. Severely weathered, soft, micaceous or porous materials may cause localized stresses to develop in the concrete by swelling and shrinking during wetting and drying and/or freezing and thawing cycles. The ultimate result is a cracked and deteriorating structure. In the southwest where climatic conditions are not generally severe, a concrete structure made with unsound aggregate may survive somewhat longer than in another part of the country where the climate is more rigorous but it is destined to ultimate deterioration. Unsound aggregates are numerous in this vicinity and if used promiscuously, trouble may be anticipated.

Some of the rock types which may be expected to be unsound from the durability standpoint are soft, weathered, severely fractured, or highly micaceous igneous rocks, micaceous schists, soft shales, loosely bonded sandstones, and porous clayey limestones. On the other hand, hard, fresh unaltered igneous rocks (granites, diorites, diabases and gabbros), hard dense limestones, and unweathered or unfractured metamorphic rocks will generally make durable aggregate materials. Usually the potential durability or soundness of an aggregate can be judged by a consideration of the following elementary features: (1) Are the rock types dense,

hard, and fresh, or porous, soft and weathered? (2) Are they fractured or severely jointed? (3) Are they coarse or fine-grained? (Generally speaking a fine-grained rock will make a more durable aggregate than a coarse-grained). (4) Do the rock types show evidence of hydrothermal or other alteration which may cause deleterious physical characteristics? (5) Do they contain a high content of mica? (6) Do they contain chemically reactive constituents which will effect an unsound condition later? (7) Are there other features such as schistosity, bedding or banding, stylolites in limestones, and so on which may influence the materials as an aggregate?

Most of these features can be observed readily in the field and on the presence or absence of such factors it can be decided from the soundness standpoint as to whether or not the material merits further consideration by laboratory tests.

**Reactivity:** Chemical reactivity between aggregates and certain constituents in a normal portland cement has been of increasing concern to concrete engineers for the past ten years. It is generally spoken of as alkali-aggregate reactivity since it has been found that the reactive constituents in the cement are the alkalies, sodium and potassium. These elements occur in only small amounts, from about 0.4 to 1.3 pct in all portland cements, but they are sufficient in these small amounts to cause a very undesirable condition in concrete with the wrong type of aggregate. Alkali-aggregate reactivity is known now to be one of the major causes of concrete deterioration in this, and several other parts of the country. Generally speaking this type of concrete deterioration is not well known, and its seriousness is not well understood, by many construction engineers so the subject is discussed here.

In 1940, T. E. Stanton, of the California Dept. of Highways, published an account of concrete failures along the coast of Southern California which were due to excessive expansion.<sup>1</sup> He found that the excessive expansion in all cases was due to a chemical reaction between the sodium and potassium in the cement and a certain mineral constituent, amorphous silica, which occurred in some of the concrete aggregates. A few months later, R. F. Blanks, of the Bureau of Reclamation, published an account of the concrete deterioration which was occurring in the then recently completed Parker Dam on the Colorado River,<sup>2</sup> and H. S. Meissner published a more detailed account of the Parker Dam condition in 1941.<sup>3</sup> Like the structures described by Mr. Stanton, the deterioration of the dam was found to be caused by a reactive constituent, a volcanic rock, in an otherwise sound aggregate. Since that time, many other concrete structures have been found to be suffering from the results of alkali-aggregate reactivity, and a tremendous amount of study and research has been performed on the subject by various federal, state, and private agencies. The results of most of these studies, together with accounts of many of the more recently discovered distressed structures, have been published in detail in engineering journals such as the issues of the American Concrete Institute, the Proceedings of the American Society for Testing Materials, and others. Therefore, anything more than a summary of alkali-aggregate reactivity as a whole, is beyond the scope of this discussion.

Probably the first question to arise in considering



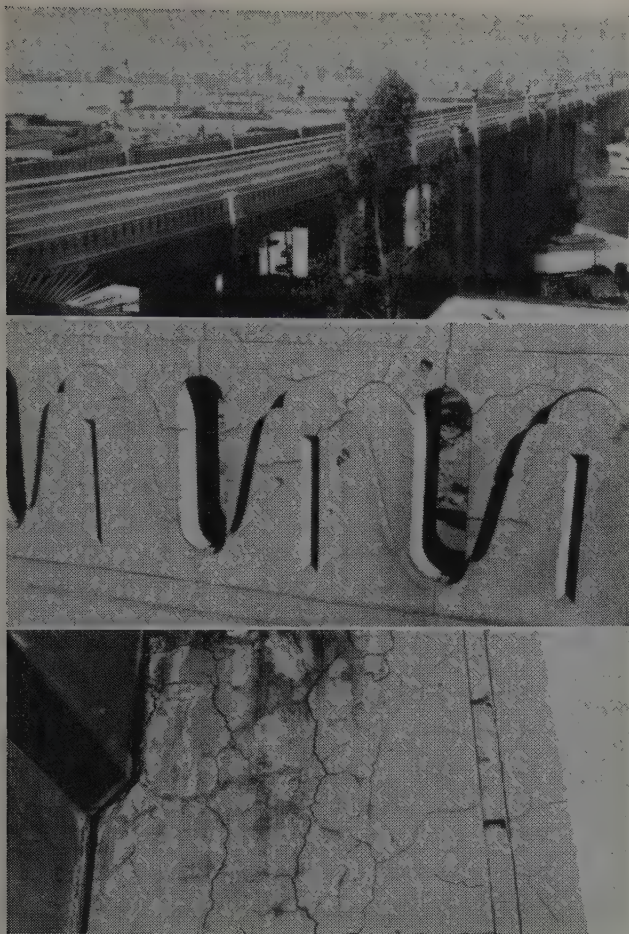


Fig. 1—Sixth Street Bridge, Los Angeles, Calif.

a. General view of bridge. b. Cracking and popouts in railing. c. Cracking in pier of bridge.

alkali-aggregate reactivity, is what constitutes a reactive aggregate? A reactive aggregate is any rock, gravel, or sand which contains one or more constituents that react chemically with the alkalis (sodium and potassium) in a normal portland cement and causes expansion of the concrete. The specific rock types found to be the most reactive are those which contain either amorphous silica (sometimes called opaline silica), or glass. Amorphous silica may occur in shales, limestones, cherts and chalcedonic fragments, while glass is frequently found in volcanic rocks. Usually the most reactive of the volcanic rocks are the more acid types, the rhyolites and andesites, but often the more basic basalts also show severe reactive tendencies.

The seriousness of distress caused by these constituents is influenced to a large degree by the amounts in which they are present in the fine and coarse aggregates. In most cases, *small* amounts of from 1 to 5 pct of reactive ingredients, in an otherwise nonreactive sand, will cause far more serious results than larger amounts. For example, many of the sands and gravels of Southern California and Arizona consist predominantly of granitic fragments with quartz and feldspar. These constituents are generally sound and entirely nonreactive. However, mixed with many of the materials is from  $\frac{1}{2}$  to 6 pct of opal-bearing cherts, shales and limestones, or glass-bearing rhyolites, andesites or basalts. These relatively small amounts of reactive constituents have been sufficient to cause serious reactive deterioration in road pavements, bridges, dams, and other concrete structures. In the Los Angeles area small amounts of opal-bearing shale

have caused severe distress in the Sixth Street bridge over the Los Angeles River channel (fig. 1), while small amounts of volcanic particles are the cause of the well-known deterioration in the Parker, Gene Wash, Copper Basin, Coolidge, and Stewart Mountain dams in Arizona (figs. 2 and 3).

**Symptoms of Reactivity:** Deterioration of a concrete structure caused by alkali-aggregate reactions is made evident by several features. Popouts, pattern cracking, exudations, and expansion or swelling of the concrete are the predominant symptoms of this type of distress.

1. Popouts are conical-shaped spalls which are slowly pushed out of a concrete surface by the force of localized reaction. They may range in size from less than one inch to more than a foot in diameter. Usually they are caused by a single pebble which will be found at the apex of the cone or at the base of the socket in the parent concrete. When fresh and moist the pebble often will be soft and gel-like in texture but it will harden rapidly on exposure to the air. A rather severe popout condition has been found in the spillway and ogee structure of Sepulveda Dam near Los Angeles which has been caused by occasional fragments of amorphous silica-bearing shale (fig. 4).

2. Pattern cracks are another symptom of reactivity. Cracks of this nature may be caused by reactive constituents in either the fine or the coarse aggregate, or both. The cracks, once started, usually become longer, wider, deeper, and more numerous as time goes on until serious disintegration of the concrete has resulted. Pattern cracking may be caused by factors other than reactive aggregates,

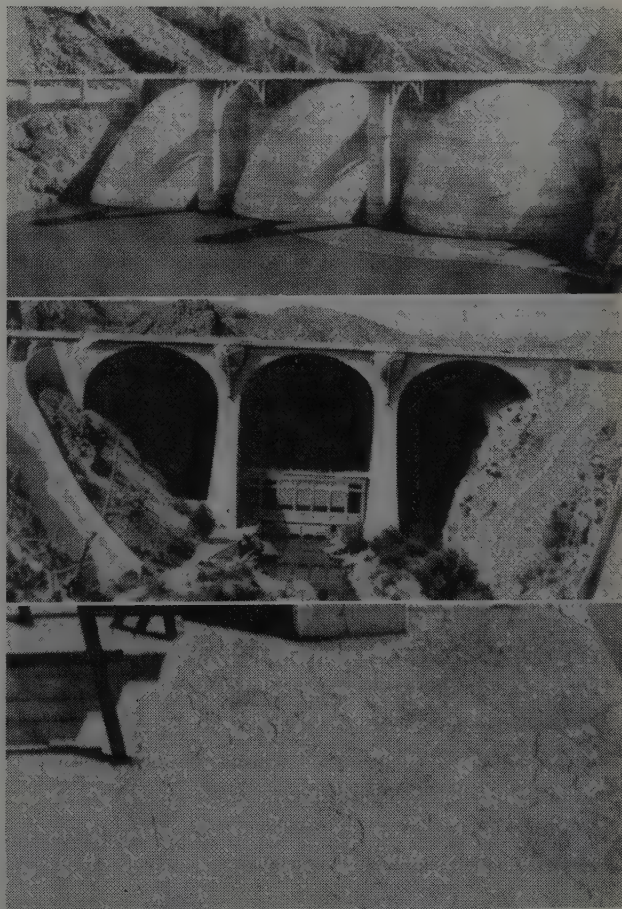
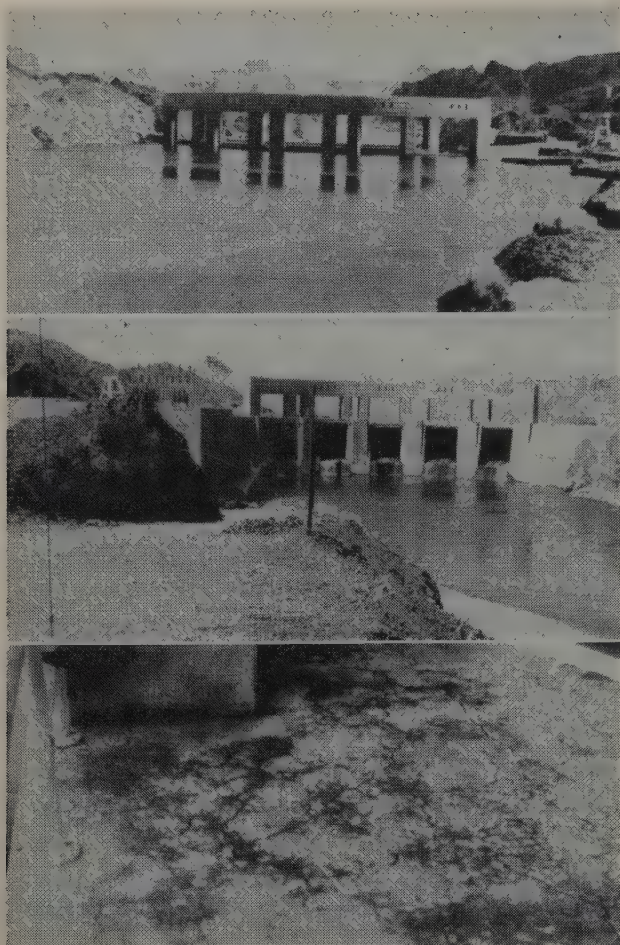


Fig. 2—Coolidge Dam, Gila River, Ariz.

a. Upstream face. b. Downstream face. c. Severe cracking in walkway on top of dam.





**Fig. 3—Parker Dam, Colorado River.**

a. Upstream face. b. Downstream face. c. Severe cracking on crest of dam.

and caution is needed when diagnosing such cracking as being due to the reactivity. Poor workmanship, overfinishing, too wet a batch, and incompatibility due to thermal properties of the aggregates will also cause this type of cracking. In the field, it is often impossible to decide which factor was the primary cause. On the other hand, a careful study of polished sections or thin sections of cores from the distressed concrete will usually indicate the probable cause. Under the microscope the thickness of a finished surface and its relation to the pattern cracks can be observed. The relation of the coarse and fine aggregate to the cracks, the presence of reactive grains and pebbles, reaction rims around aggregate particles, and gel-like reaction products can all be studied.

3. Exudations are occasionally found when the concrete is subjected to moist conditions. These exudations occur in small blobs of a jelly-like substance which exude from the concrete. When fresh and moist, it is very soft but on exposure to air the substance hardens, changes from colorless to white or brown, and generally carbonates rapidly. Chemical analyses of the substance have shown it to be a silica, or silicate gel, in which sodium, potassium, and other ions are contained. In general, the substance has a high-silica content, up to 80 pct in some cases, while the total sodium and potassium content may vary from almost negligible to as high as 26 pct or more. The presence of exudations, or gel-like rims around an aggregate particle in concrete is irrefutable evidence of alkali-aggregate reactivity (fig. 5).

4. Popouts, pattern-cracking, and exudations are surface manifestations of reactivity. Deep down in a mass of concrete, such as within a large concrete dam, reaction takes another and probably a more serious form. Here the concrete expands or swells and sometimes the hydrated cement paste loses its life and becomes chalky. Expansion is not confined to localized areas but the entire structure may grow. Arch dams may bow upstream a matter of several inches, power installations and gate structures are thrown out of alignment and long, deep cracks may occur from the stresses developed when the concrete is confined.

*Chemistry of Reactivity:* The actual chemical reactions which occur between reactive aggregates and the cement alkalis in forming a gel substance are not clearly understood. Several hypotheses have been suggested, but to date none has been actually proved.

One theory is that the sodium and potassium react directly with the unstable amorphous silica or volcanic glass to form a simple alkali-aggregate gel. Under this theory, once the alkalis have reacted, and have been used up in the reaction, no further distress should occur. Tending to substantiate this idea is the fact that 100 pct of highly reactive material in concrete laboratory specimens has been known to cause little or no expansion with a high-alkali cement, possibly indicating that the alkalis were used up before the concrete became rigid.

A second hypothesis suggests that the gel is the final product of a series of chain reactions and that somewhere along the series of reactions the sodium and potassium are released to react again with any available reactive aggregates. Tending to substantiate this idea is the length of time in which reaction goes on in a structure. Once started it appears to continue indefinitely, although at a retarded rate, suggesting a continuous source of alkali.

A third theory maintains that the sodium and potassium act as catalyzers to start the reaction and, after starting, it is carried on by the normal calcium hydroxide of the cement paste.

A fourth theory is that the reaction once started, may involve the sodium and potassium-bearing aggregate minerals, possibly the feldspars, and thus derive a fresh supply of alkali from these otherwise nonreactive constituents. There is very little petrographic evidence to substantiate this theory, however. Feldspars and other minerals are attacked to a very limited extent by some portland cement hydrates but generally such an attack is no more than a surface phenomenon.

From these hypotheses, it can be seen that the chemistry of alkali-aggregate reactivity is still in a confused state and more work is needed on this phase of the subject.

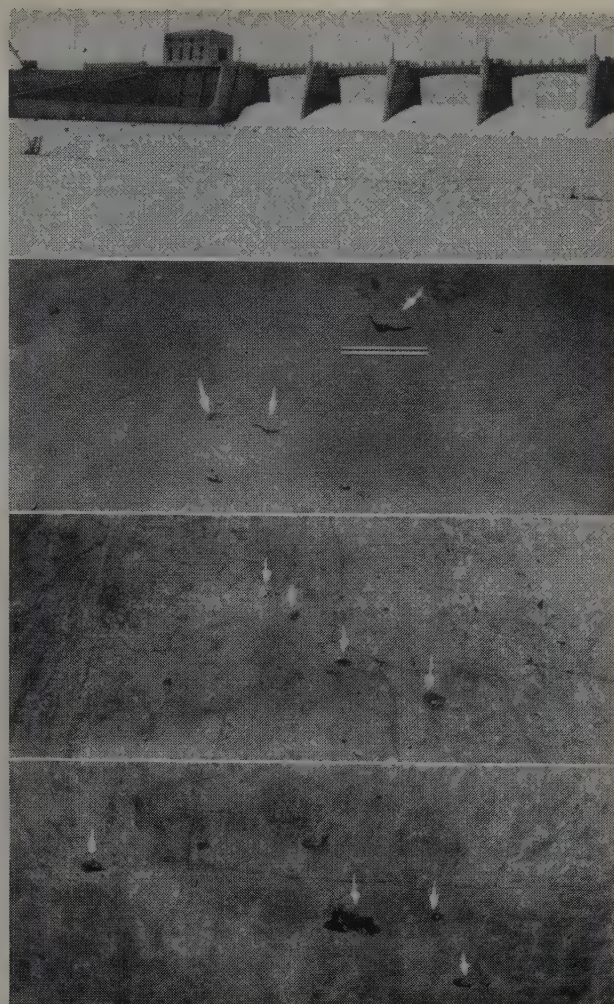
*Mechanics of Reactivity:* The mechanics of the reactions, like the chemistry is not well known but a few hypotheses have been advanced of which that suggested by W. C. Hansen,<sup>4</sup> of the Portland Cement Association, appears to be the most reasonable. Silica can react with alkali hydroxides only in the presence of liquid water and when this reaction takes place, a relatively soluble alkali silicate is formed. Because of this solubility, the alkali silicate, theoretically, should diffuse throughout the cement paste of a concrete and cause no expansion, particularly since the alkali silicate is known to occupy less volume than that required by the silica and the alkali hydroxide. Actually, however, sili-



cate ions in solutions of alkali silicates tend to associate into complex ions which, experiments have indicated, may not be capable of diffusing through the hardened cement paste. The cement paste then acts as a semi-permeable membrane which will permit the passage of water and the alkali hydroxides but prevents the passage of the complex silicate ions. This results in the development of an osmotic pressure cell. Under such conditions, the alkali silicates formed on the surface of a reactive-aggregate pebble tend to draw water from the cement paste and form a reservoir of liquid which exerts a hydrostatic pressure against the paste. The water also brings additional alkali hydroxide into contact with the reactive pebble and additional alkali is formed. Since this alkali silicate cannot diffuse through the cement-paste membrane, it must occupy the same space as the original silica of the reactive pebble and, since the volume of the alkali silicate is greater than that of the silica from which it formed, pressure is exerted on the cement paste which is augmented by the hydrostatic pressure. Since this condition continues as long as alkali hydroxides and silica are available, tremendous pressures may result and rupture of the concrete follows. Although theoretically, crystalline silica (quartz) probably follows the same series of reactions as amorphous (opaline) silica, the rate of reaction of quartz with alkali hydroxides is so very much slower than that of amorphous silica that it is negligible in concrete.

**Prevention of Reactivity:** Two preventive measures for minimizing or inhibiting the deleterious activity of a moderately reactive aggregate have been suggested, both of which have been used by the Bureau of Reclamation in some recent structures.

The first, is the use of low-alkali cement. Mr. Stanton, in his original paper, suggested that reactive aggregates probably would not cause excessive expansion if used with cements having an alkali content of less than 0.6 pct. Expansion tests on laboratory mortar bars made with this type of cement and known reactive sands have shown low expansion results in most cases. It is, therefore, thought that this may be indicative of the expansion that may be expected in service. However, it is still not known whether a so-called low-alkali cement will actually prevent reactivity for all time, or will simply slow it down for perhaps a few years only to have it develop and become evident later. In this regard W. F. Abercrombia,<sup>6</sup> Engineer of Materials and Tests, State Highway Department, Georgia, has reported that 25 miles of road pavement in Georgia, built in 1929 with cements believed to contain 0.3 to 0.48 pct alkalis is now severely deteriorated and filled with gel-relics, and A. D. Conrow,<sup>6</sup> Research Engineer, Ash Grove Lime and Portland Cement Co., Chanute, Kansas, has reported having developed excessive expansion in laboratory tests with reactive aggregates when used with a cement having as low as 0.09 pct total alkali. It is his belief that the reaction once started by the sodium and potassium may be carried on by the normal calcium hydroxide of the hydrated cement. W. C. Hanna,<sup>7</sup> Chief Chemist, California Portland Cement Co., Colton, Calif., has shown the results of some 4-year tests of reactive aggregates with both low and high-alkali cements. His tests showed that of about 100 specimens tested the low-alkali cement bars had as much if not more expansion than the



**Fig. 4—Sepulveda Dam, Los Angeles, Calif.**

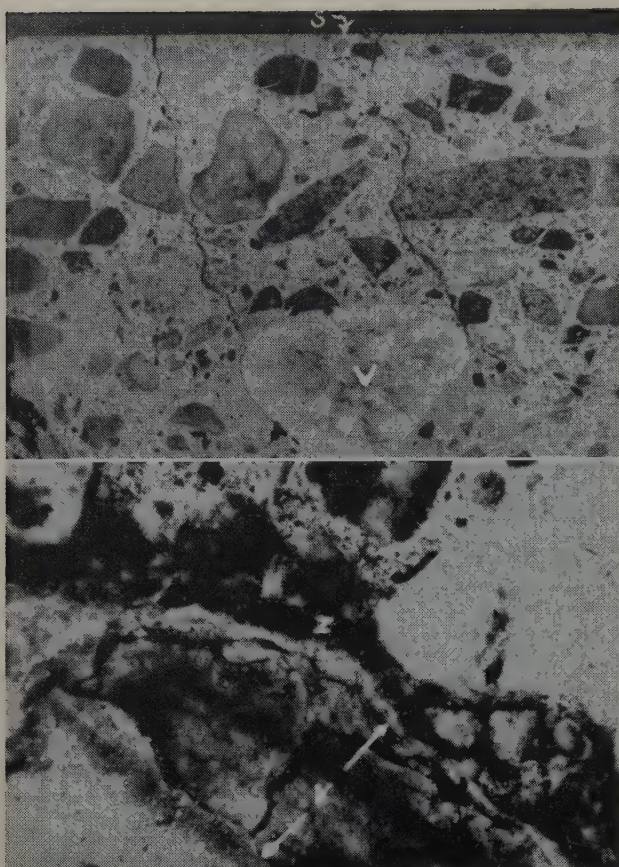
a. Portion of spillway slab, ogee, and gate structure. b. Popouts in place in spillway slab, scale is 6 in. c. Popouts removed. Note white pebbles in cavities. d. Popouts and fine pattern cracking.

high-alkali cement bars with reactive aggregates.

The second preventive measure is the addition to the concrete batch of an admixture of finely-divided (325 mesh), highly reactive material such as diatomaceous earth or volcanic ash. Materials of this nature have been found to inhibit or delay the activity of a reactive aggregate by, theoretically, using up or neutralizing the alkalis in the cement, before the concrete becomes rigid. W. C. Hanna,<sup>7</sup> has shown that finely divided calcined Monterey shale from near Los Angeles which contains diatomaceous earth (amorphous silica) was highly beneficial in retarding expansion when reactive materials were used in laboratory tests. However, again it is not known for certain whether or not the reaction is permanently prevented or only temporarily delayed. This point might be cleared up when the chemistry of the reaction is more clearly understood.

**Incompatibility:** Of even more recent development than the subject of alkali-aggregate reactivity, is the subject of thermal incompatibility between fine-aggregate, coarse-aggregate, and hydrated cement paste in a concrete structure. In 1941 J. A. Pearson<sup>8</sup> wrote on his subject, and in 1944 W. H. Johnson and W. H. Parsons<sup>9</sup> found, among other things, that the thermal coefficients of expansion of various rock types used commonly for aggregate materials varied widely. For example,





**Fig. 5—Effects of alkali-aggregate reactivity in concrete.**

a. Split 6-in. core. Shows cracks originating at volcanic pebble (v) passing to concrete surface (s). b. Micrograph showing dried gel reaction rim (y) between reactive pebble (lower left corner) and concrete mortar (z). Magnification 40x. (Reduced approximately one quarter in reproduction.)

the thermal coefficients of expansion of granites ranged between 1 and  $5 \times 10^{-6}$  in. per  $^{\circ}\text{F}$ , limestones ranged between 0.5 and  $5.1 \times 10^{-6}$  whereas hardened portland cement paste varied between 6 and  $9 \times 10^{-6}$ . Of further interest they noted that the mineral constituents of these rocks commonly found as the predominant constituents of fine aggregates, varied greatly along different axes. Quartz ranged between 4.1 and  $7.5 \times 10^{-6}$ , calcite ranged between -2.8 and  $14.3 \times 10^{-6}$ , and microcline feldspar was 0.6 and  $9.5 \times 10^{-6}$  for different axes. Still further, in 1947, Albert Weiner<sup>10</sup> showed that the thermal coefficient of diffusivity of quartz was nearly four times that of a cement mortar made with a siliceous sand. There are extreme ranges of difference in thermal characteristics and it can readily be seen that thermal compatibility between the various concrete constituents must be maintained if the concrete structure is to survive unharmed. A concrete structure is bound to crack and deteriorate with internal stresses if it contains ingredients which are expanding and contracting at difference rates and to different degrees of intensity.

**Incompatibility in Nature:** Probably the most spectacular examples of the results of thermal incompatibility are shown in nature. The manner in which granitic rocks disintegrate on the slopes of the desert mountains is rather strong evidence of the results of mineral incompatibility. Here the granite, consisting largely of quartz and feldspar, is soft, friable, and often completely disintegrated for from several inches to many feet below the surface. Wetting and drying, and freezing and

thawing are negligible factors here, whereas heating and cooling conditions are extreme. Petrographic examinations of this material seldom show extreme evidence of chemical decomposition, that is, alterations of the feldspars to clay minerals or the iron-magnesium minerals to chlorite and so on. Generally speaking, the mineral constituents are fresh in appearance except for one feature, they are shattered and filled with incipient, tiny microscopic cracks, even in the finer sieve sizes. A light touch with a needle point will usually cause many of the sand grains, quartz, feldspar, or granitic particles to crumble, while large fragments of the granite will crumble in the hands. If we go back again to the findings of Parsons and Johnson, Weiner, and others, we have what seems to be ample proof for concluding that the disintegration of the granite, and the shattered condition of the crystals, were due to differences in thermal properties. In these areas tightly locked crystals of quartz, feldspar, and other minerals, each having widely different characteristics in expansivity and diffusivity in a rigid granite mass, must cause severe localized stresses to develop under excessive temperature changes. These can be relieved only by shattering of the mass and its individual components. If this happens in nature, it can also happen when similar materials are confined as aggregate in concrete.

**Incompatibility in Concrete:** In concrete, deterioration due to thermal incompatibility is generally indicated on the surface of a structure by a series of pattern-cracks similar to those found under reactive-aggregate and other conditions. The cracks, however, are generally finer and the pattern of closer texture than in the cracking due to other causes. Cracking of this nature sometimes shows up rather drastically in the desert regions, particularly if the concrete surface is wetted, but actual destruction of the desert concrete by this means alone probably does not go much beyond the pattern-cracked stage. In more rigorous climates, however, where rains and freezes are more frequent, the condition can become very serious. The fine cracks are a means for ingress of moisture and rapid deterioration will accompany any cycles of freezing and thawing.

Thus the problem of thermal incompatibility in concrete becomes very complex. The thermal characteristics of different cement hydrates are different, those of the various aggregates are all different, and even those of the various constituents in any individual aggregate pebble are different. Therefore, to insure thermal compatibility in a concrete, each major rock type from each aggregate source should be tested in the laboratory to determine its thermal characteristics, and materials having similar thermal properties must be used if future disintegration of the concrete by this means is to be prevented.

#### Laboratory Testing Phase

Today laboratory tests to determine the suitability of a material for concrete aggregates are numerous, and in some instances complicated and intricate, requiring expensive equipment and skilled personnel. Many of the tests such as absorption, specific gravity, compressive strength, rattler, color and sodium or magnesium sulphate tests have been used for many years and are probably familiar to anyone who has had occasion to do any concrete construction work. They are adequately described in the Standards of the American Society for Test-



ing Materials. Recently, however, outstanding tests, have been developed and are now being used by many laboratories for determining potential soundness, reactivity, and compatibility, which are not as well known and a brief description of these tests is believed warranted.

**Petrographic Examination:** For a complete petrographic examination of a sand, gravel or ledge rock, several types of microscopes are used ranging from a low-power stereoscope type to a high-power polarizing petrographic type. Sand grains are identified, and physical condition and quantitative estimates of composition are made by actual counts of hundreds of grains. Gravels and ledge rocks are studied in thin and powder sections under the petrographic-type microscope. The various mineral constituents are identified, and their degree of alteration is observed. Features such as incipient fracturing or incipient weathering, not visible to the naked eye, are apparent under the petrographic microscope and may indicate an unsoundness not suspected in an ordinary investigation. Chemically reactive constituents, not identified as such in the field, can be immediately detected under the microscope, even when they occur in very small amounts. The amount in which they are present can be estimated quite accurately by grain counts of a sand or by the measurements of lineal intercepts in a rock or gravel pebble. The identity of the suspected constituent can be determined and from this the susceptibility of the material to reaction can be judged. All of these features exert an influence on the ultimate suitability of the material for concrete aggregate.

**Soundness Tests:** With the building of larger permanent concrete structures it early became apparent that aggregates to be used in those structures must meet definite specifications for soundness and durability if the structures were to survive their anticipated life span. Tests were devised to this end among which was a slow freezing and thawing test in which the unknown aggregate was made into concrete beams and frozen and thawed at the rate of two to four cycles per day. This test soon proved to be too slow for practical purposes and an accelerated test was devised wherein the aggregate was soaked in a hot solution of sodium or magnesium sulphate, cooled, and then dried to allow crystal growth of the sodium or magnesium sulphate. Crystal growth in capillaries, fractures, or pores in sand grains or gravel pebbles caused disintegration of the material and it was judged sound or unsound by the percentage loss in weight of the aggregate after a specified number of cycles. This test is still used by many laboratories and does give information as to the potential durability of an aggregate but it also leaves much to be desired.

Very recently a modification of the old freezing and thawing test was developed<sup>11</sup> which is now used by all laboratories of the Corps of Engineers. It appears to be the best test for potential soundness and durability developed to date. As in the "slow" test, the aggregate is made into concrete specimens\*

\* Using a specially prepared cement containing an entraining agent.

generally 16x3x4 in. in size, but unlike the slow method the specimens are frozen and thawed automatically at the rate of twelve cycles per 24 hr. Periodically the specimens are removed from the freezer and tested electronically (sonic method) for loss in dynamic modulus of elasticity. The aggregate

is considered to have failed in soundness and durability, that is to be unsuitable for use in the concrete of a large structure, if the modulus of elasticity falls below 50 pct of the original modulus of elasticity of the specimen in less than 300 cycles of fast freezing and thawing. In all tests of this kind, 3 to 9 specimens of a material are tested and the results averaged.

**Reactivity Tests:** The only certain method of combating alkali-aggregate reactivity, at the present time, is to detect potentially reactive constituents, in a sand, gravel, or rock before they are incorporated in concrete, and thus prevent the use of contaminated sands, gravels, and rocks as aggregate materials. To this end, certain tests have been developed to establish the reactive nature of a selected material. These tests include petrographic examinations previously discussed, chemical tests with sodium hydroxide, and a mortar-bar expansion test. Each of these methods by itself will indicate reactive characteristics in most cases but each by itself is subject to possible error. Therefore, it is generally wise to investigate a suspected material by all the methods of test, when possible, and to interpret and correlate the results of the tests with each other.

**Chemical Tests:** Two chemical tests are used to indicate potential reactivity of an aggregate in conjunction with, and to substantiate, petrographic observations. The first is a sodium hydroxide test in which 10 g of crushed sample are put in a 10 pct solution of sodium hydroxide for 21 days. The solution is then analyzed for dissolved silica. With this test, it will generally be found that sands having a silica solubility of over 0.1 pct should be regarded as suspicious while those with 0.3 pct or over as potentially reactive. Interpretation of these results must always be made with caution. They must be correlated with petrographic findings or erroneous decisions may be made. The second chemical test is also a sodium hydroxide test but requires only three days to perform. It takes into account the reduction in alkalinity of the solution as well as the amount of dissolved silica. The ratio between the dissolved silica and the reduction in alkalinity of the solution will indicate the potential reactivity of the material. Again the interpretation of the results should be correlated with petrographic observations. This test was recently developed by the Bureau of Reclamation and has been described in detail.<sup>12</sup>

**Mortar Bar Expansion Tests:** The mortar-bar expansion test is the final, and probably the most acceptable, test for determining potential reactivity of a material. In this test, a graded crushed rock or sand, containing all standard sieve fractions, is mixed with a cement of known alkali content. Usually three types of cement are used, a high-alkali cement, a low-alkali cement, and when possible, the cement with which the aggregate is to be used in the structure. Mortar bars 1x1x10 in.<sup>13</sup> are made of the mixture and allowed to cure in a moist atmosphere in sealed containers at either room temperature (70°F) or at 100°F. Periodic length measurements are made, and the severity of reaction is estimated by the extent of the expansion in a given length of time. Usually mortar bars made of the same cements but a known nonreactive sand are tested simultaneously for comparative purposes. This test is of long duration and, although the higher curing temperature accelerates reaction, it



usually requires a minimum of six months to a year for completion. The test, therefore, loses some of its value as, generally speaking, results are needed in less than a year's time. Interpretation of the results of this test may be based on a comparison of the expansion of the unknown bars with the percent figures suggested by Blanks and Meissner.<sup>14</sup> They state that bars, made with high-alkali cement, which show no more than 0.02 pct expansion in six months' time may be considered as containing a nonreactive aggregate, those showing 0.04 pct expansion as suspicious, and those showing 0.1 pct expansion or more as definitely reactive.

**Incompatibility Tests:** At the present time, there are three tests commonly used to determine thermal incompatibility in concrete. Two different tests are used to determine coefficients of thermal expansion and one test is used to obtain coefficients of thermal diffusivity.

**Thermal Expansion Tests:** Thermal coefficients of expansion of aggregates, cement pastes, or concrete may be determined by either of the two following methods:

**The Interferometer.**<sup>15</sup> With this method, small specimens are ground into tetrahedrons or other suitable shapes with an overall size of about  $\frac{1}{4}$  in. These are then placed in a heating unit between fused-quartz discs and viewed through an optical interferometer. A light of known wave length is used and interference rings passing the cross hairs of the interferometer are counted per unit of time as the temperature is increased and the specimens expand. Knowing the temperature range, the wave length of the light, and the number of interference rings which passed, the coefficient of expansion can be determined. This test has the marked disadvantage of using small specimens which may be hard to fabricate and hard to make representative of a heterogeneous material, such as a granite or other coarsely crystalline rock, or a cement mortar.

**SR-4 Strain Gauge.\*** This method has only re-

\* Obtained from Baldwin Locomotive Co., Pacific Branch, San Francisco, Calif.

cently been developed for use with concrete materials although it has long been used to measure stresses in steels. The gauges consist of a series of loops of very fine wire cemented between thin paper strips. They can be cemented to an aggregate specimen or to concrete mortar. The expansion and contraction of the specimen during changes of temperature causes a change in the electrical resistance of the gauge wire. This change in electrical resistance can be read directly in terms of expansion on an electronic strain indicator graduated into millionths of an inch. The gauges can be attached to a free pebble or to one confined in concrete. If the pebble is confined in concrete, a second gauge may be cemented to the adjacent mortar in order to determine the relative expansion between the pebble and its surrounding medium. The entire specimen is then subjected to temperature changes in constant humidity, and at specified temperatures the readings of the electronic indicator are observed. From these results, the coefficients of expansion of the aggregate pebble and of its adjacent mortar can be calculated and the compatibility of that pebble, or its rock type, with its surrounding matrix can be determined.

**Thermal Diffusivity Test:** To determine thermal diffusivity coefficients, a cylindrical specimen, usu-

ally a drilled core, is used. The specimen should have a length which is twice its width. A thermocouple is embedded in the center of the cylinder by drilling a small hole to the center of the piece. The specimen is then immersed in boiling water (212°F) for 2 hr, after which it is transferred to cold running water of constant temperature. The temperature difference between the interior of the specimen and the cold water is recorded at either specified differences in temperature, or at specified time intervals. The temperature differences are then plotted in degrees F against time in minutes on a semilogarithmic scale and the best possible straight line is drawn through the points. From the graph, the time elapsed between temperature difference of 80°F and temperature difference 20°F is inserted in an equation from which the thermal diffusivity can be calculated.

## Conclusion

From the foregoing discussion, it may be seen that the investigation of rocks, gravels, and sands for the present-day large concrete structures may be an intricate and complex undertaking if the most suitable materials available are to be used. Full geological and laboratory investigations are time consuming and may be very costly so they are justified only when large structures are under consideration. Smaller concrete structures, however, are subject to the same factors of deterioration as the large ones and some information as to their aggregate characteristics is essential. When the structure is such that full geological and laboratory investigations are not justified, future worry and maintenance may be kept to a minimum if the proposed aggregate materials are examined in the field to determine the presence of deleterious constituents. If such are present, it would be well to have some laboratory tests made to determine the degree of their influence on concrete, or, to seek another and more suitable aggregate source.

## References

- <sup>1</sup> T. E. Stanton: *Proc. Amer. Soc. Civil Engrs.* (Dec. 1940).
- <sup>2</sup> R. F. Blanks: *Engrg. News Record* (March 1941) 46.
- <sup>3</sup> H. S. Meissner: *Jnl. Amer. Conc. Inst.* (April 1941); *Proc. ACI*, **37**, 549.
- <sup>4</sup> W. C. Hansen: *Jnl. Amer. Conc. Inst.* (Jan. 1944) **15**, No. 3, 213-225.
- <sup>5</sup> W. F. Abercrombia: *Contractors and Engrs. Monthly* (March 1943) **45**, No. 3, 92-96.
- <sup>6</sup> A. D. Conrow: Discussion of ref. 7, p. 1000.
- <sup>7</sup> W. C. Hanna: *Chemical Reaction of Aggregates in Concrete. Proc. ASTM* (1947) **47**.
- <sup>8</sup> J. A. Pearson: *A Concrete Failure Attributed to Aggregate of Low Thermal Coefficient. Jnl. Amer. Conc. Inst.* (Sept. 1941).
- <sup>9</sup> W. H. Johnson and W. H. Parsons: *Factors Affecting Thermal Expansion of Concrete Aggregate Materials. Jnl. Amer. Conc. Inst.* (April 1944).
- <sup>10</sup> Albert Weiner: *Jnl. Amer. Conc. Inst.* (May 1947).
- <sup>11</sup> H. K. Cook: *Automatic Accelerated Freezing and Thawing Apparatus for Concrete. Proc. ASTM* (1945) **45**, 813.
- <sup>12</sup> Mielenz, Green, and Benton: *Jnl. Amer. Conc. Inst.* (Nov. 1947).
- <sup>13</sup> *Method of Fabrication. ASTM Standards* (1946) Pt II, 14.
- <sup>14</sup> R. F. Blanks and H. S. Meissner: *Jnl. Amer. Conc. Inst.* (April 1946).
- <sup>15</sup> G. E. Merrit: *Interference Method of Measuring Thermal Expansion. Bur. Stand. Jnl. Res.* (1933) **10**, Res. Paper 515.



## Review of Progress in the Caving of Asbestos Ore

Asbestos ore is hard, well intersected by free fissuring but not completely enough to avoid heavy secondary blasting, and is a more valuable ore than usual for caving. These factors have resulted in deviations from standard caving practice in details but not in principles.

by Gerald Sherman

**M**INING asbestos ore by caving at Thetford Mines, Quebec, has been described in the Transactions of the Canadian Institute of Mining and Metallurgy in papers presented by the staffs of The Asbestos Corp. Ltd., in 1934, and Johnson's Co. Ltd., in 1941. The production by caving prior to 1949, in round figures, has been 14,500,000 tons of ore by The Asbestos Corp. and 5,500,000 by Johnson's Co. They are the only caving operations in asbestos at the present time. The Jeffrey mine of the Canadian Johns-Manville Co. at Asbestos, 40 miles southwest of Thetford, is well along in preparations for caving, and The Bell Asbestos Mines Ltd. at Thetford Mines, will follow their examples later.

GERALD SHERMAN, Member AIME, is Consulting Engineer, Johnson's Co. and Canadian Johns-Manville Co., New York, N. Y.

AIME San Francisco Meeting, Feb. 1949, and New York Meeting, Feb. 1950.

TP 2828 A. Discussion (2 copies) may be sent to Transactions AIME before May 31, 1950. Manuscript received Nov. 2, 1949.

Nearly all of the asbestos produced by the western hemisphere is mined in those districts of Canada. Since it has been the subject of a number of technical papers, this contribution is offered as a review of progress. Some introductory information seems appropriate, for few members of the American Institute of Mining and Metallurgical Engineers are familiar with the mineral, or how asbestos deposits differ from others in caving characteristics.

Asbestos is found in peridotite, a basic intrusive rock irregularly altered to serpentine and erratically fractured, well enough to cave in bulk, but leaving many blocks of solid ore of all sizes scattered through the deposit.

Chrysotile is the variety of asbestos produced in Canada. Its fibers are the most valuable commercially, having a good proportion of long fibers, strong, silky, flexible, and suitable for spinning. The fibers occur in all lengths, from the microscopic to 3 in., or more, the shorter are more abundant. Their values depend on length combined with other qualities noted above. They occur as veinlets filling fissures in the peridotite, with crystals compactly arranged in parallel across the cracks. They are sometimes lightly frozen to the walls but the partings are distinct, often marked by particles of magnetite adhering to the fiber ends. Sometimes they break away from the rock in slabs an inch or more in thickness.

In wide fissures the fibers are not always continuous, the "broken fibers" being separated by lines of parting with occasional scattered grains of magnetite, which is an injurious mineral to be eliminated when possible. In addition to the chrysotile "cross fiber," some "slip fiber" is found in fault fractures in limited quantities of less valuable character.

With the exception of the Asbestos Corp. ore, in which the content is said to be more uniform, asbestos is eccentrically distributed through the peridotite. It is the general impression that the higher grades are more often found in hard rock. There are rich pockets and leaner areas which range down through various grades to minor barren sections. Wholesale mining is necessary to make sure of recovering all the high grade for there are no reliable indications of its existence. Ore selection only appears in constructing assay-grade limits for caving.

The mill yield of the Thetford ore body in 1934 was 6 to 10 pct of the ore milled, made up of all classes, with occasional runs up to 15 pct or so.

Quotations for Canadian asbestos\* are as follows:

	Per Ton
No. 1 Crude ( $\frac{3}{4}$ in. and longer)	\$896 @ \$960
No. 2 Crude ( $\frac{3}{8}$ to $\frac{3}{4}$ in.)	350 @ 545
Spinning (less than $\frac{3}{8}$ in.)	204 @ 378
Other classes, price diminishing with length.	
Shorts or "refuse"	24 @ 46

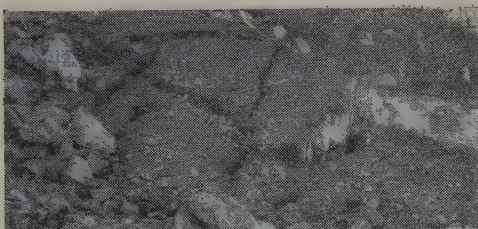
\* From *Eng. and Min. Jnl.*, Nov. 1948. Asbestos of even superior quality for electric insulation is found in Arizona limestone. It is white, containing very little combined iron, but occurs only in small quantities.

The pockets of high-grade ore in open pits containing crude asbestos in unusually close assemblies of veins, are blasted with care to avoid breaking down the long fibers. The rich portions are sorted out and cobbled by hand to free them from rock and magnetite and are separated into No. 1 and No. 2 Crude grades for market preparation.

The pit ore and the rejects from the above processes are dried and milled by alternate stage crushing and screening to release the fibers (deliberately fluffed up in milling), which are lifted out of the sands by pneumatic suction. Wood splinters which are rigidly excluded in specifications would also be picked up with them. The use of timber in mining is therefore not permissible.

In the final treatment, fibers are segregated by Quebec Standard screens into many specific grades.

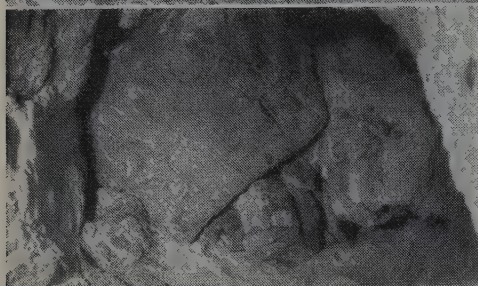




**Fig. 1—**  
Pattern of  
fracturing  
opened by  
subsidence of  
supporting  
ore. Natural  
fragmentation.



**Fig. 2—**  
Large block of  
untouched ore  
isolated by blast-  
ing about it.



**Fig. 3—**  
Large frag-  
ments of ore  
jammed in a  
draw hole  
above a  
grizzly.

Combinations of certain grades for special purposes are frequent. There must be close cooperation between the milling and sales departments, and the purchasers.

There is an increasing demand for asbestos, and now shorter fibers are being used than could have been sold a few years ago. Milling equipment has been adjusted to recover them, which will increase the profit and total quantity of fiber sold, but slightly decrease the average length and value per ton.

The similarity of the analyses of asbestos and peridotite prevents the determination of asbestos content by assay. It is estimated by "reading" diamond-drill cores from holes drilled for test. Veinlets of asbestos are measured, and other evidence appraised visually by experienced observers. As the asbestos distribution is extremely erratic, with rich and barren spots, the test holes must be closely and uniformly spaced to obtain enough samples to make up a true average. In doubtful areas holes have been drilled at the corners of 80 ft squares, but that might not be the minimum limit, for marginal ores. In proven ore bodies, accurate estimates of grade are not too important, but in locating the boundaries of an ore body, or in new low-grade deposits, the drill cores are occasionally milled in laboratory tests for a check on the readings.

#### **Thetford Mines Deposit and Ore Characteristics**

The Thetford ore body is pear shaped and about 2200 ft in length and 1300 ft in maximum width. Ore is now being mined more than 1000 ft below the original surface, but its ultimate depth is unknown. It was mined in open-cast pits to a depth of 350 to 400 ft, the usual economic limit of surface mining there.

The ore is too greatly fractured to be mined safely in open stopes, and timber underground is prohibited. By a process of elimination, caving was selected for the method of mining in spite of ore losses, dilution, and deterioration of fiber by

attrition in its passage to a draw hole. Caving from a high column of ore reduces the proportion of ground support necessary for miners' protection to a point where the cost of steel and concrete per ton of ore mined is not excessive.

The fragmentation of ore in subsidence has a controlling influence on the efficiency of caving and the cost of mining. There are many open fissures in asbestos ore as well as potential planes of weakness.

Patterns of fragmentation by free fissuring are shown in figs. 1 and 2 of surface exposures undermined by subsidence from below. The blocks so isolated may be broken in the process of drawing, but many are hard and arrive at the draw holes unchanged, possibly with their corners rounded. One mass 60 ft in length appeared, covering several draw holes.

The occurrence of serpentine, widely but irregularly distributed, is characteristic of the district. It is found on the faces of fracture planes and in pockets of loose fragments, wedge-shaped and slippery (fish scales). It is favorable for caving but makes a treacherous formation for development, particularly in raising.

There is a tendency for the fines to run out from among the larger boulders which frequently become locked together high above the grizzly level (fig. 3).

Drifts in ore 8 ft or less in width can usually be driven some distance in advance of steel sets which may be required later. If the roof is caught up before movement starts, the pressure is seldom great. The ore does not swell.

Close lagging with steel is more difficult than with timber which can be easily cut to fit. Concrete pumped or blown in is most effective in completely filling the space outside the sets.

The peridotite which surrounds the ore body is generally sound, and the shaft stations on the first level stand without support. Developments on the second level indicate the probability of weaker ground, and more steel may be needed. A trial of more massive concrete without steel is being made.

At the Canadian Johns-Manville Jeffrey mine the country rock between shaft and ore was badly broken and extremely heavy. Sets were spaced at half the customary distances, and throughout long sections, top forepoling and sideboards were necessary to hold the ground. Heavy steel beams used for spiling were bent or broken. Two parallel haulage drifts were driven to avoid traffic congestion, and will permit repairs without stopping production. The ore is well fractured but is not so heavy.

The area opened is but a small fraction of the total and may not be typical.

#### **Caving Asbestos Ore at Thetford**

In 1931 serious consideration was given to the possibility of profitably caving the asbestos ore at Thetford. The Asbestos Corp. was able to successfully transfer a highly specialized mining method developed for physically weak ores to a new and untested formation.

The general theory of caving requires no description here. Asbestos practice follows "block caving" as at the Miami Copper Co. in 1932, except that ore is drawn from the sides of the grizzly drifts. No ore pillars are left between blocks.

The essential differences from ores usually caved that affect the underground mining of asbestos are:

1. Asbestos ore is harder, and although natural fragmentation in the process of drawing is adequate



for caving, it is not complete enough to eliminate large masses of solid ore. The proportion of oversize boulders which reach the draw points is high, resembling the Climax molybdenum ore rather than the "porphyry coppers."

- a. More work is necessary at the draw holes to reduce the ore to handling size, at considerable expense. The daily production per block is low.
  - b. Delays which occur in the blocking of draw holes cause serious interruptions in systematic drawing.
2. Steel and/or concrete must be used in ground support instead of timber. It is more expensive but stands up better in grizzly drifts.
  3. To fulfill market requirements, certain grades of asbestos must be produced. Several blocks must be in production from which to select and blend the asbestos to obtain a suitable current product. For this, flexibility in rates of draw is necessary.
  4. It is possible to draw slowly from any block, or at the maximum rate, as desired. The essential strength of ore fragments is such that they are not materially crushed under accumulating pressure.
  5. The ore is more valuable than that from a typical caving deposit. More care should be taken, and expense incurred, if necessary, to obtain high recovery. Unfortunately, for lack of accurate preliminary sampling, records of ore recovery and dilution cannot be calculated.
- There is one condition in the Thetford ore body peculiar to that district. It is owned by three companies in adjacent tracts, within vertical boundaries, mined independently from three different elevations, two by caving, the third still from an open pit.

Company	Method of Mining	Elevation of Production Level above Sea, Ft.
Bell Asbestos Mines Johnson's Co. Asbestos Corp.	Open pit Caving Caving	Pit floor, 730 1st level undercut, 480 1st level undercut, 565 2nd level undercut, 165

Through the courtesy of the Asbestos Corp., Johnson's Co. was permitted to observe its caving operations after several years of production and were given information on block dimensions and the designs for steel and concrete construction in grizzly drifts, etc. Having the benefit of the example of a successful caving operation in the same ore, it was decided to follow the same methods.

The following is a description of Johnson's Co. practice, with which the author has been associated. It does not cover parallel investigations and improvements made by the Asbestos Corp. which, it is hoped, will be described at some later date.

#### Block Development

A standard block is 160 by 160 ft by 300 ft in height. To fit the ore outline it may be considerably longer. Local extensions in height are drawn with the rest.

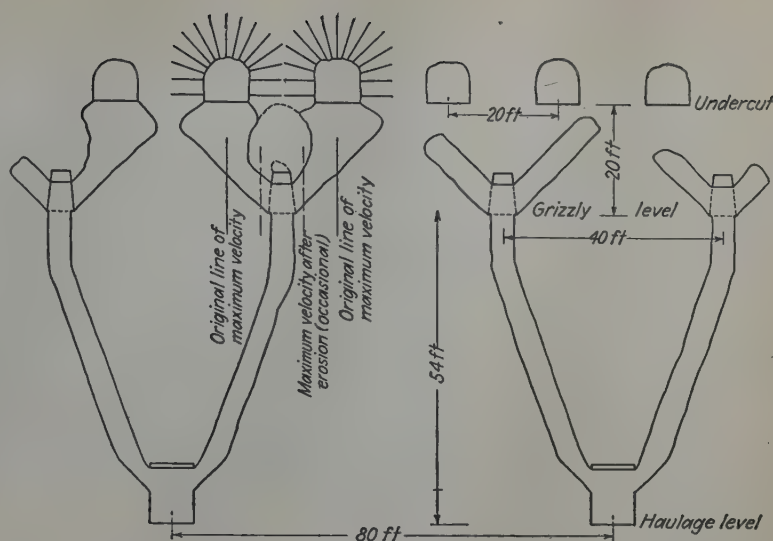


Fig. 4—Section through undercut, grizzly and haulage drifts, showing "ring drilling" for undercut.

The main development level is the grizzly drift (elevation 460 ft above sea level) which is directly connected with the shaft for service, supplies, and ventilation. The haulage level is 54 ft below.

The undercut is 20 ft above the drift, at elevation 480 ft, from which the caving height is measured (fig. 4). Corner raises and fringe drifts are driven from the undercut to the top of the ore, the latter spaced at 40 ft vertical intervals.

The grizzly drifts are driven 40 ft apart, and the draw holes and grizzlies are spaced at 26 ft 8 in. intervals in them. Finger raises are driven upward from both sides of the grizzlies and belled out to reach the undercut 10 ft from the drift centers, making the draw-hole spacing in the undercut 20 ft by 26 ft 8 in.

In the 40-ft section between grizzly drifts at the beginning of the draw, there are two draw holes in the undercut, each 10 ft from the drift center and 20 ft apart in the middle. It is found that by erosion, or blasting, the draw-hole centers are frequently shifted nearer the drifts, that is, to 6½ or 7 ft instead of 10 ft, leaving a high ridge 26 to 27 ft wide between them. The drift pillars occasionally are worn down to the concrete or steel, and arches may connect across them (fig. 5). This seldom occurs across the 27-ft ridge, which cannot be reached easily from either draw hole in attempts to bring down a hanging roof. In the first case, drawing from one hole may undermine an arch over the other, but there is little mutual assistance to be expected across a 27-ft gap, or the 26 ft 8 in., between draw points in the drift. Close spacing is obviously more effective, but there are practical economic limits in that direction.

The haulage drifts are parallel with the grizzlies, and 80 ft apart. The loading chutes are directly opposite across the drift, each receiving ore from a grizzly through a haulage raise.

Fringe drifts and corner raises are driven to facilitate caving promptly, to direct the planes of breaking, and to avoid projections of ore that may overhang from the next block and ride down on the fill (fig. 6). It is particularly important to develop a clean break on a property line.

Quebec safety regulations do not permit un-timbered vertical raises more than 40 ft in height.

Fig. 5—Top of draw point showing eroded pillar over grizzly and connection with opposite finger raise.

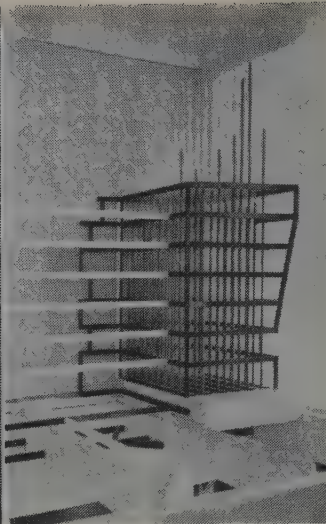


Fig. 6—Drawing model of block No. 1 sand in glass tubes indicates ore in draw points still to draw. Offset service raise, in left, rear. Inclined corner raise at fringe drifts on footwalls at right. Developments completed painted black.

The corner raises and fringe drifts of the first two blocks were driven from service raises which are in 40-ft sections, offset at the levels of the fringe drifts. A central service raise was afterward driven upward, from which all later blocks were developed by driving fringe drifts from it. Corner raises were driven through from drift to drift to connect them. These could usually be broken through without support, for no more than a hole is needed.

A small engine and cage hoists men and supplies from the grizzly level, and development ore is dropped through a chute compartment to a haulage raise. Many of the fringe drifts had to be driven in advance of requirements, but the gains in efficiency, cost, ventilation, and safety are satisfactory.

The undercuts have generally been broken by ring-drilled blastholes from 8-ft parallel drifts 20 ft apart. The drills were set up on long horizontal bars in the centers of the drifts, and the holes were drilled radially through the 12-ft pillars and over the drifts. Undercuts have recently been broken by horizontal, 40-ft diamond-drill holes charged with powder and detonated from both ends when possible. This breaks a shallower undercut that appears to be adequate in most ore and obtains some economies in drilling and blasting. It should be described when more data are available.

The steel sets for grizzly drifts, as built by Johnson's Co., are shown in fig. 7. It is a copy of the Asbestos Corp. design differing, if at all, in minor operating details. This is still standard for grizzly-drift caving. Steel alone was found to be too weak so it was reinforced with concrete (fig. 8).

Most of the grizzly drifts can be driven without even temporary propping. Since the steel is cut to standard lengths there must be some clearance for erection in the drift section, which, with the usual overbreak, makes room for an effective thickness of concrete.

The sets are bolted together, and a concrete floor laid, anchoring the posts in position. Plates are bolted to the inside flanges of the H-beam posts, etc., to serve as removable forms. Concrete, blown in by a pneumatic placer to fill all space between steel and solid ore, stiffens the steel sets materially, although there is not the concrete-steel bond obtained in reinforced concrete.

### Drawing Ore

The cost of drawing ore at Thetford is high. A great deal of secondary blasting is necessary to get the many large boulders through the draw holes and grizzlies. The draw holes are 16 sq ft in area, and the grizzly bars are 16 in. apart.

To break down the arches that often form too high above the draw holes to be easily reached from below, requires superior judgment and experience. Time must be allowed for necessary safety precautions. Production there is suspended until the arch is broken.

Records for a month have been assembled in table I of the approximate figures of consumption of labor and explosives for a comparison of efficiency data.

Table I. Drawing Ore, Segregation of Labor and Explosives

	Labor, Pct	Explosives, Pct
Breaking down arches:		
Drilling	21	32
Bombing	17	14
	38	46
Working ore through:		
Draw holes	38	39
16-in. grizzlies	24	15
	62	54

It has not been possible to change the standard grizzly-drift layout and design for the first level to increase the draw-hole area or shorten the spacing between draw holes.

The advantages of larger openings for the passage of bigger boulders are obvious. The remedy for the treatment of high-hanging arches is not as simple. When, at Miami, a pipe through broken ore brings capping in prematurely, that draw hole is blocked off and those immediately surrounding it are drawn in the hope that one or another will break the pipe and the flow of ore be resumed. If the holes are too far apart they will be of less mutual assistance.

It is thought that similar disturbances of ore above the grizzly level by the drawing of adjacent holes may break an arch, but they must be close to be effective. Increasing the draw-hole area is not likely to have much influence on the formation or breaking of high-hanging arches.



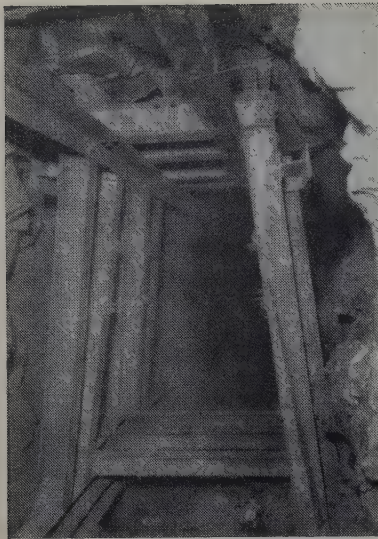
The remnants of ore drawn from the ridges about the draw holes are expected to be recovered from the lower block, if there is one. What is left lies above the steel and concrete of the grizzly drift. The probabilities of recovering that ore are not good and it would have little present worth. However the

The maintenance of grizzly drifts has cost less than expected. At least two thirds consists of repairs to the grizzly sets (cabanes) caused by blasting or the erosion of drift pillars over them, which sometimes exposes the steel. It is assumed that repairs will be in a proportion to the ore drawn. That due to blasting will be in direct proportion to tonnage. General repairs will be light in the beginning and will increase in time until replacement is extensive. There have been few replacements of cabanes to date, and only one drift collapsed from general pressure. In that case it is thought that undercutting left a pillar standing too small for the weight thrown on it.

The first block of Johnson's Co. was estimated to contain 1,112,700 tons of ore. According to the records, 1,074,000 tons were drawn, leaving some







**Fig. 8—Steel grizzly drift set at cabane showing grizzly bars and top of haulage raise. Floor concrete not poured.**

thousands of tons of known recoverable ore. Without knowing the original asbestos content, it is impossible to estimate either asbestos recovery or its deterioration by attrition in its transit to the draw holes.

The irregularities of ore fragments in size, added to great variations in the rate of drawing because of secondary blasting and interruptions because of hanging arches, obstruct effective attempts at systematic drawing. The conditions are distinctly unfavorable for high-ore recovery.

In ore as valuable as the asbestos ore caved, it is wise to thoroughly investigate the conditions essential for optimum recovery.

#### Height of Column of Ore Drawn

It is believed that the height that yields the optimum profit at Thetford is about 300 ft. That is the height\* at which the curve of cost per ton in rela-

\* F. W. MacLennan: Miami Copper Company Method of Mining Low-grade Orebody. *Trans. AIME* (1930) 91, 39.

tion to height, flattens off. Beyond that height, the economies in cost are trifling. Theoretically, the curve should represent the profit/height ratio but, in the uniform low-grade ore of Miami, the curves would practically duplicate each other.

At Thetford, where the cost preliminary to production and the ore value are both greater, the profit/height curve would be similar, but there are not sufficient data for its accurate construction.

To offset the gain in mining cost that might be obtained by a higher column caved, there are two adverse factors: the cost of grizzly-drift repairs, and the effect of height on ore recovery and dilution.

The cost of grizzly-drift repairs has averaged a little less than 4 cents per ton, but repairs during the last third of the ore drawn, cost at



**Fig. 9—Johnson-Bell property line, black. White line indicates outcrop of break on which Bell ore slipped.**

least one cent per ton above the average. The trend indicates a further cost increase for a block above 300 ft in height.

The proportion of ore lost, and top dilution, should not theoretically increase with height. Side dilution will be in proportion to the areas exposed to dilution. To preserve the ratio of ore to side areas in high blocks, the bases must be increased, possibly to an unwieldy area for drawing. It is also believed that side dilution will then increase faster in direct proportion to the height, because, during ore (and tailings) movement while subsiding, there is less resistance to pressure from capping in the previously mined block beside it.

It is generally assumed that uniform, systematic drawing is desirable and that it is attempted. It is practically impossible at Thetford. The cross currents that occur in the most uniform ore, systematically drawn, will become more eccentric when fragmentation is as irregular as at Thetford. Local interruptions in drawing caused by hanging arches still further obstruct uniformity, as retimbering does at Miami.

Although average results will ultimately follow principles, a loss of drawing control is more important in valuable ore. High columns are more vulnerable and the consequences more serious.

Although a moderate height has been recommended for caving asbestos ore, other conditions may affect the height selected. The second level of Johnson's Co. will draw from blocks 400 ft high, and Johns-Manville's first level will be 350 ft below the open-pit floor.

#### Trespass on a Property Line

In the early thirties, the three operations in the Thetford pit were not too far apart in elevation. The Asbestos Corp. went underground first and its greater production carried its active levels farther below the others. Johnson's Co. followed it down at a slower pace, and the Bell Asbestos Mines still more slowly.

The bottom of the Bell pit is approximately 250 ft above Johnson's Co. undercut and 565 ft above the Asbestos Corp. undercut on its second level (fig. 9). It is 315 ft between the drawing levels of the latter companies.

There was no barren capping within the pit areas when caving started, but a backfill of mill tailings has been deposited on the subsiding blocks and maintained at the Bell pit level, with a surcharge above, to assist in supporting the vertical faces of ore in place on the property lines. It may be considered an artificial capping to be treated as customary when drawing ore and avoiding dilution.

This practice was introduced by the Asbestos Corp. and followed by Johnson's Co. It has been generally successful to date, except where a major structural break dipping from the Bell pit, carried some of the Bell ore over the line into the Johnson's Co. workings. It was perhaps complicated by a temporary lack of sufficient resistance from the tailings, as occurs in the movement down to follow subsiding ore, which is frequently in surges.

The loss of ore was appraised after a joint examination, and compensation assessed in accordance with a method prescribed in a previous agreement.

#### Ventilation

As no rock gases have been discovered at Thetford, and as there is no decaying timber, ventilation



is mainly for the quick removal of powder gases from working places and their replacement by fresh air. If that is accomplished, there should be a good working atmosphere in the mine. Ventilation is a matter of efficient distribution rather than of the quantity of air forced into the mine.

In the cold winters of Quebec, ice will form in a permanent downcast opening. In the small area of Johnson's Co. mine there is a safety exit in depth through another mine, which can only be used in emergency. The shaft is partitioned between the skip and cage compartments (fig. 10). A ventilation fan is situated on the grizzly level near the shaft, which draws air down one half of it to the bottom (haulage) level, up through the workings, to and out through the grizzly drifts (where most of the blasting is done), to the fan, and forced up through the other half of the shaft to the surface. Doors at the two shaft stations can reverse the air through the shaft, thawing any ice near the collar by air at rock temperature, without changing the current through the mine or fan. It has not been necessary to do so frequently, even in a cold winter.

Only one air lock at the fan is needed, none in the haulage drifts, but there should be doors or booster fans to direct air to the producing blocks, and prevent short circuiting.

The distances are short, and mine resistance including the shaft bottleneck is not excessive.

In November 1948, gas, thought to have been methane, was discovered in a raise of the Jeffrey mine. It exploded and burned two men rather badly. A small flow from a fissure burned when lit. Methane-detecting equipment is now kept underground. It is unlikely that enough will be found to affect a ventilation system.

At the Jeffrey mine, the undercut level above the scam drifts will be only 350 ft below the floor of the open pit when it is abandoned. The producing area is too large for ventilation from one point (fig. 11).

The advantages of a shallow depth with convenient corner raises can be utilized by laying out independent unit-ventilation areas, each including several caving blocks with a fan of appropriate capacity and two raises, one for a fresh air inlet and the other upcast for the exhaust. They may be corner raises or one could be a service shaft if more convenient.

A pressure system has advantages. It would carry fresh air down to a connection with the haulage level and force it through scam drifts to the exhaust drifts and upcast raise.

All haulage drifts would be in the fresh air circuit fed with air under pressure. Leakage through air ventilation doors supplemented by air directly from the fans would carry air to the main shaft which would be upcast.

The course of exhaust gases would be short and direct.

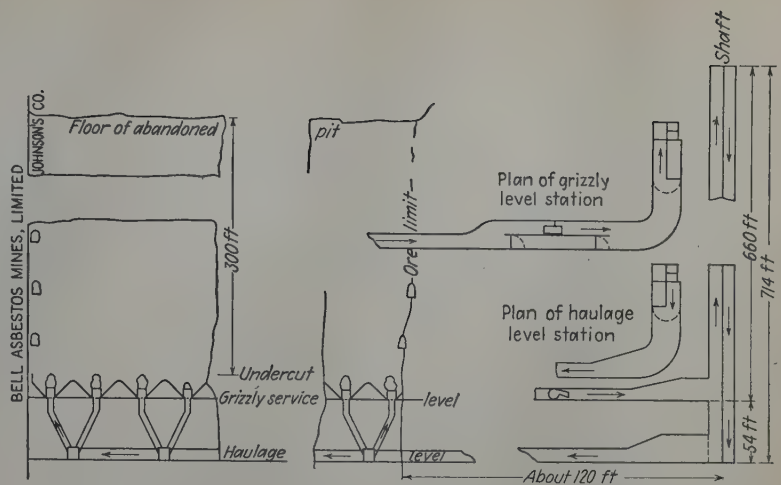


Fig. 10—Diagram of ventilation circuit in Johnson's Co. mine.

### Transportation Equipment

The haulage equipment (table II) for the three operations is planned for the projected rates of production, and the anticipated fragmentation of the ore as drawn; 4000 tons per day for two 8-hr shifts for the Asbestos Corp.; 2500 tons for Johnson's Co.; and 12,000 (plus) for the Jeffrey mine of the Canadian Johns-Manville Co.

Table II. Haulage Equipment in Caving Operations

Cars	Co.	Car Cap., Tons	Type	Dump	Track	
					Gauge, In.	Rails, Lb
Asbestos Corp.		5	Granby	Automatic, to one side	30	56
Johnson's Co.		5	E. Long	Automatic, to one side	30	60
Jeffrey mine		10	Special Rectangular, solid body.	2 car tippie	36	80

Minimum curve radius, 40 ft for the Asbestos Corp.  
50 ft for others.

Locomotives	Type	Cap., Tons	Volt, Dc	Haulage Drift Section
Co.				
Asbestos Corp.	Trolley	8-9	220-250	8 ft x 8 in.
Johnson's Co.	Trolley	6	125	8 ft x 8 in.
Jeffrey Mine	Trolley Battery	15	250	8 ft 4 in. x 9 ft 6 in. above ties

A small battery locomotive is used by Johnson's Co. for service on the grizzly level.

The Jeffrey cars are designed for strength, and capacity per square foot of drift cross section. Large

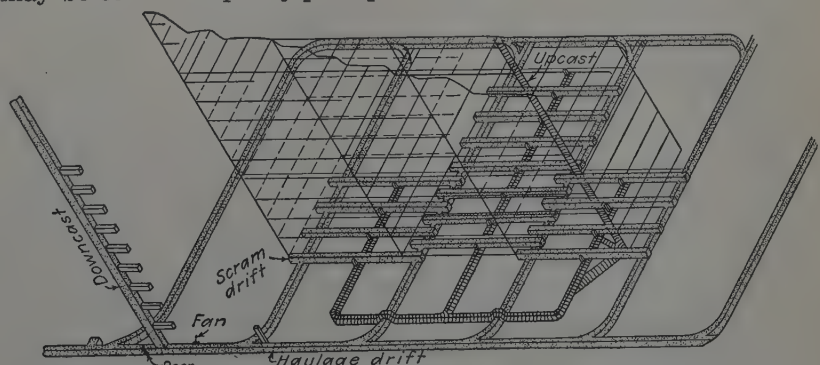


Fig. 11—Diagram of unit ventilation area in Jeffrey mine of Canadian Johns-Manville Co., under pressure system.

boulders will be dropped several feet into them.

The rail weights are for main line tracks. Branch lines for loading ore at Johnson's Co. mine have 40 lb rails.

A heavy copper booster cable was strung by Johnson's Co. from the shaft and cut in at the end of the main line haulage track to permit the use of 125 v current within the ore body. Automatic reels were used in development and found so convenient that their use was continued for haulage through the loading zone. They eliminated trolley wires, rail bonds, and protection for loaders at the chutes.

Loading chutes at Thetford are 48 in. wide, controlled by Cramp chain gates. Johnson's Co. chutes have hinged lips to bridge the gap between chute and car and reduce spilling in loading.

Fine serpentine and fiber on the tracks reduce driving-wheel traction which limits the live train load. Heavier locomotives would increase it.

### Hoisting

All mine shafts are of the rectangular type with a large cage compartment, two skip compartments, and a fourth for a ladderway, cage counterweight, air and water columns, and power lines. The dimensions are proportional to production, and the large cages will accommodate locomotives and cars in running condition. The Asbestos Corp. cage has special fittings for lowering long steel sections conveniently.

Underground equipment for mines, crushers, pumps, locomotives, etc., are increasing in unit weights and numbers used. Supply service through the shafts for mines of large production must be planned for the rapid handling of men and supplies, with a margin for increases.

Steel beams are employed for all shaft sets. Johnson's Co. uses copper steel with British Columbia fir, creosoted, for lagging.

At the Jeffrey shaft which passes through badly shattered rock with more than usual overbreak, and which will serve immense ore reserves, concrete is poured about the steel sets in place, flush with the beam flanges of wall and end plates. Johnson's Co. recently sank its shaft 400 ft deeper, concreted in a similar manner, because test holes indicated doubtful ground.

Three sizes of skips are used: 6 tons for Johnson's Co., 7 for the Asbestos Corp., and 10 for the Jeffrey mine.

Two of the three shafts were sunk through considerable depths of water-bearing glacial sands and gravels.

	Depth to Water, Ft	Depth to Rock, Ft
Asbestos Corp.		15
Johnson's Co.	24	94
Jeffrey shaft	35	79

The Johnson's Co. and the Jeffrey shafts were sunk to rock under contract by the Foundation Co. of Canada, which carried down large open wells, inside of which hollow concrete cylinders were poured. The concrete sections are sealed into rock, below which the shafts assume their dimensions. Quicksand in the Jeffrey shaft made it an extremely difficult job.

The concrete cylinders are also used as foundations for the headworks. The Jeffrey shaft continued the concrete cylinder upward in a reinforced sec-

tion as a self-contained head frame for shaft guides, sheaves, dumping gear, etc.\*

\* Can. Min. and Met. Bull. (Jan. 1949).

### Scram Drift Method of Drawing Ore

The cost of drawing ore through small draw holes and 16 in. grizzlies is very high because of secondary blasting in cramped quarters to reduce large ore fragments to handling size, and because of trouble with hanging arches.

In the scram drift-scraper system developed at Climax, much larger passages for ore are provided for transportation to the underground crusher, which will protect the skips.

The Jeffrey mine, of the Canadian Johns-Manville Co., will follow Climax practice in using draw holes 8 by 4½ ft above scram drifts 7-ft wide. Heavy, powerful scraping equipment and mine cars will deliver the ore to jaw crushers at the shaft with openings 48 by 60 in., which will be the bottleneck for size. Secondary blasting will be greatly reduced in amount, and accomplished under favorable conditions.

Johnson's Co. developed an experimental block of ore for mining through scram drifts. As production will be on a smaller scale, draw holes 7 by 4 ft will deliver the ore to 6-ft scram drifts. Some ore has been drawn which indicates an important gain in drawing efficiency, although, because an underground crusher has not yet been erected, more secondary blasting is done than will be necessary later. Two standard caving blocks on the first level are being prepared for caving by the Climax method. The second level transportation will be designed for better efficiency under new conditions.

It is doubtful if there will be much reduction in the number of hanging arches or the manner of treating them. The arrangement of scram and haulage drifts will permit convenient changes in draw-hole spacing. Tests can be made to determine how they influence the number of hung draw holes, drawing costs, and ore recovery.

If there are fewer interruptions in drawing, there might be a gain from more systematic drawing. Trouble is anticipated in allocating correct quantities of ore to each draw hole, which would have the reverse effect on drawing control.

### Summary

Since caving began in asbestos ores in 1934 there have been minor changes in caving practice until recently, owing to the inflexibility of the grizzly system using steel and concrete.

As in other mining operations, the use of labor-saving equipment has increased. Jumbos for drill mounting, and power loaders are used in development.

Owing to the high cost of steel and the occasional difficulty of getting it, trials are being made of replacing some of it, at least, with heavier masses of straight concrete. This is mixed and placed by mechanical means.

The major change is in the shift from drawing through grizzlies to larger draw holes and scram drifts that are better suited to handling ore incompletely fragmented in caving.

### Acknowledgment

In closing, the author wishes to express his thanks to the three companies for permission to use their operating data, and to their staffs for assistance in collecting it.



# Development in the Use of Steel for Underground Support

by F. J. Haller

The need for permanent, fireproof support indicated structural steel sets. Experience over the past six years, involving more than five miles of permanent underground openings, has proved that steel is permanent, safer and, in the long run, cheaper than either treated or untreated wood supports. Excavation is reduced, as are installation, transportation, and handling costs. A number of hazards are reduced or eliminated by the use of steel.

IN 1943, we found, in the new Mather operation, a very unusual and disappointing condition in the footwall rock where all of our main haulageways were to be located. With the exception of a few hundred feet out of the thousands then planned and since driven, all of the openings require support. In general, the ground is not particularly heavy, at least by our standards, but it must be supported and the back carefully covered. Also, our operations require rather wide flat openings (16 to 18 ft) and rather flat curves at the turn-outs, all of which adds to the problem. When this completely unforeseen condition was discovered, ordinary timbering was the only answer, because of the impossibility of obtaining adequate steel supplies in time due to war shortages. Then by the time steel supplies were obtained and a practice developed, we found ourselves with between four and five miles of main haulage drifts and flat openings of a permanent nature, all of which were supported by hardwood timber, some of which had already rotted to an unsafe degree. Also, our drifting program was adding some two miles per year to the above figure.

F. J. HALLER, Member AIME, is Superintendent, Mather Mine, Cleveland-Cliffs Iron Co., Ishpeming, Mich.

AIME Columbus Meeting, September 1949.

TP 2840 A. Discussion (2 copies) may be sent to Transactions AIME before May 31, 1950. Manuscript received Aug. 22, 1949. Revision received Jan. 17, 1950.

To give all of the details of our experiments, studies, and experiences would fill a large book, but we were satisfied that our organization had done an excellent job, starting from scratch. You can appreciate our surprise when we discovered that we had added very little to an existing record. Attention was called to a footnote in a copy of "Carnegie Pocket Companion" year 1923. The note read: "Full information as to uses of H-Beams is given in pamphlet entitled 'Steel Mine Timbers.'" A search of our Engineering Library brought to light a pamphlet printed in 1911 showing extensive use of steel for

this purpose. Except for welding in place of riveting, we have added very little to the designs in that book. We have, however, developed an experience which proves, at least to our own satisfaction, that steel mine supports pay off under the proper conditions. We are now convinced, in spite of the relatively high initial cost of materials, that steel sets, because of comparatively low installation cost, are actually cheaper than either treated or untreated wood sets, if they are properly designed and adapted.

**Several Sections in Use:** Convenience guided our first experiments in steel sets, since for a number of years, we had used the 4-in. WF, 10 lb per ft section in 10-ft lengths as safety forepoles along with wood forepoles. This section replaced the 40-lb rail which had formerly been used for the purpose. Having the 4-in. WF sections on hand, it was natural that our first experiments in steel sets involved their use. Later, the rolling of the 10-lb section was discontinued in favor of the much stronger 13-lb section, which is now standard for both legs and caps in areas of no particular weight, but which, none the less, require permanent support.

Our experiments involved the use of the 10- and 13-lb sections as legs and 6- and 8-in. I beams, 16 and 23 lb per ft, respectively, as caps. Experience soon indicated that, if the ground were heavy enough, and the opening wide enough to require a cap stronger than 13-lb, the 8-in, 23-lb should be used, eliminating the 6-in. I beam. We now know that the 8-in. WF section of the same weight, which we propose to use in the future, is considerably stronger for the amount of money involved.

**Comparative Strengths:** Tables I to IV giving strengths of several species of timbers as compared with the structural sections that we now use have been prepared in accordance with accepted practice. Experience indicates that failure due to compression perpendicular to the grain of the wood does not destroy the effectiveness of a wood set frequently enough to be a practical factor. Therefore, no table of these values is included.

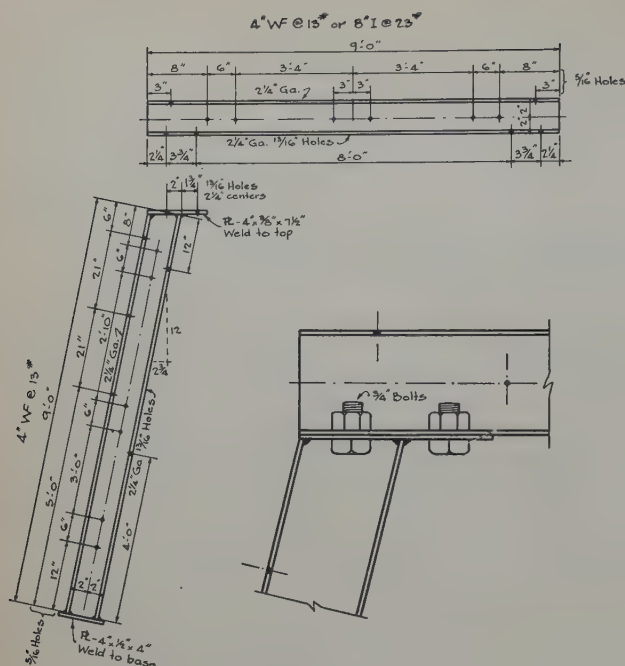


Fig. 1—Diagram of steel sets.

Table I. Timber Caps, Uniform Load, 8-ft Span

Ultimate Breaking and Safe Working Loads in Accordance with the Stresses Recommended by the U.S. Forest Products Laboratory

Size Diam In.	Maple		Tamarack		Spruce	
	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb
12	134,000	24,100	102,000	20,500	81,000	18,400
13	169,000	30,600	129,000	26,000	103,000	23,400
14	220,000	38,200	162,000	32,600	128,000	29,200
15	259,000	46,900	198,000	40,000	157,000	35,900
16	314,000	56,900	240,000	48,500	191,000	43,500
17	378,000	68,500	290,000	58,300	229,000	52,300
18	448,000	81,000	343,000	69,100	272,000	62,000

Ultimate and Safe Working Loads Due to Horizontal Shear in Accordance with the Shearing Stresses Recommended by the U. S. Forest Products Laboratory

Size	Ultimate	Safe	Ultimate	Safe	Ultimate	Safe
12	248,000	24,700	146,000	17,900	129,000	16,200
13	291,000	28,800	171,000	20,900	151,000	18,900
14	337,000	33,600	198,000	24,200	175,000	22,000
15	387,000	38,400	228,000	27,900	202,000	25,200
16	441,000	43,800	260,000	31,600	230,000	28,700
17	496,000	49,300	292,000	35,700	258,000	32,300
18	558,000	55,400	328,000	40,200	290,000	36,400

In table I, it will be noted that in practically all cases the shearing stresses govern the safe working loads.

Table II. Steel Caps, Uniform Load, 8-ft Span

Ultimate Loads (Elastic Limit) and Safe Loads (A.I.S.C. Spec.)

Size	Ultimate	Safe
4 in. WF 10 lb <sup>a</sup>	12,100 lb	7,000 lb
4 in. WF 13 lb	15,900	9,100
8 in I 23 lb	46,700	26,700

<sup>a</sup> No longer rolled.

At one time we used a 6-in. I beam, 16 lb per ft, with an ultimate load of 29,400 lb and a safe load of 16,800 lb. We no longer use this section, changing from the 4-in. WF, 13-lb to the 8-in. I beam, 23-lb where heavy conditions are encountered or where spans exceed 8 ft.

The first use to which the tables can be put is to rule out the breaking loads in favor of the lower horizontal shear loads, with the exception of the 12-in. maple cap, where the breaking load governs. In all other cases, the ratio of span and diameter is such that horizontal shear governs the safe load. The 4-in, 13-lb WF section as a cap initially has approxi-

mately 40 pct of the safe load strength of a 12-in. diam green hard maple, 45 pct of tamarack and 50 pct of the strength of spruce. These strength ratios become less favorable for steel as the diameter of the timber increases. However, within a very short time, wood loses the advantage in strength due to aging. The 8-in, 23-lb I beam, when used as a cap, compares very favorably with the 13-in. diam maple, tamarack and spruce, the last of which it exceeds by some 3000 lb. Inasmuch as judgment of ground is the ruling factor, we quite frequently use the 8-in. section where heavy conditions are expected. Where light conditions are expected and weight is encountered later, additional sets of the 13-lb section are installed.

Table III. Timber Columns, 9 ft 0 in. Long

Ultimate Breaking and Safe Loads in Accordance with the Stresses Recommended by the U. S. Forest Products Laboratory

Size Diam In.	Maple		Tamarack		Spruce	
	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb	Ultimate, Lb	Safe, Lb
12	455,000	164,000	393,000	136,000	300,000	110,000
13	532,000	192,000	461,000	159,000	354,000	129,000
14	617,000	223,000	534,000	184,000	411,000	150,000
15	710,000	256,000	614,000	212,000	472,000	172,000
16	806,000	292,000	699,000	241,000	537,000	196,000
17	911,000	330,000	789,000	272,000	606,000	221,000
18	1,043,000	369,000	885,000	306,000	680,000	248,000

Table IV. Steel Columns, 9 ft 0 in. Long

Ultimate Loads (Elastic Limit) and Safe Loads (A.I.S.C. Form.)

Size	Ultimate	Safe
4 in. WF 10 lb <sup>a</sup>	70,000 lb	32,300 lb
4 in. WF 13 lb	94,700	42,800
8 in I 23 lb <sup>b</sup>	111,200	56,600

<sup>a</sup> No longer rolled.

<sup>b</sup> Up to now we have had no occasion to use this section as a column.

In comparing steel with wood for timber columns or legs, once again the original strength is definitely in favor of the wood. The 4-in., 13-lb section, with its safe load of 42,800 lb, has only 26 pct of the strength of a 12-in. green maple, 31 pct of the strength of tamarack and 39 pct of the strength of

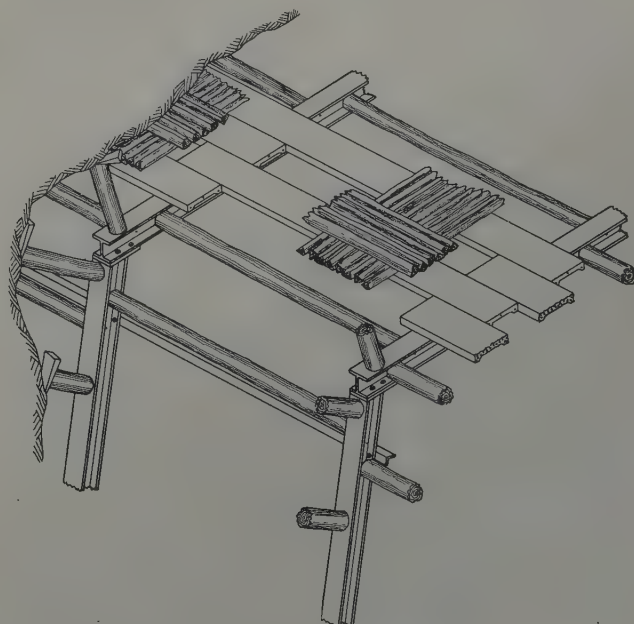


Fig. 2—Diagram of spragging.



spruce. Once again, the rapid loss in strength of wood makes the use of steel economical. Further, the strength of steel is predictable and the strength of wood definitely unpredictable. The tables are based on certain specific moisture contents of wood which are subject to considerable variation.

Another factor which up to now has not been reduced to formula, definitely favors the steel supports. The fact that the cap is securely bolted to the top of each leg with four  $\frac{3}{4}$ -in. bolts greatly increases the strength of the set as a whole when compared with wood sets, which cannot be securely fastened together. A combination of top and side weight quite frequently destroys the effect of a wood set, while steel of a lighter theoretical strength under the same conditions does an adequate job of support.

Up to now, we have had no occasion to use anything stronger than the 4-in., 13-lb section as legs, except in areas of extreme weight, where an entirely different section is used. This will be discussed later.

We have prepared for this discussion a number of illustrations showing the method of shop fabrication and underground installation. As mentioned previously, we have added little except experience and the use of welding in the place of riveting.

Fig. 1 is a detailed shop drawing. Accuracy is obtained by performing all drilling and welding in a jig. Reasonable accuracy is necessary, since it is important that all four  $\frac{3}{4}$ -in. bolts be used at both ends of the cap for proper rigidity.

Proper spragging is important if steel is to be installed as the heading is advanced (fig. 2). Conventional wood lagging and blocking can be used above permanent carrying poles where fireproofing is not important. "Minecrete" backpoles are  $2\frac{1}{4} \times 8$  in. and carry a uniform load of approximately 2700 lb over a 5-ft span (figs. 3 and 4).

Fig. 5 illustrates permanent, fireproofed carrying poles and blocking versus temporary wood poles, lagging and blocking. New, stronger Bethlehem "hat-section" carrying poles now are in use in heavier areas.

**Proper Application Important:** To enlarge on my qualifying remarks regarding the proper design and application of steel sets: there are places where we cannot afford to use steel and also there are numerous places where we cannot afford *not* to use steel sets.



Fig. 3—Treatment of permanent plat opening close-covered with "Minecrete" back-poles and blocked with cinder blocks where fireproofing is essential.



Fig. 4—18-ft span adequately supported.

Permanent fireproof side lagging was accomplished with "Minecrete" poles.

Normally, our mining openings, such as sublevel transfer drifts and exploration drifts in ore, are not intended to last beyond the normal 3 to 4 year life of untreated timber. Also, we can usually expect weight far in excess of the strength of the lighter, cheaper steel sets. Therefore, we use large size green timber, quite frequently reinforced with additional lining sets between the originals, in all cases where judgment indicates that the weight will be great and the length of life reasonably short. In general, our main level haulage drifts in ore fall into the same category, with a number of exceptions, where steel sets close together have done a satisfactory job.

The openings where steel supports are an economic necessity are, for the most part, permanent main level openings around the shaft stations and permanent main level haulage drifts in the foot-wall rock. There have been no instances in the several miles of main level haulage drifts and permanent openings where the use of steel did not prove economic. We have also found that the use of steel is a must in any areas where fireproofing is indicated, particularly in the vicinity of the shaft, in isolated airways, and in the vicinity of powder magazines. We have also found that steel sets are most desirable in areas which were originally supported with wood sets, where weight has not proved excessive and where a change in plans requires support beyond the life of the timber. This, of course, is true in our main level drifts driven during the time when adequate steel supplies were not obtainable.

**Smaller Opening Required:** Originally, we found it very difficult to control blasting in a main level rock heading to permit the installation of steel sets as the drift was advanced. Constantly changing ground and an unavoidable turnover in personnel made it impractical to install these sets because of the large percentage of legs and caps which were damaged in blasting to the extent that they had to be replaced. During the past year, however, we have successfully driven more than a mile of main level rock drift with steel sets at 5 ft centers without any appreciable damage to the supports. This is a large factor, since the opening required for 12-in. diam timber sets is some 25 to 30 pct larger than the opening required for steel.

**Comparative Costs:** The following presents average costs, including freight and handling, for prefabricating steel sets as done in our company's gen-



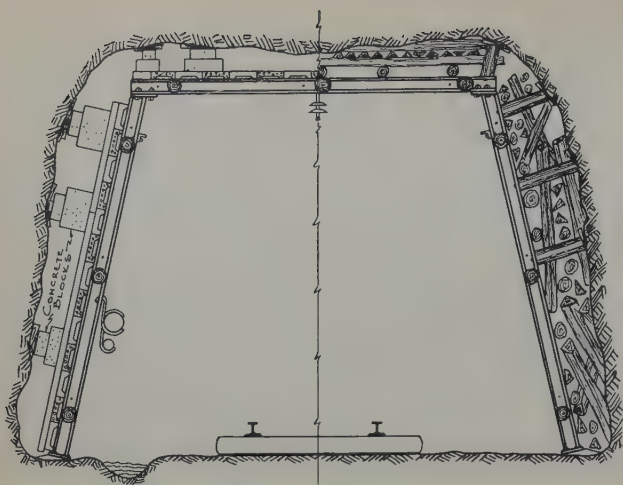


Fig. 5—Diagram of permanent fireproofed carrying poles and blocking vs. temporary wood poles, lagging, and blocking.

eral shops, as compared with the cost for wood sets. I do not propose to discuss the comparative costs or advantages of untreated wood versus treated wood, since in practically every instance where the life of untreated wood is to be exceeded, steel sets are indicated. Our average cost for a prefabricated steel set, including shop costs, freight and handling to the shaft collar, is \$20 for a three piece 9-ft set of the 4-in., 13-lb WF section. This compares with average costs for hard maple, including freight, unloading, and framing, of \$8.50 for 12-in. diam, \$10 for 14-in., and \$10.50 for 16-in. I do not have comparative costs for treated timber of the same grade and diameter, but understand that, in general, the cost of peeling, seasoning, and treating about equals the original purchase cost. If the steel set is to be used as an original installation in a rock drift, the first saving is the above-mentioned 25 to 30 pct decrease in the volume of rock excavated. To reduce this saving to actual money would be very difficult, but it is obvious, at least in our case, that it is considerable. The time for installation is approximately one-half of that for wood sets, with a cash saving which varies with the wage scales and skill of the men, but which averages \$4 per set. The best comparison between installation costs can be obtained where a main level haulage drift is being repaired, with the wood sets being removed and new steel sets being installed. We have several crews of repairmen which

give us an average of \$10 per set installation costs. This includes a complete job with the new steel set installed, the old wood set removed from the drift, and the back completely covered and blocked. Under these same conditions, with the haulage trains going back and forth, with the necessity for removing the air lines, water lines and trolley wires, it is practically impossible to conduct this repair work except on weekends at penalty rates without seriously hampering the tramming operations. The small 4-in. section permits this work to be done at straight time rates, since the legs can be slipped into place behind the air and water lines, and the caps installed without removing the trolley wire. Under these conditions, the installation cost of a 12-in. wood set exceeds \$20 with the same work completed. Obviously, a \$20 set which can be installed for \$10 costs about the same as an \$8 or \$9 set with \$20 for installation. The steel set, if put in an area not beyond its capacity, will last forever. The wood set untreated will last 4 years and, quoting other authors, "if treated will last not more than 15, with an average of less than 10 years."

In the repairing of important transfer drifts under large stopes or block caves, the same advantages of the steel sets hold true. So many more can be installed by the same number of men under heavy conditions that, in a great many cases, it is far cheaper to repair a wood drift with steel than to repair it with more wood.

**Transportation and Handling Reduced:** The transportation of timber and timbering supplies is another important cost item which is difficult to reduce to dollars and cents. In the case of the 9-ft, 4-in. steel sets, the average load on a timber truck is 18 sets, or 54 pieces, which compares with 7 pieces of 12-in. diam hardwood. In other words, transportation and handling cost from the timber yard or shop to the working place is approximately  $\frac{1}{8}$  the cost of wood. This is a very important item in our operation, where labor accounts for approximately 60 pct of the cost of the product.

As previously mentioned, the average mining drift in ore has too short a life to warrant steel in the original installation and is usually too heavy to be supported by the lighter, cheaper sections. However, there are a great many exceptions to this general rule in the 4000 ft of combined development openings driven at this property each month over the past several years. There are innumerable instances in stope and block cave transfer drifts where steel is absolutely essential, particularly in supporting the area in the immediate vicinity of mills and draw raises. Numerous methods are used and in some instances we have gone to a  $6\frac{1}{2} \times 12$ -in. WF, 28-lb section for both legs and caps. This section, which is approximately  $2\frac{1}{4}$  times as strong as the 8-in. standard I beam, serves admirably in areas where the side pressure is equal to, if not greater than, the weight on the cap.

In a number of instances, we have driven semi-permanent top timber carloading transfer drifts in rock and contact material. Invariably, since the excavated material is not salable, we are desirous of cutting down the amount excavated. The use of steel supports in these openings is mandatory because of the indicated length of life, the decrease in the excavation, and the absence of great weight as a general rule. However, in a number of these drifts where great weight was later encountered, additional lining sets have done an adequate job.

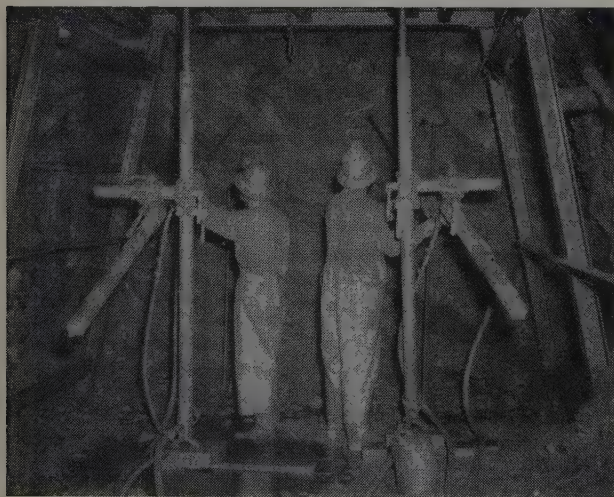


Fig. 6—Typical sublevel development in rock. Wood carrying poles may or may not be replaced with permanent "Minecrete" poles and Bethlehem "hat-section", depending on anticipated length of life.



Recent Developments in the Manufacture of

## Lightweight Aggregates

by John E. Conley

and

John A. Ruppert

Shortages of cinders, largely resulting from industrial plants converting to fuel-oil, together with an enhanced building activity, have seriously affected the cinder block and lightweight concrete manufacturers. In the search for other sources of lightweight aggregate, various clays, shales, slates and natural occurring lightweight rocks such as pumice, scoria, volcanic ash and the like have received special attention. Methods and equipment for producing suitable lightweight aggregates have been considered. In addition to an expanded use of the rotary kiln, the traveling-grate and continuous-sintering machines are being installed in many localities.

**L**IGHTWEIGHT aggregates have been in use for many years in the United States but are now receiving more and more attention by manufacturers and users of concrete shapes. These shapes comprise building blocks, special forms, such as door sills, lintels, floor slabs, and beams. Lightweight aggregate also is adapted for monolithic construction. The increased building activity since termination of the war, together with the widespread shortage of cinders resulting from a general trend toward the use of oil and pulverized-coal fuels, has aggravated the situation.

In some localities, new producers of lightweight aggregates have been stimulated to increased efforts to supply the demand. In addition, various estab-

*JOHN E. CONLEY, Member AIME, is Senior Chemical Engineer, and JOHN A. RUPPERT is Chemical Engineer, Eastern Experimental Station, U. S. Bureau of Mines, College Park, Md.*

*AIME San Francisco Meeting, February 1949.*

*TP 2834 H. Discussion (2 copies) may be sent to Transactions AIME before May 30, 1950. Manuscript received June 9, 1949.*

*Papers by authors of the U. S. Bureau of Mines staff are not subject to copyright.*

lishments of the United States Government have recently given attention to this problem. The National Bureau of Standards, in the Commerce Department, and the Bureau of Reclamation, in the Interior Department, have completed a series of tests on concretes made from some of the available commercial aggregates in order to evaluate their qualities. The tests have been made to focus attention on the desirable properties of good aggregates. This work was done at the request of the Technical Office of the

National Housing Agency and Housing and Home Finance Agency. Some time previously, the former office had conducted a survey of the locations where lightweight aggregates are produced.<sup>1</sup> A study of this information discloses that the producing centers are widespread and that certain areas, notably the region east of the Mississippi River and especially the southeastern states in the sections along the Atlantic Coast, have only a limited number of such plants. The situation has been alleviated to some extent with new plants having been erected near Richmond, Va., and Lansford, Pa. Experiment Stations of the Federal Bureau of Mines at Tuscaloosa, Ala., Norris, Tenn., and College Park, Md., have investigated types and sources of potential raw materials such as clays, shales, and slates, as well as methods and procedures for utilizing these materials for their conversion into aggregates, and have published the data as a Bureau of Mines report.<sup>2</sup>

**Structural Uses of Lightweight Aggregates:** Production of masonry blocks in the United States has averaged close to a billion units a year for the past three years, and of these it is estimated that more than 40 pct consisted of the lightweight type. It is anticipated that the future will witness an increase in this percentage in view of the superior insulating properties, decrease in dead weight, and ease of handling, transporting, and laying units of this type.

Lightweight aggregates are used extensively in multistoried buildings in which the weight of the walls and floors is carried by a structural steel framework. The substitution of a lightweight concrete for normal concrete in a building of this type has brought about in many instances some or all of the following advantages:

1. A reduction in the amount of structural and reinforcing steel needed.

2. Better heat and sound insulation in walls and floors.

3. Lower costs for the construction of concrete forms.

The advantages gained by using lightweight concrete is illustrated by changes in the original design just before construction of a 14-story annex to a large department store in Cleveland, Ohio.<sup>3</sup> The initial plans of the building contemplated three mezzanine floors using concrete of crushed stone and sand. Before ground was broken, however, the owners decided in favor of seven additional mezzanines, or a total of ten. Foundation limitations made the use of lightweight concrete mandatory, and tests were carried out by a private laboratory to arrive at proportions that would consistently yield concrete of the required strength and workability. Specifications were set up that readily met the 3000-lb compressive-strength requirement. With no sacrifice in strength, it was possible in this instance to effect approximately an 8000 ton reduction in weight to be supported by the steel columns.

By utilizing lightweight concrete, there also exist many instances in the United States where it has been feasible to add a greater number of stories to existing buildings than would otherwise have been possible with the conventional sand-gravel concrete.

During World War II, lightweight aggregates were used in making concrete ships and barges. Considerable savings were effected by this method of converting dead weight to cargo capacity.

The use of a lightweight aggregate in place of gravel or crushed rock in the concrete roadway of the San Francisco-Oakland Bay Bridge, completed in 1936, is said to have effected a saving of three million dollars because of the decreased quantity of structural steel necessary.

In recent years, there has been an increased use of superlight aggregates in concretes that are characterized by their excellent heat-insulating values. Because of their extremely low weight, concretes made from this type of aggregate cannot be expected to have high strength characteristics. Where strength is essential, more cement must be used, and an increase in weight results. However, strengths are adequate for such uses as fireproofing for steel beams, non-load-bearing subfloors and insulating sections of walls and roofs. Farmers in the colder areas of the United States have discovered that a subflooring of insulating concrete will prevent condensation in barns, thereby keeping the floor dry and warm and contributing measurably to the health of the animals housed therein.<sup>4</sup> Another advantage of concretes of this type is that they usually can be nailed or sawed.

**Types of Aggregates:** The aggregates finding commercial acceptance today in the United States might be roughly classified into two groups. The first comprises the extremely light materials, such as exfoliated vermiculite and expanded perlite, weighing 3 to 25 lb per cu ft and utilized in concretes where high-insulating characteristics are desirable. These materials usually yield concretes having 28-day compressive strengths in the range of 100 to 1000 psi. Because of their extreme bulk, aggregates of this nature seldom are transported long distances in their expanded forms but are shipped in the raw state and processed in the areas where they are consumed.

Aggregates of the second group include those

weighing 30 to 70 lb per cu ft and capable of yielding concrete with 28-day compressive strengths of 1000 to 5000 psi or more in some instances. These medium- and high-strength aggregates might be further classified as to methods of manufacture as follows:

1. Naturally occurring aggregates.
2. Expanded or foamed slags.
3. Bloated aggregates produced in rotary kilns.
  - a. Crushed to size after firing.
  - b. Crushed or extruded to approximate size before furnacing and then fired without agglomeration.
4. Sintered aggregates.

Methods for producing aggregates of both groups will be discussed briefly with particular emphasis on history and recent developments.

**Vermiculite:** Vermiculite is a hydrated magnesium aluminum silicate that possesses the extraordinary characteristic of expanding from 6 to 20 times its original volume upon being heated. In the United States, it is mined in Montana, Wyoming, Colorado, South Carolina, and North Carolina. The expansion or exfoliation appears to be due to the generation of steam from water that occurs between cleavage planes of the mineral, and the processed material may have a bulk density as low as 6 lb per cu ft. Exfoliated vermiculite has long been used as a loose-fill insulation but has been employed only in recent years as an aggregate in concrete and plaster. Concrete made from this material usually weighs 25 to 50 lb per cu ft, depending on the quantity of cement used in the mix.

**Perlite:** A relative newcomer to the lightweight aggregate field, perlite is an acidic lava containing 2 to 5 pct water and is capable of expanding with disruptive violence 6 to 10 times its original size when subjected to rapid heating. Deposits are located only in the western areas of the United States, but processing plants are being erected in numerous cities, one as far east as Carnegie, near Pittsburgh, Pa.<sup>5</sup> With some types of furnaces, it is possible to expand suitable perlites to a product weighing as little as 3 lb per cu ft; however, material this light, because of its very friable nature, is generally used as a loose-fill insulation rather than as an aggregate. Perlite that has been expanded to a lesser extent is used to make concretes weighing 35 to 60 lb per cu ft depending upon cement content.

**Naturally Occurring Aggregates:** These include pumice, pumicite, volcanic ash, scoria, and diatomite. Pumice, the most widely used of these materials at present, is of volcanic origin and occurs only in the western regions of the United States. Considerable activity has taken place during the last three years, both in the development of new deposits and the study of techniques of mixing pumice concrete.<sup>6, 7</sup> Pumice is mined in California, New Mexico, Oregon, Utah, Idaho, and Washington. Some of the more progressive producers have set up screening as well as crushing equipment, so that the product can be shipped separately in two or more sizes. The consumer is thereby able to choose the proper ratio of coarse to fine aggregate for each batch of concrete. Ungraded crusher-run pumice has the disadvantage of tending to segregate during shipment so that there is a danger of successive batches of concrete being nonuniform.



Compressive strengths of 2000 to 3000 psi are claimed for properly mixed concrete containing pumice from one source. Higher values are possible when sand is incorporated.<sup>60</sup> It must be kept in mind that deposits of pumice occur in only five states and even within a single deposit it varies considerably in strength and other properties. Therefore, the results of strength tests made on concrete from a specific pumice may not be valid for pumice mined elsewhere.

Pumice weighs approximately 30 lb per cu ft for the coarse sizes and 45 lb per cu ft for fines. One producer has quoted a price range from \$2.07 to \$2.40 per cu yd at his plant, depending upon the quantity ordered.

One disadvantage of pumice is its relatively high water absorption, although some authorities claim that presoaking and proper handling will overcome this difficulty.

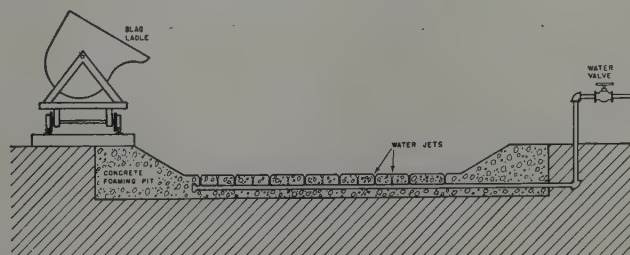
Pumice concrete has been widely used in California, especially in the construction of various buildings in Los Angeles and vicinity. Those of the Pacific Telephone and Telegraph, Prudential Insurance, and General Petroleum companies are examples. In the latter instance, it has been reported that an additional expenditure of \$61,000 for pumice concrete decreased the structural steel cost by \$180,000. Better heat and sound insulation as well as increased resistance to earthquake damage are other possible advantages.

**Expanded Slags:** Expanded or foamed slags, under various trade names, have been marketed for a

group of rotating paddles by means of buffer plates. A controlled stream of water plays upon the moving portions of the machine and the buffer plates. The violent agitation coupled with the action of the water turning to steam causes the slag to expand during its course through the device. The product is crushed and graded in the usual fashion.

Although the machine-foamed aggregates are satisfactory for many uses, apparatus-maintenance costs are appreciable, inasmuch as parts exposed to molten material at 2550°F tend to burn up and must be replaced. In an effort to reduce production costs and still produce an acceptable foamed slag, a British organization is attempting to introduce into the United States a recently developed, modified pit process which is said to have attained a considerable degree of success in England. Pilot-plant tests are now being conducted on American slags of the steel mills in the Pittsburgh area. The new method differs from the old pit process in that only a very small quantity of water is introduced into the pit before pouring the slag therein. Immediately after pouring, a considerably larger amount of water is ejected upwards by means of nozzles embedded in the floor of the pit. The thin layer of water on the floor initiates the foaming of the slag and that fed through the jets at the bottom of the pit completes the operation. The degree of foaming can be controlled by both the total amount and rate at which the water is injected up in the molten mass. It is alleged that 10 to 20 tons of slag can be foamed in less than 2 min. Pits of this type can be built of concrete and

Fig. 1—Schematic drawing of section through concrete slag-foaming pit.



number of years in the United States and even longer in Europe. Usually, these products are made by causing a controlled quantity of water to come in contact with a molten blast-furnace slag, the resulting steam acting as a vesiculating agent. The amount of water used is critical, as an excess will produce a granulated slag which is fragile and unsuited for use as an aggregate whereas an insufficient quantity will bring about the formation of a heavy product.

The simplest method of manufacturing a foamed slag consists in allowing the molten material to run into a pit shaped as a truncated cone with a depth of approximately 15 ft and a width about 20 ft at the top and 15 ft at the bottom. Prior to the introduction of the slag, a quantity of water amounting to roughly one-third of the weight of the molten material is allowed to run into the pit. The expanded mass is subsequently removed by a clamshell bucket and transported to a crusher that produces the desired sizes. In a variation of this method water is sprayed on the stream of molten slag as it is poured into the pit.

One machine that has found considerable use allows the slag stream to fall upon a series of paddles that rotate in a horizontal plane about a central drive shaft. The paddles break the stream into small particles which are deflected to a second and third

located outdoors. If desired, they can be made shallow and with sloping sides so that the foamed material can be removed immediately after processing by means of a mechanical pusher, caterpillar tractor, or bulldozer. The sponsors claim that the aggregate made by this patented process<sup>8</sup> is of a quality comparable to the machine-made type and can be manufactured at a much lower cost. The construction of the pit used to foam the slags is illustrated in fig. 1.

Weights of foamed slags made in the United States are approximately 45 lb per cu ft for the coarse sizes and 55 lb per cu ft for the fine. Prices averaged \$1.88 per cu yd f.o.b. plant in 1948. Because of the low price of foamed slag as compared with other manufactured aggregates, this product finds considerable favor with concrete block manufacturers. Most of the output is used for this purpose.

**Bloated Aggregates:** There are many localities where a demand for lightweight aggregates exists but which are remote from natural deposits or blast-furnace installations producing foamed slags. Therefore, because of high freight rates in relation to the value of the product, these areas are obliged to rely on aggregates produced by treating certain argillaceous materials, such as clays, shales, or slates. These are heated to bring about expansion or bloating. Materials of this type were studied by the Federal Bureau of Mines.



In order for the bloating of an argillaceous material to occur, two conditions must be fulfilled. First the material must become soft and somewhat plastic under the influence of heat, and second, gases must be evolved simultaneously with the existence of the pyroplastic condition. The resultant product is a glassy mass containing numerous small blebs or vesicles. Uniform cells of small size insure maximum aggregate strength and, when separated by cell walls and with a minimum of connecting pores, favor minimum water absorption.

Up to the present time, virtually all commercial production of bloated aggregates has been in the rotary kiln by means of one of two methods. The first method consists of feeding the raw material in the form of fairly coarse sizes, allowing a certain degree of fusion or agglomeration to take place while firing and afterward crushing the expanded product to sizes suitable for aggregate use. The second method involves either crushing or extruding the feed to a size somewhat under that desired in the final product and bloating the material in the kiln as separate particles. Agglomeration is avoided, thereby eliminating final crushing.

Bloated aggregates have been manufactured sporadically and on an experimental basis since 1913, but commercial production did not get underway until 1920 at a plant in Kansas City, Mo. Standard practice for manufacturing aggregate by the first mentioned method, in which the aggregate is crushed after firing, consists in heating shale or other argillaceous material crushed to minus  $1\frac{1}{2}$  or 2 in. size in a 6x60 ft kiln using oil, coal, or natural gas as fuel. The material is discharged as clinker which, after cooling, is crushed and graded, producing an aggregate of moderately high strength suitable for use in monolithic concrete structures. Production of a 6x60 ft rotary kiln for a 24 hr day is of the order of 100 to 150 cu yd, generally closer to the former figure and fuel consumption may be 10 to 16 gal of oil or 250 to 350 lb of coal per cubic yard of aggregate.

An  $8\frac{1}{2}$ x70 ft kiln has been installed recently at a plant near Portland, Ore.<sup>9</sup> With the exception of two similar kilns 175 ft long, which are in operation at Ventura, Calif., this is believed to be the largest-diameter rotary kiln now in operation in the United States for processing a lightweight aggregate. It is rotated at a speed of 2 rpm and pitched at a slope of  $\frac{3}{8}$  in. per ft. Feed is a shale crushed to minus 2 in., and production is alleged to be 268 cu yd per 24 hr day which is far in excess of the reported output of any other single kiln making lightweight aggregate. For instance, the same diameter but longer kilns of the Rocklite plant at Ventura, Calif., are claimed to produce only 175 cu yd of aggregate per day. However, a part of the difference might result from the type of raw material used. Whether or not this will set a trend in kiln sizes for this industry remains to be seen.

Bulk densities of this variety of aggregate are approximately 40 lb per cu ft for the coarse sizes and 60 lb per cu ft for the fine. The present price ranges upward from \$4.00 per cu yd f.o.b. plant.

Among the problems that beset the operators of a plant using a rotary kiln to produce an expanded aggregate are those of preventing the material within the kiln from sticking to the lining or agglomerating into balls so large as to make their removal difficult. It is unusual to find a natural argillaceous material that will bloat and yield a

good lightweight aggregate without showing some tendency to adhere either to itself or the kiln lining or both. Although some materials will bloat over a wider temperature range before clinkering or sticking than others, the problem is ever present. The trouble is largely unavoidable, because the charge must be relatively soft and plastic and gas must be evolving before vesiculation can take place. Ring formation within the kiln may be aggravated by the deposition of ash from powdered coal, in many localities the most economical type of fuel. In an effort to minimize excessive agglomeration and sticking, a number of installations are equipped with two burners. One burner is directed along the axis of the kiln and heats the material nearly to the bloating temperature while the other plays directly upon the charge within a few feet of the discharge point. Thus, the second burner actually raises the material up to bloating temperature. The successful firing of a material, such as a raw shale, in a rotary kiln requires a high degree of skill on the part of the operator, who must keep the temperature within a relatively narrow range. Variation of temperature will produce a heavy, underbloomed aggregate if firing temperatures are too low or will fill up the kiln with a sticky, gummy mass presenting problems of removal if the temperature is allowed to get too high.

Various schemes have been suggested to avoid processing troubles caused by the adhesive tendencies of bloating materials as well as to produce an aggregate consisting of discrete, well-rounded particles, each having a uniformly cellular interior sealed off by a hard impervious surface. Firing can be accomplished without causing the particles to agglomerate on their way through the kiln. Subsequent crushing thereby will be rendered unnecessary. Of these proposed methods, most have been abandoned because of high production costs.

Considerable work was done on a patented process<sup>10, 11</sup> in southern California, nearly 20 years ago whereby clay was pugged with water and heavy fuel oil. The latter functioned as a bloating agent. The plastic mass was extruded through a multiple die, and the "macaroni" was cut into short lengths to make pellets. Before firing, the particles were coated with a powdered refractory material having a higher fusing point than the clay. Upon traveling through the kiln each particle was able to maintain its separate identity and yielded spherical or rounded shapes with sealed outer surfaces.

At present, a producer of lightweight aggregates in Strawn, Texas, is known to be making a similarly pelletized product, although only on a limited scale. During World War II, a special pelletized lightweight aggregate was made in a plant at Ellenton, Fla. Three grades of aggregate were produced, as follows:  $-\frac{5}{8} + \frac{3}{8}$  in.,  $-\frac{3}{8} + \frac{1}{4}$  in., and  $-\frac{1}{4}$  in. These materials were shipped to Tampa, Fla., for use in constructing concrete ships. However, it is believed that production costs in the war-built plant were too high to justify peacetime operation.

Another process<sup>12</sup> along similar lines has been attempted in which a crushed diatomaceous shale has been fed to a kiln simultaneously with a small quantity of powdered limestone or dolomite. Upon entering the hot zone, the surfaces of shale fragments become sticky and pick up a coating of lime which prevents agglomeration. A portion of lime also may react with the silica present in the shale, thereby forming a shell of calcium silicate on the



surface of each particle. This process is believed to have had a commercial trial.

Another lightweight-aggregate plant which functioned during World War II to supply a high-grade lightweight aggregate for concrete-ship construction was that at Santa Maria, Calif. Their product is the commercial aggregate sold on the market as "airox." The raw material used to produce airox is a diatomaceous earth with some clay and has a high-oil content that can be reclaimed by heating or distillation. To expel the excess oil and obtain a satisfactory lightweight aggregate, the crushed raw material was treated during the war in two rotary kilns operated in series. The hydrocarbon content of the material was burned out in the first kiln, while the sintering or incipient fusion necessary to produce a strong aggregate was accomplished in the second kiln. The costs of this method of handling the aggregate were high and warranted only on a wartime basis.

Since the war, the kiln treatment has been discontinued and the equipment dismantled and sold. However, by merely placing the suitably crushed

workability, and low-water absorption have been made by the established rotary-kiln processes. The relatively high-production cost of aggregates of this type prevents them from finding wide acceptance by the manufacturers of masonry units, especially since certain characteristics, as for example workability, are not too important for this use. Therefore, at present, more efficient types of furnaces other than the rotary kiln are needed to produce medium-quality aggregates for utilization in concrete blocks.

Development has proceeded in the direction of sintering machines in which the raw material and a solid fuel are placed on grates provided with a fan for supplying air for combustion of the added fuel. Machines for this purpose have taken the form of endless-chain, tilting or rotating annular grates. It would appear that a sintering machine, if it could be made to operate successfully, might possess definite advantages. These sintering machines could then be used in place of rotary kilns for the manufacture of a medium-grade, low-cost aggregate suitable for ordinary masonry blocks. Apparent advantages are:

**Fig. 2—Large lightweight aggregate plant constructed to treat coal wastes in Pennsylvania anthracite region.**



diatomite in heaps or windrows and initiating combustion of the oil contained in the material by lighting with a torch, an acceptable lightweight aggregate can still be produced. Control of the product is obtained by the method of preparing the piles or heaps for burning and by watching the size gradation. Actual burning requires up to 48 hr, depending on the handling technique and results desired. It is claimed that the finished product can be placed in the finished-product bins at the plant on a basis competitive with crushed pumice.

The operators of the plant located at Ventura, Calif., near Los Angeles, have been fortunate in discovering a deposit of soft shale that appears to be unique insofar as its ability to undergo bloating in a rotary kiln without sticking or agglomerating is concerned. The raw shale is crushed and screened to three sizes each of which is fired separately. The material, in traveling through the hot zone, becomes slightly sticky, but the particles maintain their individuality upon being discharged from the kiln and are not in an agglomerated condition. Furnacing this material requires no prior processing other than crushing and grading to size, and it is believed that this installation is the only one today in the United States in which a raw, untreated shale is bloated in a rotary kiln without having to be crushed after firing. The product consists of well-rounded particles, each of which has a cellular interior sealed off by a smooth outer skin. This is true of the fine sizes, as well as the coarse. As a result, an aggregate is produced having many desirable characteristics, such as high strength, low-water absorption, and good workability. Bulk densities of this aggregate are about 40 lb per cu ft for the coarse and 50 lb per cu ft for the fine sizes.

**Sintered Aggregates:** Excellent aggregates exhibiting the qualities of high strength, light weight, good

1. The use of a low-cost fuel is permissible. A high-ash coal that might cause ring formation when burned in a rotary kiln might be used to better advantage in a sintering machine. Certain shales exist in which the carbon content is high enough to provide part if not all of the fuel requirements.

2. Agglomeration and fusing of the material being fired would present less of a problem. Shales and slates with small ranges between bloating and fusing temperatures could be handled with more success in a sintering machine than in a rotary kiln. Thus a wider variety of raw materials would become available for conversion into lightweight aggregates.

3. For a given aggregate capacity, a sintering machine would require less capital outlay. Kilns and their accessories are relatively expensive and for a 6x60 ft installation output is limited to a maximum of approximately 150 cu yd per 24 hr.

Twenty years ago there were reports in the literature on the experimental use of sintering machines for manufacturing aggregate.<sup>13, 14</sup> An aggregate named "Lytag" was to have been placed on the market in the early thirties, although it never appeared, possibly because of cessation of building activity at that time. Mention is made of a wide variety of materials that could be used to make this aggregate, including clays, shales, slates, schists, and refuse from the coal mining industry. Processing included crushing the raw material to minus 4 mesh, blending with granulated coal and adding a small quantity of water. The mixture was then discharged in a permeable condition on the traveling grate of a machine similar to the Dwight-Lloyd continuous-grate sintering apparatus. After the charge was ignited, the air for combustion was drawn downward by means of a suction fan. It is alleged that the character of the sinter could be varied by ad-



justing the fuel:clay ratio, the quantity of air drawn through the charge, depth of charge, and rate of travel of the grate. Other patents<sup>15, 16, 17, 18</sup> also disclose the use of a continuous chain grate for the manufacture of lightweight aggregates from argillaceous materials.

Several patents<sup>19</sup> were issued in the early thirties to John E. Greenwalt on methods and apparatus for producing lightweight aggregates by means of a stationary sintering pan that could be tilted to discharge the product.

Passing on to postwar times, three new developments have taken place in traveling-grate machines. The first of these, and the only one as yet in continuous commercial operation, is in the anthracite region of Pennsylvania and was specifically con-

being installed at Marietta, Ohio,<sup>21</sup> and another at Detroit, Mich. The continuous grate is about 50 ft long and 5 ft wide and runs between sprockets approximately 5 ft in diam. An idea of the general type of grate used can be had by referring to the sketch shown in fig. 3. A simple arrangement for using this machine in a lightweight-aggregate plant, together with the other essential units, is shown schematically in fig. 4. This machine differs principally from the one just described for treating coal-mine wastes in that air for combustion is drawn downward through the bed of material on the grate by means of a rectangular suction box connected to a fan driven by a 350 hp motor. Another dissimilarity is that the function of this machine is to form a lightweight material not so much by a bloat-

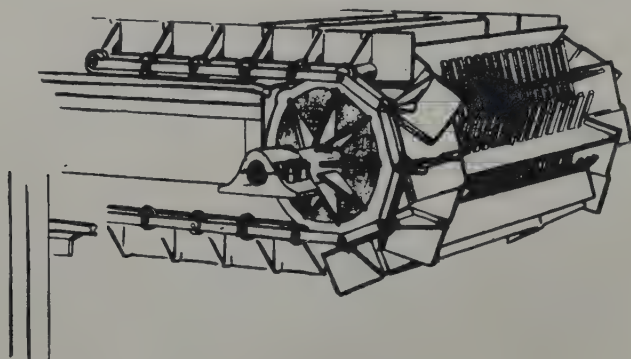


Fig. 3—Rough sketch of traveling grate type of sintering machine illustrating heavy construction of parts.

structed for producing lightweight aggregate from coal-mining wastes. Without doubt, this installation is one of the most interesting and most important developments in the industry.

Fig. 2 is a photograph of the lightweight aggregate plant during construction. Evidence is given in the figure that a large capital investment is involved in plant facilities. The plant is between the towns of Tamaqua and Coaldale in the anthracite coal-mining section of Pennsylvania. Several units are installed at this plant, and a production of 125,000 tons per year is anticipated.<sup>20</sup> The raw material used is a metamorphic carbonaceous shale mined along with anthracite and separated therefrom in a coal-preparation plant above ground. Each bloating furnace consists of an endless traveling grate, said to be the largest ever constructed, operating very much like a chain-grate stoker. An ignition mixture containing anthracite, which is used to promote the bloating process, is fed onto the grate and passes through an ignition chamber. Just beyond the chamber the shale is fed onto the layer of ignited fuel. As it passes through the furnace, combustion of the carbonaceous matter produces a temperature of approximately 2800°F in the bed of material. The particles become softened and are bloated by the entrapment of gases resulting from the burning of the carbonaceous content of the shale itself as well as the release of certain other volatile constituents. Incipient fusion converts the material into a form said to be chemically inert and structurally strong. The product emerges from the furnace in the form of a continuous slab which is reduced to minus 2½ in. size in a primary crusher and then stockpiled. Sizes suitable for use as aggregate are produced in a secondary crushing and grading operation. The final product is said to weigh about 40 lb per cu ft for the coarse and 62 lb per cu ft for the fine sizes.

Another type of endless-grate sintering device is being constructed and sold by a manufacturer of concrete block-making machinery. One unit is now

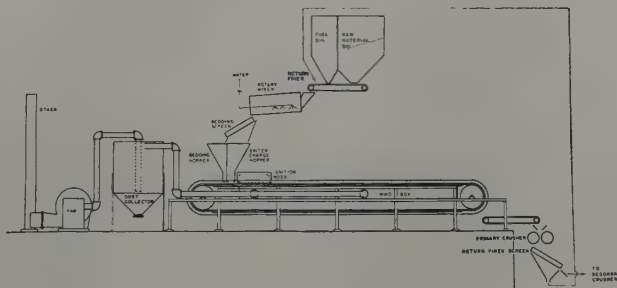
ing action, although this may occur to a degree with some materials, but to agglomerate small granules of clay or shale that go to make up the feed so that the resulting aggregate derives its lightness mainly from the air spaces remaining between the fused particles. Although similar in many respects to the conventional Dwight-Lloyd sintering machine, this device differs by having the grate divided into rectangular compartments by walls transverse of the machine and perpendicular to the grate bars. The grate bars themselves are of a rather ingenious construction being interlaced when the grate is positioned above the suction box but breaking apart when passing over the sprocket at the discharge end and performing simultaneously the functions of discharging the cake and cleaning themselves. In operation, a clay, shale, or other suitable raw material is crushed to minus ⅛ in., sized, and mixed with about 7 pct of finely divided solid fuel such as coal, lignite, or cinders along with enough water to give bulk to the mixture and obtain a satisfactory consistency for sintering. The charge is then passed through a screen that allows the larger particles to fall into one hopper positioned above the traveling grate and the fines to fall into a second hopper behind the first. Material from the first hopper falls onto the surface of the grate and forms a bedding approximately 1 in. thick for a subsequent 8-in. layer of finer material that falls from the second hopper. If desired, material that has previously been sintered may be used for the bedding layer. The charge is then conveyed under an ignition hood where a flame is momentarily applied to the top surface as a blast of air is drawn downward through the grate by means of the fan. The ignited charge continues to advance over the suction box as the burning zone propagates downwards. The rate at which the grate advances is controlled so that the charge is completely sintered by the time it is discharged. The finished cake is crushed to minus 4 in. size, and the minus ¾ in. material resulting from this preliminary crushing



is returned to the machine for further sintering. The remainder is crushed and graded to aggregate size. A production capacity of 30 cu yd of aggregate per hour is claimed for this machine.

A third variety of traveling-grate machine is being marketed in the United States by another manufacturer specializing in concrete-block machinery. Possible future installations may be at Denver, Colo., and Bridgeport, Conn. The prototype of this apparatus is in the New York City area where experimental work has been done with fly ash as a raw material.<sup>22, 23, 24</sup> In appearance, the machine is similar to the Dwight-Lloyd circular-grate type sintering machine first patented in 1909 (U. S. Patent 916,392). The device consists of a series of hinged pans or grates disposed adjacently to one another in an annular configuration, rotatable about its axis. Beneath the grates is positioned a C-shaped suction box that draws combustion-supporting air through the material being sintered. The fly ash, which is said to contain its own fuel supply, is mixed with water in a continuous mixer and flows into the sintering pans, which are continuously rotated. However, before the sintering pans are filled, a 1 in.

**Fig. 4—Schematic arrangement of sintering machine for making lightweight concrete aggregate.**



layer of previously sintered and crushed material is spread over the grate to form a permeable bed. About 5 in. of raw material is placed on top of the bedding. After a section is filled, it is ignited by a burner for 1 min, afterward passing over the suction zone for 18 min while combustion is maintained. After sintering is completed, the grate retracts, and the clinker falls out whereupon it is crushed and graded. The grate has by this time rotated through 360° and is ready for refilling and repetition of the cycle. Provision can be made for a preliminary crushing, whereby the fine, insufficiently burned material can be returned for further sintering. It is reported that this machine is adaptable for processing clays, shales, and numerous other materials, as well as fly ash. Solid fuel to the extent of from 4 to 10 pct of the raw material must be added when sintering noncarbon-containing substances. Output for this machine is claimed to be 150 to 200 cu yd of aggregate per 8 hr day.

## References

- <sup>1</sup>Construction Branch, National Housing Agency. *Lightweight Aggregates for Concrete*. Office of Housing Expeditor, Washington, D. C.
- <sup>2</sup>J. E. Conley, Hewitt Wilson, T. A. Klinefelter and others: *Production of Lightweight Concrete Aggregates from Clays, Shales, Slates and Other Materials*. Bur. of Mines. R.I. 4401 (1948) 121 pp.
- <sup>3</sup>W. M. Avery: *Lightweight Ready-Mixed Concrete Reduces Dead Load 8,000 Tons*. *Pit and Quarry*. (May 1948) 41, 189, 190, 192.
- <sup>4</sup>L. A. Castell: *Lightweight Insulating Concretes*. *Rock Products*. (Feb. 1949) 52, 153-155, 164, 165.
- <sup>5</sup>Pittsburgh Firm Begins Manufacture of Perlite for Masonry Units. *Concrete*. (Aug. 1948) 56, 32.
- <sup>6</sup>John W. Shaver, A. V. Roemisch, Walter K. Wagner and Jack E. Counts: *The Story of Pumice*. *Concrete*. 56 (a) Aug. 1948, pp. 3-7, 39; (b) pp. 10-12; (c) Sept. 1948, pp. 20, 22, 24, 35; (d) Oct. 1948, pp. 8-10, 34; (e) Nov. 1948, pp. 20-22, 35; (f) Dec. 1948, pp. 3-5, 37.
- <sup>7</sup>W. K. Wagner: *Design and Control of Pumice Concrete Mixes*. *Rock Products*. (Sept. 1948) 51, 125-127, 136, 137, 138.
- <sup>8</sup>Marcel Gallai-Hatchard: *Method for the Production of Porous Slags or Other Smelts*. U. S. Pat. 2,443,103 (June 8, 1948).
- <sup>9</sup>Harry F. Utley: *Pacific Northwest Gets Adequate Supply of Lightweight Aggregate*. *Pit and Quarry*. (Apr. 1949) 41, 93, 94, 99.
- <sup>10</sup>Charles Knox Harding: *Process of Producing Lightweight Aggregate and Product Thereof*. U. S. Pat. 2,015,381 (Sept. 24, 1935).
- <sup>11</sup>Sydney T. Evanstad: *Art of Making Lightweight Concrete*. U. S. Pat. 2,199,046 (Apr. 30, 1940).
- <sup>12</sup>Carl M. Price: *Process of Making Aggregates*. U. S. Pat. 2,112,380 (Mar. 29, 1938).
- <sup>13</sup>Frank A. Glass: *Lytag—A Lightweight Aggregate Made by Sintering Process*. *Brick and Clay Record*. (Dec. 4, 1928) pp. 829-830; *Process of Producing Hard, Cellular, Clay Products*. U. S. Pat. 1,538,482 (May 19, 1925).
- <sup>14</sup>H. Herbert Hughes: *Scope of the Lightweight Aggregates Industry*. *Trans. AIME* (1931) 96, 187-203.
- <sup>15</sup>Eben Rodgers, Jr., and George A. Duncan, Jr.: *Method of Making Light Porous Materials*. U. S. Pat. 1,842,801 (Jan. 26, 1932).
- <sup>16</sup>C. E. Cooper and H. S. Gilbertson: *Method of Producing Lightweight Aggregates from Breaker Waste and the Like*. U. S. Pat. 1,896,884 (Feb. 7, 1933).
- <sup>17</sup>Thorne E. Lloyd: *Method of and Apparatus for Producing Cellular Aggregate*. U. S. Pat. 1,932,415 (Oct. 31, 1933).
- <sup>18</sup>Maxwell M. Upson: *Method of Treating Refuse and Making Building Material Therefrom*. U. S. Pat. 1,936,810 (Nov. 28, 1933).
- <sup>19</sup>John E. Greenwalt: *Method of Manufacturing Artificial Aggregate for Mortars and Concretes*. U. S. Pat. 1,786,713 (Dec. 30, 1930); *Method of Sintering Clay*. U. S. Pat. 1,786,714 (Dec. 30, 1930); and *Method of Disposing of Sewage Sludge*. U. S. Pat. 1,895,159 (Jan. 24, 1933).
- <sup>20</sup>*Lightweight Aggregate Produced in Modern Plant by Lehigh Materials Co.* *Pit and Quarry*. (Aug. 1948) 41, 137.
- <sup>21</sup>Marietta Concrete Corporation Builds \$260,000 Aglite Plant. *Pit and Quarry*. (April 1949) 41, 128.
- <sup>22</sup>L. Gelbman: *Sintering Apparatus*. U. S. Pat. 2,403,433 (July 9, 1946).
- <sup>23</sup>First Sinter-Lite Plant is Making Lightweight Aggregate from Fly Ash. *Pit and Quarry*. (Jan. 1947) 39, 199-200.
- <sup>24</sup>R. F. Leftwich: *New Lightweight Aggregate from Fly Ash*. *Concrete*. (Jan. 1946) 54, 14-15, 39.

# Western Phosphate Industry

by Roscoe E. Bell

An economic appraisal of the western phosphate industrial potentials made between 1945 and 1948 revealed possibilities for a considerable expansion in the production of high analysis phosphate fertilizer. The most significant facts are: (1) a rapidly increasing and a much larger potential market which might be reached from the western deposits, (2) the existence of large undeveloped deposits of phosphate rock, (3) potential energy from sulphuric acid, coal, or hydroelectric power in the general vicinity of the deposits. The challenge to the mining and metallurgical engineering profession is in the economical development of phosphate mines and in improving processes for utilization of the western ores and the recovery of byproducts to make the expanded western industry satisfactorily compete with the industry in the East and South in meeting the fertilizer needs of the Midwest and Far West.

**B**ETWEEN 1945 and 1948 the author made studies of the western phosphate industry and its potentialities. These included an appraisal of the opportunities for western development of the industry, studies of production costs, of market potentials, of transportation economics, and of industry location.

This paper will be directed toward pointing up the factors which have an important bearing on the establishment of fertilizer plants in the western area.

**Industry Location Factors:** There are, of course, an infinite number of possible organizational setups which might require consideration. However, the

ROSCOE E. BELL, formerly Coordinator of the Western Phosphate Program, U. S. Department of the Interior, is now Associate Director, Bureau of Land Management, U. S. Department of the Interior, Washington, D. C.

AIME San Francisco Meeting, February 1949.

TP 2667 H. Discussion (2 copies) may be sent to Transactions AIME before May 30, 1950. Manuscript received March 3, 1949.

phosphate industry is simple enough so that studies of the most probable alternatives can be made conveniently. The salient factors which require special consideration in such a study are: (1) the location of the raw materials, (2) the location, size, and character of the present and potential markets, (3) the location of energy sources and materials required for fertilizer processing, (4) the location and capacity of present manufacturing facilities, (5) the cost of assembling the materials and energy needed in manufacturing, and (6) the cost of distributing the manufactured product.

**Raw Material Supplies:** Rock phosphate is the raw material for the manufacturing of phosphate fertilizer. In the United States the important known deposits of this material are located in three general areas (fig. 1). The western deposits, comprising about 60 pct of the nation's supply, are in the four-state area—Idaho, Wyoming, Utah, and Montana. The Florida deposits comprise roughly 38 pct of the national supply. The Tennessee deposits, comprising about 2 pct of the national supply, have been the most important source until recent years. The western deposits, which are virtually undeveloped, have not been used to any extent to meet the nation's need for phosphate rock and phosphate fertilizer.

**Fertilizer Markets:** The large national markets for fertilizer are in the midwestern and eastern United

States (fig. 2). The principal far-western markets are along the Pacific Coast, especially in California. Until recent years the western markets have been extremely limited, whereas the markets in the eastern United States have long been established and manufacturing facilities have been provided to meet these needs (fig. 3). In recent years the western markets have expanded very rapidly, continuing to grow throughout the war years and postwar period at a similar rate. The question always arises as to whether this is a temporary war-swollen market which will shrink back to its original size as soon as the present agricultural price levels drop. In order to answer this question consideration must be given to the character of the market (fig. 4). I have considered seventeen western and midwestern states in the western market area where transportation costs from the western deposits are equal to or lower than those from Florida. They include all states west of a line drawn through Chicago, Kansas City and the western boundary of New Mexico. In the prewar period this was a small market totaling only 37,000 tons of  $P_2O_5$  annually. The 1946 total for this same area was approximately 200,000 tons. Considerably more fertilizer would have been used in 1946 had adequate supplies been available.

Estimates of future markets are difficult to make, but we do have some indices that are useful. The

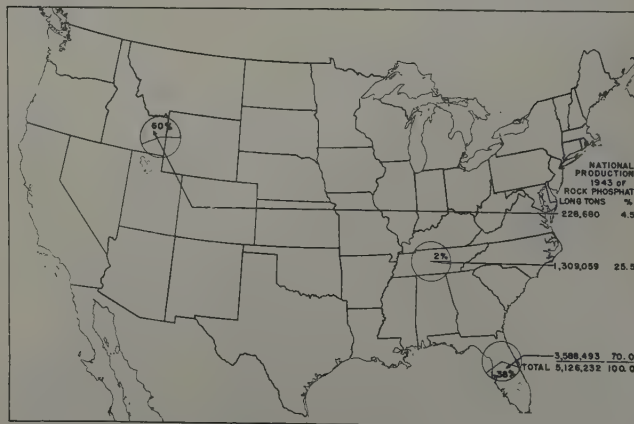


Fig. 1—Major deposits of phosphate rock in the United States.

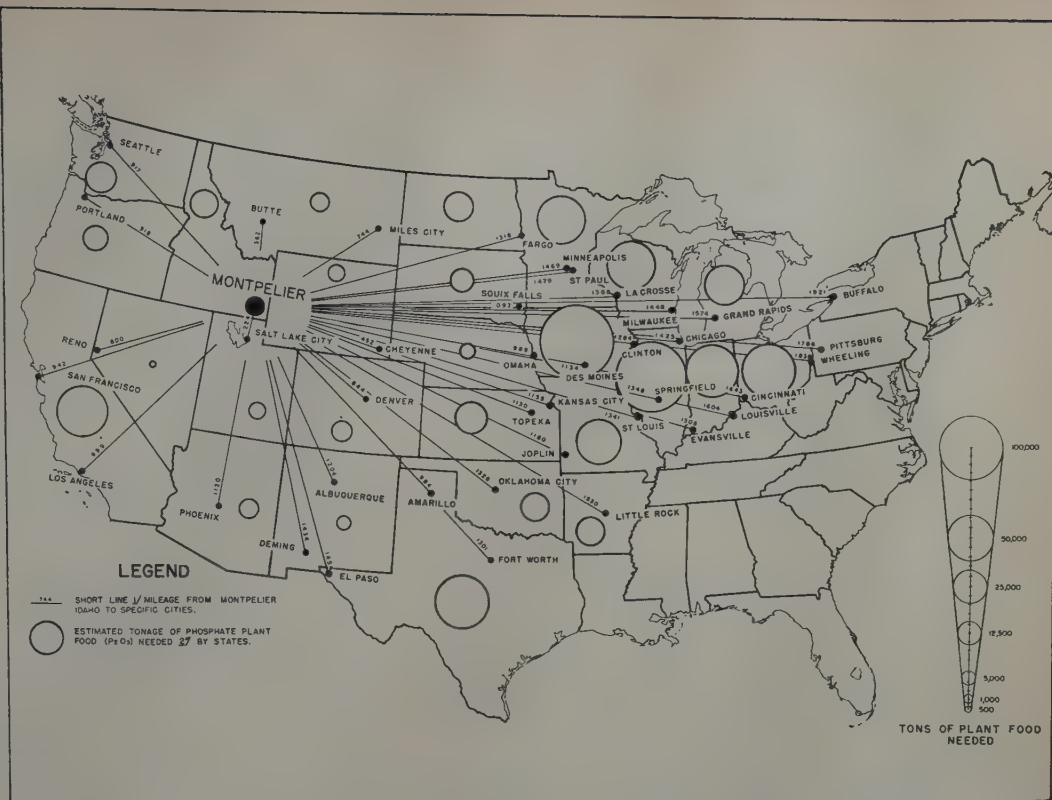
Source: Mansfield: *Ind. and Eng. Chem.* (1942) 34, 9-12; and Phosphate Fertilizer Consumption, Production Capacity, and Reserves in Western United States. U.S.D.A. Int. Bur. Comm. on Postwar Prog. 1944; U.S. Dept. of the Interior, Bonneville Power Admin., Div. of Ind. and Res. Dev. 1946.



**Fig. 2—Rail mileages from Montpelier, Idaho, to important potential markets for western phosphate products.**

<sup>1</sup> Southwestern  
Traffic Manual, St.  
Louis S.W. Ry.  
Lines, Nat'l Ry.  
Publication Co.  
<sup>2</sup> State Production  
Adjustment Com-  
mittees, Prod. and  
Marketing Admin.,  
U.S.D.A. 1947.

In these cases annual soil phosphate removal by crops was substituted. U.S. Dept. of Interior, Western Phosphate Program, Economic Studies by Bonneville Power Admin., June 1947.



agronomist knows that phosphoric acid is present in the soil in very small quantities, and that it is removed by crops in important but varying amounts, and is frequently a critical plant nutrient element. It is especially critical in the production of the legume crops essential to good crop rotation and soil conserving cropping systems. Phosphoric acid once removed from the soil is replaced primarily in the form of commercial fertilizer. The removal of  $P_2O_5$  by crops can be calculated and can be fairly accurately estimated over broad areas. This, however, does not give an accurate index of the critical shortages in the soil or the areas where fertilizer application will pay. It does give a clew as to the *ultimate need* for fertilizer.

**Soil Phosphate Requirements:** The agronomists of the West have done considerable experimental work on fertilizer use and know something about the needs of particular soils. The farmers of the West have in recent years, under the stimulation of the agricultural conservation program and the trials established by the colleges and the fertilizer com-

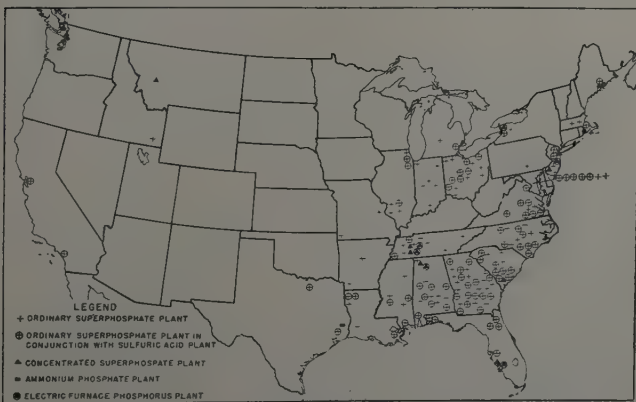
panies, tried out fertilizer in many areas. Estimates of fertilizer needs have been compiled by counties and states throughout the United States and these give some index of future fertilizer needs. These figures are, with all of their limitations, interesting and significant although probably not more than 75 pct accurate.

Soil scientists have made analyses of many soils both by chemical and biological tests. However, these analyses are notoriously misleading in some cases. This is especially true in the wide variety of soils which occur in the western United States.

All of the above indices of fertilizer need point to a continued increase in the fertilizer needs of the western soils.

The estimated "need" for fertilizer may be used as an indication of possible future market requirements, table I. It is apparent that in 1946 consumption of  $P_2O_5$  was only about half of the estimated need in the western area. The estimated need was roughly half of that which is removed annually in crops. It would, therefore, appear that the area is one in which added fertilizer use is very probable if not inevitable in future years. The rapid increase in fertilizer use in this area has been stimulated partly by the high crop price levels. However, it has been associated with a decrease of general production throughout the area where fertilizers have not been applied. In much of the area the high fertility of the virgin soils is nearly exhausted. It is also significant to note that in the older farming areas of the United States, such as the southwestern states, fertilizer application must be double or treble the removal by crops if productivity is to be maintained at high levels.

This rapid rise in the consumption of fertilizer has made it impossible for western manufacturers to produce all of the fertilizer required. Consequently, there has been importation from eastern plants to meet the western needs (fig. 5). Actually



**Fig. 3—Location of phosphate fertilizer plants in United States, 1947.**

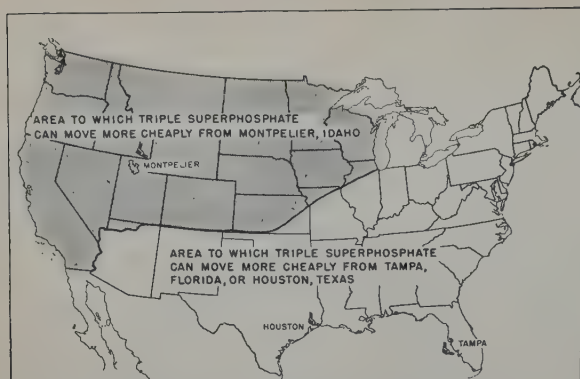


Fig. 4—Approximate trade areas for western- and southern-produced triple superphosphate.

Freight rate schedules from Montpelier based on those in effect June 30, 1946, U.S. Dept. of Interior, Western Phosphate Fertilizer Program, Economic Studies by Bonneville Power Admin., June 1947.

Table I. Phosphate Fertilizer Consumption, Need, Removal by Crops, and Manufacturing Capacity

	United States	17 Western States	4 North Central States	27 Eastern and Southern States
<b>Fertilizer Consumption (Tons <math>P_2O_5</math>)</b>				
1935-39	703,926	37,338	89,621	576,967
1944	1,288,302	135,172	198,745	954,385
1945	1,338,062	170,106	200,006	967,950
1946	1,534,704	199,682	254,378	1,080,644
1947	1,716,897	239,021	294,916	1,182,960
<b>Fertilizer "Need" (PMA Estimates) (Tons <math>P_2O_5</math>)</b>				
	2,863,296	457,142	880,449	1,525,705
<b>Fertilizer Removal by Harvested Crops (Tons <math>P_2O_5</math>)</b>				
	1,699,064	869,829	316,348	512,887
<b>Fertilizer Manufacturing Capacity (Tons <math>P_2O_5</math>)</b>				
	1,987,000	85,000	200,000 (est.)	1,702,000

some fertilizer originating in Florida, processed at Baltimore, and shipped to mixing plants in Portland, Oregon, finds its way into the markets of Idaho and Montana.

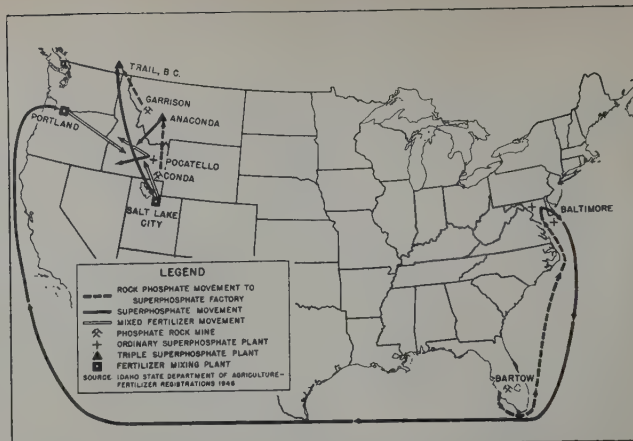
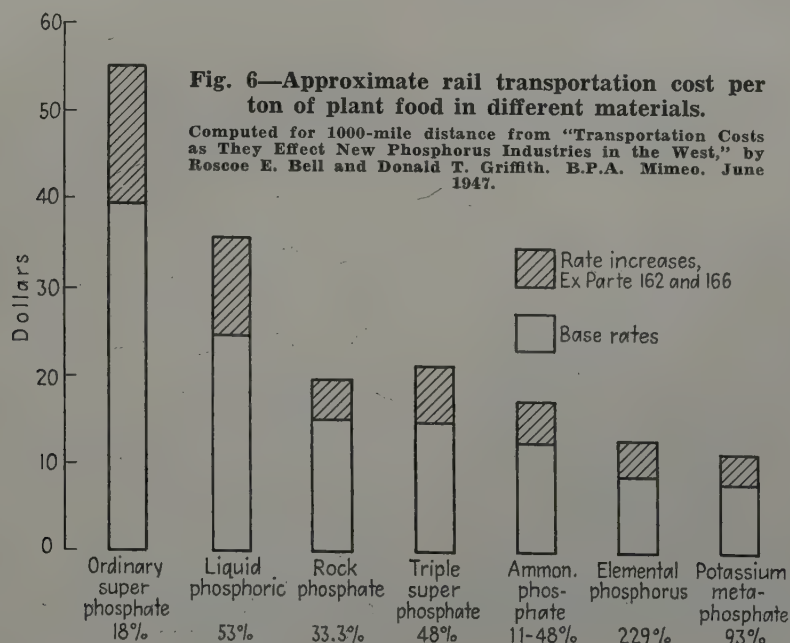


Fig. 5—Movement of fertilizers containing phosphate into Idaho, 1946.

U. S. Dept. of Interior, Western Phosphate Fertilizer Program, Economic Studies by Bonneville Power Admin., June 1947.

The existence of the nation's greatest supply of raw materials, together with the nation's most rapidly growing market in the western United States, has naturally attracted attention as to the possibilities of the future growth of the industry in the West. Apparently two important factors basic to the industry are present in the western area and if the other factors are present or can be provided, production can be economically expanded.

**Manufacturing Processes:** Phosphate rock may be converted into soluble phosphate fertilizer by treatment with sulphuric acid or by smelting in an electric or blast furnace. The sulphuric acid treatment may result in the production of normal or ordinary superphosphate which contains 18 to 20 pct  $P_2O_5$ , or triple superphosphate may be produced which contains 40 to 50 pct  $P_2O_5$ . The electric or blast-furnace processes result in the production of phosphorus which may be converted to phosphoric acid and treble superphosphate, ammonium phosphate or other derivatives.

In the western area, within comparatively short distances from the phosphate deposits, there are potential sources of reagents or energy for the production of phosphates and fertilizers by any one of the three methods. Metalliferous sulphide ores



provide byproducts which can be used for the production of sulphuric acid. The Pacific Northwest is the nation's area of greatest potential low cost hydroelectric energy. Coking coal is also present in the intermountain area of Wyoming and Utah relatively close to phosphate deposits. It is, of course, possible to ship rock phosphate from the West to the midwestern United States for conversion to fertilizer. The location of the industry is dependent upon the relative costs of combination of the factors of production and the transportation to market.

It is apparent that the large markets are a rather long distance from the western phosphate deposits. Distances of 900 to 1500 miles from the western phosphate area to important market centers are common. Thus transportation costs become an important factor in reaching the markets economically, and therefore the more concentrated fertilizers have greater transportation advantage. In looking toward future markets it is essential to produce the type of a product which can reach the consumer at a low price (fig. 6). Thus the comparison of the delivered cost of ordinary superphosphate and treble superphosphate shows that in the western and midwestern markets the delivered cost of treble superphosphate is considerably lower than ordinary superphosphate per unit of plant food. It has frequently been pointed out that treble superphosphate is more expensive to produce than ordinary super-

**Table II. Effect of Fertilizer Concentration on Retail Cost**

Cost Item	Cost per ton of P <sub>2</sub> O <sub>5</sub>	
	Ordinary <sup>a</sup> Super- phosphate 18 pct P <sub>2</sub> O <sub>5</sub>	Treble <sup>b</sup> Super- phosphate 43 pct P <sub>2</sub> O <sub>5</sub>
Fertilizer f.o.b. factory	\$ 32.00	\$ 36.50
Bags and bagging at \$3.00 per ton	16.70	7.00
Freight at \$5.80 per ton	32.00	13.50
Sales at \$4.00 per ton	22.00	9.00
Total	\$152.70	\$116.00

<sup>a</sup> O.P.A. ceiling Pocatello, Idaho MP R 205.

<sup>b</sup> O.P.A. ceiling for treble, minus 7.62 freight allowance apparently included in ceiling delivered to jobber. MP R 205.

phosphate. There is often a big difference between manufactured cost and delivered cost, table II. The various items which come in between the manufacturer and the consumer are more related to bulk than to plant food content, thus the lower production cost of ordinary superphosphate is more than offset by the higher delivery costs, as is shown by a comparison of delivered costs of plant food at selected points in the western trade area, table III.

**Table III. Retail Ceiling Price Per Ton of Phosphate Plant Food (P<sub>2</sub>O<sub>5</sub>) in Ordinary Superphosphate and Triple Superphosphate, at Various Points, 1945**

City, State	Ordinary Super- phosphate	Triple Super- phosphate
Portland, Ore.	\$175.00	\$127.21
Pocatello, Ida.	169.45	120.93
San Francisco, Calif.	154.17	120.93
Fargo, N. D.	166.11	134.30
Minneapolis, Minn.	158.33	131.05
Madison, Wis.	144.45	129.67
Kansas City, Mo.	156.11	124.78
Des Moines, Iowa	158.33	129.55
Chicago, Ill.	138.89	128.00

Source: O.P.A. Maximum Price Regulation 135, 2nd Rev. Nov. 21, 1945.

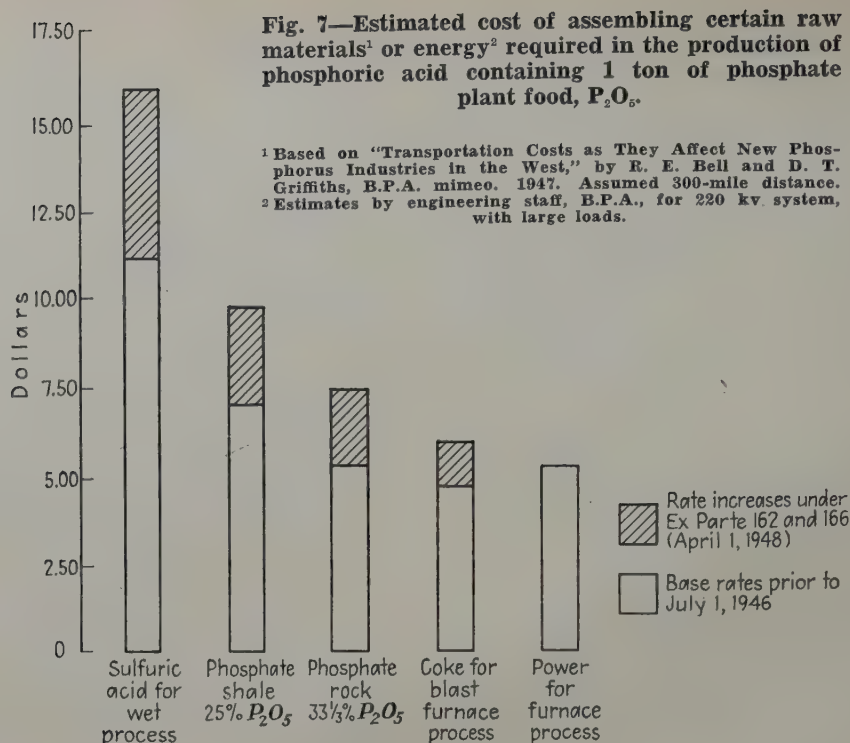
There is another aspect of the future fertilizer market which is extremely significant. More than half of the fertilizer used in this country goes into mixed fertilizers which are produced by mixing the so-called "simples" such as superphosphate, muriate of potash, and ammonium sulphate. The trend in manufacture of mixed fertilizers is toward higher analysis as a means of delivering the product to the farmer at a lower cost per unit of plant food. High analysis mixed fertilizers cannot be produced with low analysis "simples" such as ordinary superphosphate. The manufacturing capacity of this country available for the production of triple superphosphate is far below the amount of phosphate fertilizer used in mixed fertilizers. Thus the big market for future development is in the production of concentrated fertilizers. This is especially true in the West where shipping distances are great and where an industry pattern of ordinary superphosphate fertilizer plants has not already been established.

**Possible Locations of Factories:** Granting the need for additional western manufacturing capacity for high analysis fertilizer, what are the most probable alternatives as to type and location of plants? The following deserve some consideration: (1) sulphuric acid process, manufacturing near the phosphate mines, (2) sulphuric acid process manufacturing near smelters where sulphuric acid manufacturing plant capacity can be established, (3) electric furnace processing near the hydroelectric plants on the Columbia River, (4) electric furnace processing near the phosphate mines, and (5) blast furnace processing near the phosphate mines.

The distribution of western phosphate rock deposits, present and potential sulphuric acid plants, and hydroelectric facilities would require shipping of rock or sulphuric acid or transmission of electricity about 300 miles to assemble the factors of production (fig. 7). A comparison may be made of the economy of transportation of sulphuric acid and rock phosphate or transmission of electric energy over these distances. It is apparent that it is more economical to transport rock phosphate to the acid than to transport acid to the rock. It is, also, more economical to transport sulphur than to transport acid and considerably more economical to transmit electricity than to transport either rock or acid. An important advantage of location of the processing plant close to the phosphate mines is that it provides an opportunity to utilize low-grade phosphate rock which otherwise could not be used economically.

It is evident then that the alternatives which deserve the most consideration for western plant establishment are the sulphuric acid process at one of the smelters such as Salt Lake City, Anaconda, or in the Coeur d'Alene district in northern Idaho, an electric furnace plant or a blast furnace plant near the phosphate mines.

In the midwestern market western phosphate would be in competition with Florida phosphate fertilizer produced by the wet process using Texas or Louisiana sulphur as a source of sulphuric acid. Another alternative to consider is the possibilities of manufacturing concentrated fertilizer at a midwestern point using Florida (or western) rock and Texas or Louisiana sulphur. Western phosphate rock cannot reach the midwestern market as cheaply as Florida rock because it is more expensive to mine, it may require more expensive preparatory treat-



ment, and it does not have the advantage of water transportation that Florida rock has.

**Cost Comparisons:** A comparison of possible costs of manufacturing and transportation indicates that if cheap byproduct acid is available in the western states, the concentrated superphosphate produced at a western point will be able to reach the large midwestern markets at a slightly lower cost than Florida treble superphosphate produced as outlined above. The comparative cost of transportation and production of treble superphosphate at a midwestern point is considerably greater than production in the West or in Florida. Fertilizer produced in the West by the electric-furnace process with power at 2¼ mills (0.225 cents) per kw-hr is estimated to be slightly higher in delivered cost than wet-process production of treble superphosphate in Florida. The production of elemental phosphorus and its transportation to a midwestern point for use in the manufacture of phosphate fertilizer offers a possibility of reducing the costs of levels roughly equivalent to the costs of plant food in Florida-produced treble superphosphate. The difference in the cost of transportation of the various fertilizer products and elemental phosphorus is very great with ordinary superphosphate, the most expensive and elemental phosphorus cheaper even than triple superphosphate.

**Conclusion:** In the foregoing comments I have attempted to point out:

1. That the potentialities of the western phosphate industry are great, especially if the fertilizer can be produced at a cost that will enable western-produced phosphate to enter the midwestern area.
2. That a western concentrated fertilizer industry can provide lower cost fertilizer to the farmers in the midwestern area than can be provided by local production in that area of ordinary superphosphate from Florida rock and Gulf Coast sulphur.
3. That under average conditions which may be expected to exist in the West, western-produced, concentrated phosphate fertilizers can compete with those produced in the South. This is especially true

if additional supplies of low cost byproduct sulphuric acid becomes available in the vicinity of the phosphate deposits.

The production of phosphate by the electric-furnace method has possibilities if low-cost power becomes available in the vicinity of the phosphate deposits. It is important, however, to point out that there are possibilities, because of the wide variation in western phosphates, of setting up a plant under considerably better than average conditions. The most important opportunities for developing better than average conditions are:

1. In the discovery of rock phosphate deposits, which can be mined at low cost through open-pit mining methods.
2. Reducing the cost of rock phosphate at the plant through mining thick beds and beneficiation of the low-grade ores.
3. Reducing the net cost of rock phosphate through byproduct recovery. Vanadium values in western phosphates may run to several dollars per ton of rock.
4. In the use of low-grade phosphate rock in the electric furnace, cost may be reduced by selecting rock which does not require sintering or other pretreatment before furnacing.

Any one of these factors has sufficient influence on the final cost of fertilizer to have a very important bearing on the economics of the operation. There is sufficient variation in conditions in the western phosphate area to make it profitable to search for the most favorable deposits and to develop processes which result in the lowest net cost of production.

These are challenges to you as mining and metallurgical engineers. Upon the answers which you work out to these questions depends the future of the western phosphate industry. The industry has wide horizons and there are great opportunities for development of this western industry in the service of the entire nation through more economical food production and conservation or rebuilding of the nation's soil resources.



# Corrosion Resistant Materials and Coatings in Trail Chemical Operations

by E. A. G. Colls

Discusses corrosion in Trail chemical plants producing ammonia, sulphuric, nitric and phosphoric acids, ammonium phosphates, sulphate and nitrate, together with miscellaneous allied material problems and their solution using erosion or corrosion proof materials. Conclusion, that for work of this nature, most costly material may well be cheapest in the long run.

IN all branches of the chemical industry, corrosion plays a very costly part unless it is suitably combatted, and as a result it is probably correct that chemical and design engineers are more corrosion conscious in the field of chemical industry than in others.

---

E. A. G. COLLs is Manager of the Chemicals and Fertilizers Division, The Consolidated Mining and Smelting Company of Canada, Trail, B. C.

Third Annual Northwest Industrial Minerals Conference, sponsored by the Columbia Section in cooperation with the Oregon and the North Pacific Sections, AIME, May 1949.

TP 2836 H. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received May 26, 1949.

---

Most metals occur in nature in combination with other elements. The refined or reduced state to which man converts them for his use, by removing the combining elements, leaves them in an unstable condition. Therefore in most of the media to which the metals are exposed they are constantly combining with the constituents of their environment, or in other words they are continually being corroded. Similarly, nonmetallic materials are corroded by undergoing chemical change.

The rate at which corrosion takes place varies widely from metal to metal and medium to medium.

In practically all cases the corrosion of metals in solutions (electrolytes) is electrolytic in nature and the rate of corrosion is dependent on many factors such as temperature, pressure, internal stresses, galvanic action, metal-ion concentration, and so on.

In the chemical industry substances must be handled which are particularly corrosive to materials of construction. At the chemical plants of The Consolidated Mining and Smelting Co. of Canada, Ltd. (otherwise known as Cominco), situated at Trail, B. C., Canada, the following are some of the materials that have to be handled:

**Gases.** Sulphur dioxide, sulphur trioxide, carbon dioxide, carbon monoxide, oxygen, hydrogen fluoride, ammonia, hydrogen sulphide, and oxides of nitrogen.

**Liquids.** Solutions of ammonium sulphate, ammonium sulphite, ammonium nitrate, caustic soda and potash, sulphuric acid, nitric acid, hydrofluosilicic acid, hydrofluoric acid, phosphoric acid, aqua ammonia, and formic acid.

**Solids.** Gypsum (containing phosphoric acid), ammonium sulphate, ammonium nitrate, phosphate rock, caustic soda, potash, and phosphate fertilizers.

It should be noted that most gases are inert when dry and at normal temperatures and pressures but may become extremely corrosive at elevated temperatures. Also, most solids are inert at normal temperatures.

In the preceding section, some of the materials handled in Cominco's chemicals and fertilizers division at Trail have been listed. In order to obtain a clear picture of the corrosion problems encountered there, a brief résumé of the chemical processes involved in fertilizer production will be given.

One of the basic chemicals, sulphuric acid, is produced from  $\text{SO}_2$  (formerly waste) derived from lead-smelter and zinc-roaster operations. The acid is obtained either by an indirect process which includes absorption of the  $\text{SO}_2$  in ammonium sulphite solution followed by acidification and production of 100 pct  $\text{SO}_2$ , or by the direct contact acid process. To obtain ammonia, another basic chemical, nitrogen and hydrogen must be produced first. The nitrogen is separated by liquefaction and distillation of air or is produced from nitric-acid plant tail gas (formerly waste) by burning out the oxygen with hydrogen. The hydrogen is produced from distilled and alkalized water by electrolysis or from the oxygen-steam blast coke producer. The mixture of nitrogen and hydrogen under high pressure and high temperature forms synthetic anhydrous ammonia. From the combination of the ammonia with the sulphuric acid is produced crystalline ammonium sulphate, a finished fertilizer.

Another range of fertilizer products is manufactured by first dissolving phosphate rock with sulphuric acid to make phosphoric acid, and then, after separating out the waste gypsum, neutralizing this acid with ammonia to make a series of ammonium phosphate fertilizers.

Finally, in other plants, ammonia is burned with air and oxygen to produce nitric oxides and these by absorption in water, are converted into 56 pct nitric acid. The nitric acid so produced is neutralized under closely controlled conditions to form ammonium nitrate, and a strong solution of this nitrate is sprayed down a tower countercurrent to an up-rising current of air to form a useful prilled (pelleted) form of ammonium nitrate that makes an excellent fertilizer product. This product is named "Nitraprills."

The products mentioned above, ammonium sulphate, ammonium phosphates, and ammonium nitrate, are primary fertilizer products. Some secondary products are produced also and as these also present corrosion problems, very brief mention should be made of them. They are aqua ammonia (the solution product of ammonia in water and used at Cominco's for absorbing  $\text{SO}_2$ ), oxygen (the by-product of hydrogen production by the electrolysis of water and of the air liquefaction and distillation process),  $\text{H}_2\text{S}$  (produced in the gasification of coke), and fluorine (as  $\text{SiF}_4$  or  $\text{H}_2\text{SiF}_6$  evolved when mixing sulphuric and phosphoric acids in the phosphate plant, the fluorine being derived from the phosphate rock).

The combatting of corrosion in the production of the foregoing fertilizers or in the production of the simpler compounds entering the various processes, imposes very major considerations of economy, utility, and safety. The attack upon mild steel tanks, piping, valves, fans, and other equipment can be so rapid when handling corrosive liquids or gases

that it would be quite uneconomic to operate a process using such corrosion-susceptible materials of construction. In the same way, the usefulness of many materials is limited because of possible pollution or degradation of a product in a vessel or container in which corrosion byproducts are evolved. Furthermore, under conditions of high temperature, high pressure, or high vacuum, the materials of construction naturally play a most important part. If, for example, high-pressure piping or other high-pressure equipment is subjected to internal and unseen corrosion, disastrous results involving extreme hazards to personnel and equipment will sooner or later result. One is, therefore, always faced by such problems in the chemical industry and, while for some purposes no completely satisfactory material of construction is yet available, progress in this direction is constant and increasingly encouraging.

Cominco maintains an elaborate testing and reporting procedure by which existing materials are under constant study. New materials are tested under both laboratory and plant conditions as soon as they appear on the market, and changes to new materials with superior qualities are being made constantly. At the same time, our Research and Development Division is engaged upon the same problems from both fundamental and practical angles. The materials available and of proved utility and economic cost used in Cominco's chemical plants today, cover a wide range of products. These include special paints and cements, brick, sprayed metals, carbon compounds, Vinyon cloths, wood, glass, hard and soft rubber, neoprene, flint, ceramics, transite, quartz, calorized pipe, and the metals lead, aluminum, zinc, silver, copper, nickel, tin, admiralty metal, and the stainless steels. In addition, of course, there are the commoner materials of construction which are still useful for certain applications, mainly cast iron, mild steel, bronze, and so on. Perhaps the simplest way of indicating the practical application of corrosion-resistant materials in the chemical plants is to treat the individual plants or processes in turn.

**Sulphur Dioxide Recovery and Acid Plants:** In the  $\text{SO}_2$  recovery plant and the acid plants at Trail, the commonest form of construction for Glover towers and wash towers handling around 6 to 7 pct  $\text{SO}_2$  gas makes use of lead, lined with acid-proof brick. For absorbing  $\text{SO}_2$  gas, whether weak (1.5 pct) or strong (6.5 pct  $\text{SO}_2$ ), lead-walled, wood-packed towers are used. Ammonium sulphite solution coolers are constructed of aluminum tubes in steel tanks. More recently "Alrocked" aluminum tubes have been used or the complete cooler has been constructed from aluminum. For storing ammonium sulphite solution, wood-stave, lead-lined tanks are used. For coke-packed driers, steel construction lined with lead and acid-proof brick is employed. For storing ammonium sulphate solution (40 pct) and for pump tanks, wood-stave, lead-lined construction is used. For pumping ammonium sulphate solution under high head, two-stage 275 psi pumps are constructed from A.I.S.I. type 316 stainless steel. Coolers for  $\text{SO}_2$  gas are made from standard cast-iron pipe, and acid converters and heat exchangers are constructed with mild-steel shells and trays or tubes. In some cases, calorized pipe is used for the hotter parts of this particular equipment. The 93 pct sulphuric acid is stored in ordinary mild-steel tanks, which have a life of about 18 years. Generally speaking, lead is quite satisfactory for the



cold acid up to 80 pct and for the hot acid up to 70 pct. In the higher acid-concentration ranges, carbon steel and cast iron and high-silicon iron are useful.

A high chromium-nickel steel containing silicon, molybdenum, copper, and a small amount of manganese, looks like a promising material for acid pumps, although no plant operation has yet been experienced with this.

**Hydrogen and Ammonia Plants:** In the hydrogen and ammonia plants, corrosion problems due to caustic, oxygen,  $\text{CO}_2$ , formic acid, copper ammonium formate, hydrogen, aqua ammonia, and hydrous ammonia are encountered.

The alternating current to direct current power conversion for the purpose of hydrogen production is accomplished by various types of mercury arc rectifiers. In some of the rectifier cooling water systems not more than one half of one per cent of sodium di-chromate to inhibit corrosion of the steel bowls is used.

For the production of distilled water necessary for making up the electrolyte in the hydrogen plant, extremely pure distilled water is required, and to this end Escher-wyss water-distillation plants are used in which the heat-exchanger tubes are tinned. They have an excellent life. The remainder of the distilled water is recovered from condensed steam and the total distillate fed to the hydrogen cells is required to have a conductivity not higher than  $20 \times 10^{-6}$  mho and as low as possible a content of solids of magnesium and calcium.

The caustic soda or potash used in these cells must be pure to avoid corrosion effects which can be very marked in an electrolytic cell. It is specified that caustic soda contain not more than 0.02 pct chlorine and not more than 0.013 pct sulphate radical. The potash is specified to contain not more than 0.03 pct chlorine and not more than 0.013 pct sulphate. Our patented Trail type hydrogen cells have concrete cell tops fabricated from best grade portland cement with suitable reinforcing, the cement being mixed with crushed quartz. By using concrete collecting bells rather than metal ones, we have thus avoided one of the worst corrosion problems in a hydrogen cell. Our hydrogen-cell gas mains are now made up from transite pipe in lieu of steel pipe which was used formerly. The electrodes in the hydrogen cells consist of mild-steel plate, in the case of the cathodes, but of heavily nickeled mild-steel plate in the case of the anodes. Apart from the undesirably high over-voltage that would result from using uncoated mild steel for both cathode and anode, the corrosion encountered would be entirely uneconomic.

Ceramics also find quite a use in scrubbing towers and wash towers for packing rings and so forth.

In the ammonia plant itself, while low-carbon steel is the standard material for high-pressure piping, valves and other equipment, there are special materials for certain uses. High-pressure forgings for the actual ammonia-synthesis converters are made from nickel-chromium, medium-carbon forging steel corresponding to A.I.S.I. type 3130. In mixed-gas compressor intercoolers, either steel, galvanized iron, or admiralty metal (70 pct copper, 1 pct tin, 29 pct zinc) are used. The internal baskets, heat exchangers, and so on, in the converter employ either mild-steel baskets and tubes or 4 to 6 pct chromium-steel tubes and tube plates.

High-pressure gaskets are of considerable importance because of the need for minimizing leak-

ages. Unless the lens-type steel-ring gasket is used, soft annealed silver or copper or aluminum gaskets prove very useful. Silver in the soft annealed condition is the most useful gasket material for both the large high-pressure forging joints or for smaller joints and valves. Aluminum gaskets, while not seriously affected by anhydrous ammonia, are seriously attacked by aqua ammonia and, in this case, Monel metal finds useful application.

Cooling water may present problems; for example, a curious coincidence was once encountered in the ammonia plant where very severe corrosion conditions were found to be present, apparently due to the cooling water. Corrosion in this case was so severe as to give considerable concern, but it was finally traced down to bacteria in the water, namely *Crenothrix*. By chlorination of the water, however, the bacteria were killed and corrosion from this cause was stopped.

For building up corroded piston rods, shafts or other pieces of such equipment, the metal spray gun is now proving a useful tool, and mild steel, lead, brass, copper and stainless steel are sprayed. In fact any metal that can be drawn into wire or produced as a powder, can be used for this purpose. Plastics also are now entering the field in this connection.

**Phosphate Plant:** The plant with the worst and most diverse corrosion problems found in the chemical operations, is undoubtedly the phosphate plant, but even here very satisfactory materials are available. Considerable reliance is placed upon lead and stainless steel. Lead, both hard and soft varieties, is used for launders, agitators, pipelines, vacuum-filter lining, fluorine strippers, and many other pieces of equipment. Both the phosphoric acid reaction agitators and the large Dorrco vacuum filters are lined with 20 lb lead. The former are of wood-stave construction lined with hard sheet lead and Duro acid-proof brick, but a more recent improvement on this type of construction used hard lead walls, brick-lined, and supported by an external skeleton of steel bands. In an even later improvement, but still under construction, are similar agitator tanks of A.I.S.I. type 318 stainless steel. While the life of a hard, lead-lined, wood-stave tank runs from 15 to 16 years, and the steel skeleton lead-lined tanks will probably last longer, the stainless steel tanks, if properly protected against erosion, should last almost indefinitely. The large Dorrco vacuum filters which separate 30 to 33 pct phosphoric acid from gypsum are of steel frame construction lined with soft lead for protection of the shell. On this lead lining, wooden corrugated panels are placed as filter-cloth backing medium to provide channels for the acid to escape through, and on these wooden panels are placed panels of fairly coarse Vinyon cloth, overlaid by finer weave Vinyon as the actual filter medium. These panels are held in place by stainless steel A.I.S.I. type 318 straps.

Swenson evaporators on phosphoric acid service also have hard lead walls but the tube bundles therein are built up of Karbate tubes. Karbate is graphite bonded by plastic.

The fertilizer reaction agitators in which phosphoric acid is neutralized by ammonia to produce an ammonium phosphate slurry are built of wood staves, and lined with lead and acid-proof brick, but the shafts and impellers used to promote agitation are constructed of A.I.S.I. type 318 stainless steel and the ammonia inlet pipes are of A.I.S.I. type



316 stainless steel. Further uses for lead in this plant are found not only in the form of sheet material for wrapping conduit, pipe, valves, and so on, which may come in contact with corrosive liquids, but also for phosphoric acid pipelines and for flooring and stair treads in certain wet areas. Stainless steels of the A.I.S.I. type 316 or 318 are used extensively for building impellers, for mechanical agitation, or for the moving parts of Wilfley or La Bour sand pumps.

Another corrosion-resistant material used extensively in the phosphate plant is rubber, both soft and hard. It is used for covering or lining. In grinding the phosphate rock (in 20 pct phosphoric acid), the process is carried out in Allis-Chalmers pebble mills. These mills of steel construction are first lined with rubber. On this lining  $\frac{3}{4}$  in. maple is set in place and then 6x5x10 in. blocks of silex, which are set in acid-proof cement. These blocks are extremely abrasion-resistant and are of a flint-like texture. The actual grinding medium used consists of 4 to 5 in. diam flint pebbles.

In the reaction between sulphuric and phosphoric acids, fluorine as silicon tetrafluoride is produced and this is stripped from the mixed acids by air agitation. The stripper is a hard lead chamber. The effluent gases containing combined fluorine are scrubbed in an unlined, wood-stave tower using 20 to 22 pct  $\text{H}_2\text{SiF}_6$  scrubbing liquor. The ducts employed for carrying the silicon tetrafluoride gas and the pump tanks involved in the equipment are rubber lined, and the thickeners used consist of wood with rubber lining. Pumps, pump lines, and spray nozzles used in this part of the plant are built of Monel metal or copper. The product,  $\text{H}_2\text{SiF}_6$ , is shipped from the plant in a tank car which is rubber lined.

**Sulphate Plant:** Turning now to the sulphate plant, in which ammonia and sulphuric acid are reacted together to produce ammonium sulphate in a solution, which is then concentrated by vacuum evaporation, centrifuged, and dried, simpler but nevertheless continued corrosion problems are encountered. The ammonium sulphate solution in the first place is stored in wood-stave, lead-lined tanks and is pumped from there using Wilfley A.I.S.I. type 316 stainless steel pumps into very large Krystal type vacuum evaporators and crystallizers. The evaporator section is built of steel plate on which neoprene lining has been placed. Installation of this lining has to be a very perfectly executed job to insure against blistering under the high (24 in. mercury) vacuum. The crystallizer section of the equipment is also of steel construction but is lined with Goodrich triplex rubber. This material has, incidentally, stood up for some 13 to 15 years with only occasional patching being necessary. Neoprene itself has given excellent service in the evaporators. It is entirely likely that if and when new Krystal units were to be built, they would be constructed throughout of stainless steel. The mother liquor and crystals circulated through the foregoing circuit are pumped by Morris rubber-lined pumps equipped with A.I.S.I. type 316 stainless steel linings. The centrifuge baskets also consist of stainless steel. In this plant, considerable use is made of hard lead for floor covering to collect possible spills or overflows which can be returned to circulation. In addition acid-proof brick set in acid-proof cement has proved a satisfactory flooring. In the sulphate plant, as an

experiment, electrical conduit has been installed using aluminum tubing.

**Nitrate Plants:** In the nitric acid and ammonium nitrate plants, A.I.S.I. type 347 stainless steel is being used almost exclusively for handling 56 pct nitric acid in converters, absorbers, preheaters, boilers, condensers, and storage tanks. In the nitrate plant itself, in the evaporators, preheaters, and neutralizers in which nitric acid is made to react with ammonia to form ammonium nitrate (up to 94 pct strength) type 347 stainless steel is used. This material is also used in the strong liquor tanks and pump tanks. However, the drying is carried out in mild steel, steam heated driers. The prilling (or pelleting) of the concentrated nitrate solution is carried out in 100 ft towers. These towers present a problem in that the use of wood is of course precluded due to the hazards of nitration, and mild steel corrodes so fast that one is left with the option of using either stainless steel as a lining on other materials or of using suitably treated concrete. Aluminum sheathing is also a satisfactory lining. We have used all three methods of construction. For nitrate-plant floors extensive use has been made of acid-proof brick set in acid-proof cement which gives a very pleasing appearance in addition to being extremely durable and easy to keep clean. When making ammonium nitrate for explosive purposes, glass-lined crystallizers are used. These are horizontally-revolving, mild-steel shells lined with glass.

**Summary:** It is rather interesting to note that out of the large number of stainless steels available today, we are, in the main, dependent on only three types. These are A.I.S.I. type 316 which is an 18/8 steel containing molybdenum, A.I.S.I. type 318 which is similar to type 316 but contains columbium as a stabilizer, and A.I.S.I. type 347 which is an 18/8 steel stabilized with columbium, but containing no molybdenum.

The columbium stabilized steels are necessary where welding has to be carried out on the steel and subsequent heat treating is impracticable, which, incidentally, is generally the case.

Extra low-carbon stainless steels such as A.I.S.I. type 304 E.L.C. and type 316 E.L.C., have recently appeared on the market and may render stabilization with columbium unnecessary. While we do not as yet have any equipment in operation fabricated from these steels, we are investigating their possibilities.

One other unusual material might be mentioned and that is the special wood used for a specific purpose in our recirculated cooling water towers. For this purpose California redwood appears to be ideal. It has stood up in these towers for many years with no apparent ill effects from the continual wetting and drying.

Other materials too numerous to mention having highly specialized application must of necessity be omitted in a paper of this length. However, it is apparent from the foregoing that chemical plants are inherently dependent upon the very best materials of construction most suited for the particular purpose in hand, and that without these materials the life of equipment and the maintenance costs become altogether uneconomic. There can be no question that in general from the chemical plant operators' point of view the best material available for any particular job at once becomes the cheapest material for that job, and this is strikingly borne out by actual maintenance and replacement costs.



# Use of an Induced Nuclear Reaction for the Concentration of Beryl

by

A. M. Gaudin,

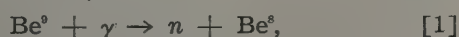
John Dasher,

James H. Pannell,

and Wilfred L. Freyberger

A new sorting process for beryllium minerals is described. This depends upon emission of neutrons upon irradiation by gamma rays, a nuclear reaction which is specific for beryllium at the appropriate energy level. Signals given by neutrons when ore passes on a belt are changed to mechanical commands through amplifiers and other electrical equipment.

WHEN beryllium is bombarded by gamma rays under suitable conditions it evolves neutrons. This nuclear reaction,



is utilized in the process that is described in this paper. The equation, which physicists often abbreviate to  $\text{Be}^9(\gamma, n)\text{Be}^8$ , shows that a photon,  $\gamma$ , is absorbed by the  $\text{Be}^9$  isotope of beryllium which then emits a neutron,  $n$ , leaving  $\text{Be}^8$  as a residue.\* It has

\* This new beryllium isotope  $\text{Be}^8$  is unstable and decays spontaneously and rapidly to two helium atoms, but as this reaction is of no importance in the process described in this paper, the matter is mentioned merely for the sake of completeness.

been determined that the threshold for this reaction is 1.63 mev; that is, gamma rays having an energy less than 1.63 million electron volts do not have enough energy to remove a neutron from the beryllium nucleus. The threshold value for the beryllium ( $\gamma, n$ ) reaction is lower than that for any other element. The next most sensitive element is heavy hydrogen (deuterium) whose gamma neutron reaction,  $\text{D}^2(\gamma, n)\text{H}^1$ , has a threshold of 2.2 mev. Thus, gamma rays having an intensity between 1.63 and 2.2 mev will evolve neutrons from beryllium and from beryllium only.

Nuclear reactions can be considered as occurring between a projectile and a target. The probability of the occurrence of any such encounter can be expressed in terms of the "cross section" of the target nucleus for the projectile and projectile energy. This cross section, then, is the area normal to the path of the projectile controlled by the target for the purpose of the reaction under consideration. Cross

sections are usually expressed in barns, in which a barn is  $10^{-24}$  sq cm. The cross section for Eq 1 is very small, even for nuclear reactions, and of the order of  $10^{-3}$  barns. It is therefore necessary that a high flux of gamma radiation at the appropriate energy be used to obtain an appreciable neutron yield.

In view of the nuclear properties of beryllium it seemed that if pieces of beryllium ore on a moving belt were bombarded with an intense beam of gamma rays, those pieces which are rich in beryllium minerals would emit neutrons at a sufficiently high rate to make it possible to detect that emission, and to use it to actuate a mechanical device for removal of the active from the barren particles.

The principal economic source of beryllium is the mineral, beryl. Beryl occurs mainly in pegmatites which are characteristically coarse grained. The suitability of mechanical concentration depends upon the size at which the desired mineral can be liber-

A. M. GAUDIN, Member AIME, is Director, JOHN DASHER, Member AIME, is Executive Officer, JAMES H. PANNELL is Chief of the Physics Section and WILFRED L. FREYBERGER is Physicist, Massachusetts Institute of Technology Mineral Engineering Laboratory, Watertown, Mass.

The work upon which this paper is based was sponsored by the Raw Materials Operations Office of the U. S. Atomic Energy Commission under contract AT-30-1-Gen-211.

AIME New York Meeting, Feb. 1950.

TP 2812 B. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Nov. 1, 1949.

ated. Fortunately this desired mineral with its specific nuclear reaction often occurs in coarse crystals which would tend to make utilization of the reaction practical.

A series of experiments was conducted to determine if this idea could be reduced to practice.

**Detecting Neutrons:** Even if the photon used in Eq 1 has no excess energy above the threshold, the neutron emitted leaves the residual nucleus at high speed. Methods for detecting *fast* neutrons are quite inefficient, and it was therefore considered that the neutrons evolved from beryllium should be slowed down, or *moderated*, before detection was attempted.

In order to moderate the neutrons evolved, a hydrogenous material was desired. Plastics and rubber might have been used, but for our experiments a paraffin wax block with a minimum wax thickness of 2 cm between beryl and the detector was shown to work efficiently.

One method of detecting slow neutrons is absorption by some element such as manganese which yields a radioactive product. This method, however, requires integration over an appreciable time and would not be useful for picking.

Another method of detecting slow neutrons involves a fission counter which consists of a foil plated with a thin coating of U-235. Slow neutrons are detected by causing fission which gives a detectable pulse from the foil counter. Because of the other uses for fissionable material it was doubtful if this type of counter could be justified for the purpose of concentrating beryl.

The method adopted for detecting slow neutrons involved the use of counters filled with boron trifluoride gas. The  $B^{10}$  isotope, which normally comprises 18 pct of natural boron, undergoes the following reaction:



An energy of 2.3 mev is released in the reaction and the corresponding pulse is easily detected. Moreover, the cross section is so large (3525 barns for  $B^{10}$ ) that several per cent of the neutrons entering a counter are absorbed by the boron atoms. Various types of  $BF_3$  counters were obtained and tried. Best results were obtained with counters of 1 in. diam and 10 in. long filled to a pressure of 28 to 50 cm of mercury with  $BF_3$  gas enriched in  $B^{10}$ .

Since some weak pulses may arise from action of the high gamma flux on the counters, a pulse-discriminating step must be included in the amplifying circuit. The instruments used in transforming the neutron signal to an electrical pulse of standard size and shape included, therefore, a preamplifier, a fast linear amplifier, and a pulse-discrimination-level circuit.\* Similar techniques have already been

\* The first experiments utilized a model 501 amplifier, model 500 preamplifier, model 1090 5000-volt negative power supply (Instrument Development Labs., Chicago) and model 101A scaling circuit (Atomic Instrument Co., Boston). For convenience, and in anticipation of better results, the polarity of the H-V supply was changed at a later date. This necessitated changes in the preamplifier and it was decided to make one similar to the Bell-Jordan type A1B (W. H. Jordan and P. R. Bell, *Review of Scientific Instruments* [1947], 18, 703). A Glassmike condenser was used in the H-V filtering network and a ceramic condenser for coupling the input. A number of preamplifiers of this type have been made and all were entirely satisfactory. When a model 204A (Atomic Instrument Co.) amplifier became available, it was used in place of the 501 as it contained a rise-time selector. Connections from the 204A were as follows: discriminator output to scaler input, high-level output to oscillograph input. Acquisition of a Dumont 248-A oscillograph, containing a triggered sweep of adjustable time, was of considerable aid in all of the neutron-counting work.

used in radioactivity measurements where alphas

are counted in the presence of a high beta activity.

**Sources of Gamma Radiation:** To obtain the intense gamma radiation desired, the simplest possibility was to use radium. Although radium does not emit gamma rays, its daughter-element products do. About 7 pct of the gamma rays from radium and its descendants have an energy of 2.19 mev which is just under the deuterium threshold, and a total of 27 pct of the radium gammas have energies in excess of 1.63 mev, the beryllium threshold.

Another possible source of gamma rays is radioactive isotopes produced in a pile. Of these,  $Sb^{124}$ , with a 60-day half life and 1.72 mev gamma, appeared to be the most promising. Radium was the more economic source when this investigation started. However, multi-curie  $Sb^{124}$  sources are now available from pile irradiation of natural antimony at less than \$100. A sample of  $Sb^{124}$  is now in use.

A third possibility involves the use of Van de Graaff X ray generators. Small Van de Graaff generators which give a two-million-volt X ray (2 mev  $\gamma$ ) are in use at a number of hospitals. A larger instrument, adjustable to give up to 4 mev gammas was available for intermittent use at MIT. Initially radium was used for convenience and subsequently the equipment was tested under the Van de Graaff generator.

**Results of Experiments with the Mineral Stationary:** The gamma rays came from a 1-g radium source (1 curie). The radium source was surrounded by an annular container filled with beryl and quartz, and the radium and beryl were placed in a hole in a wax block which contained the counter tubes. To

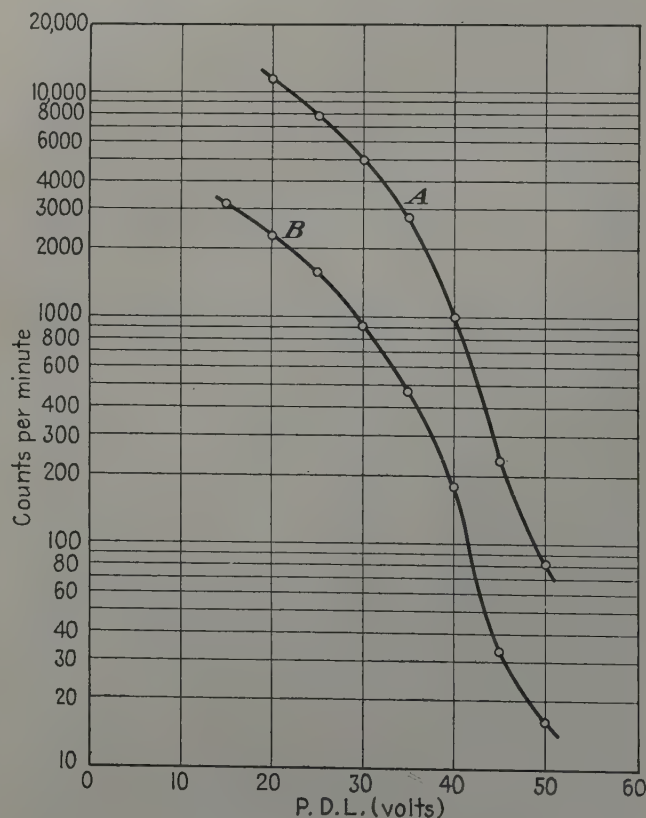


Fig. 1—Bias curves with first 1-g radium source.

A. Neutrons from 500 g pure beryl and background.  
B. Background due to radium alone.



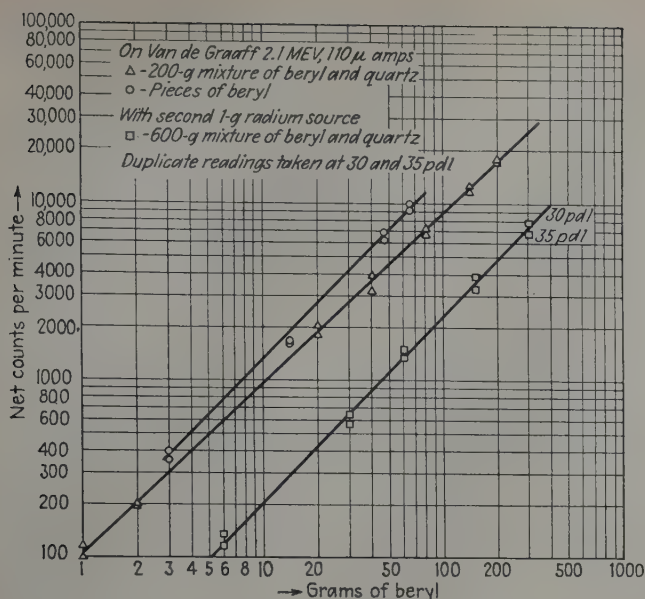


Fig. 2—Relation between neutron counts and quantity of beryl.

avoid interference from gammas, various settings of the pulse discrimination level were used in an attempt to obtain the maximum ratio of neutrons from beryllium to background. The result of one such experiment is shown in fig. 1. It is seen that as the pulse discrimination level (pdl) was increased, the ratio of beryl counts to background remained constant; that is, the background was not due to the small pulses caused by the gamma radiation. Upon further investigation this background was found to be due to the fact that the 1-g radium source emitted neutrons, and the high background experienced was due to true neutron counts. Experiments were then made with this apparatus with the MIT Van de Graaff generator. With this source of gammas, a negligible background was obtained.

After the discovery that neutrons were emitted by the first radium source, this source was exchanged for an especially purified source. The second 1-g radium source was found to emit only 10 pct as many neutrons as the first. With the second source, counting rates for pure beryl 50 times as high as the background could be obtained, while with the first radium source, counting rates were only 5 times the background.

Experiments were made with various pieces of beryl and mixtures of beryl and quartz. Results are shown in fig. 2 which shows that the counting rate is directly proportional to the weight of beryl (slope of one on the logarithmic plots). For the Van de Graaff, counting rates of about 100 counts per minute per gram of beryl (approximately 10 pct BeO and 4 pct Be) were obtained. With the second radium source, counting rates were 20 per min per g. In these experiments only one counter was used. Although picking might have been accomplished with the second radium source, it was decided to conduct picking experiments under the Van de Graaff generator.

**Picking Experiments:** A picking machine was constructed including a wax block, four counters, and preamplifiers connected by coaxial tees to the

electronic circuits. The block was mounted on a frame and a semicircular tunnel was provided through it for a 3-in. wide rubber belt. In the middle of the wax block there was a hole for the X ray beam to irradiate the pieces of beryl. On the belt at the point of irradiation was a kicker operated by a solenoid which was actuated by the electronic circuit. The kicker had a maximum rate of 5 cycles per second. The electronic circuit was set in the most sensitive position so that only 1 or 2 neutron counts were needed to actuate the kicker. In this position there were also 3 random kicks per minute caused by spurious counts. The belt was operated at speeds of 6 and 10½ in. per sec which gave irradiation times in front of the 2-in. long kicker of 1/3 to 1/5 sec. Pulses from the counters and preamplifiers were fed to the linear amplifier and the pulse-discrimination-level circuit of the scaler. The scaler input was also sent to an RCA rate meter and relay circuit which actuated the solenoid. This rate meter was far more intricate than necessary for this job as it was set to operate the kicker on one pulse or on two pulses in 1/5 sec. Fig. 3 shows a diagram of the circuit.

Various sized pieces of beryl ranging from 1 to 10 g each were placed on the belt 2 in. apart, and, after the generator was started and adjusted, the belt was started and the pieces passed in front of the kicker. The results obtained are given in table I. Note that by increasing the current and decreasing the distance from the target, 1-g pieces were picked with certainty at the higher belt speed with irradiation times of 1/5 sec per piece.

Next, mixtures of beryl and quartz were prepared. The sizes tested were minus ¾-plus ½-in. and minus ½-plus ¼-in. Various methods of feed-

Table I. Results of Picking Sized Beryl

Run	Belt Speed, In. per Sec	Target-Beryl Distance, Cm	Current μ amps	Weight of Pieces, G, Each	Number of Pieces		
					Feed	Conc.	Tailing
1	6	90	100	15	3	3	0
2				10	6	6	0
3-6				5	24	23	1
7-8				3	13	10	3
Rerun tailings				3	3	2	1
9-11	10.5	90	100	1	21	9	12
Rerun tailings (runs 9 and 10)				1	8	6	2
Rerun tailing (run 10 rerun)				1	1	0	1 <sup>a</sup>
12-14				5	24	22	2
15-16				3	17	11	6
Rerun tailing (run 15)				3	3	2	1 <sup>a</sup>
17	10.5	58	100	3	8	8	0
18				1	10	3	7
Rerun tailing				1	7	3	4
Rerun tailing (run 18 rerun)				1	4	3	1 <sup>a</sup>
19-20				1	19	19	0

<sup>a</sup> Each of these three pieces was spectroscopically examined and found to contain ½ normal BeO content.

ing were tried but none was developed which would give a steady feed to the belt. The experiments were conducted with a Jeffrey vibrating feeder fitted with a wooden V-shaped trough in the feeder pan and a stirring device incorporated in the hopper. The feeding rate was adjusted by a variac, but the feeder frequently jammed. Speeding up the vibrator dislocated the jams, but then caused excessively heavy feeding momentarily. When these piles passed in front of the guides used to align the pieces before they reached the kicker, about 10 pct of the material fell off into the discharge chute. This difficulty could probably have been remedied by mechanical im-

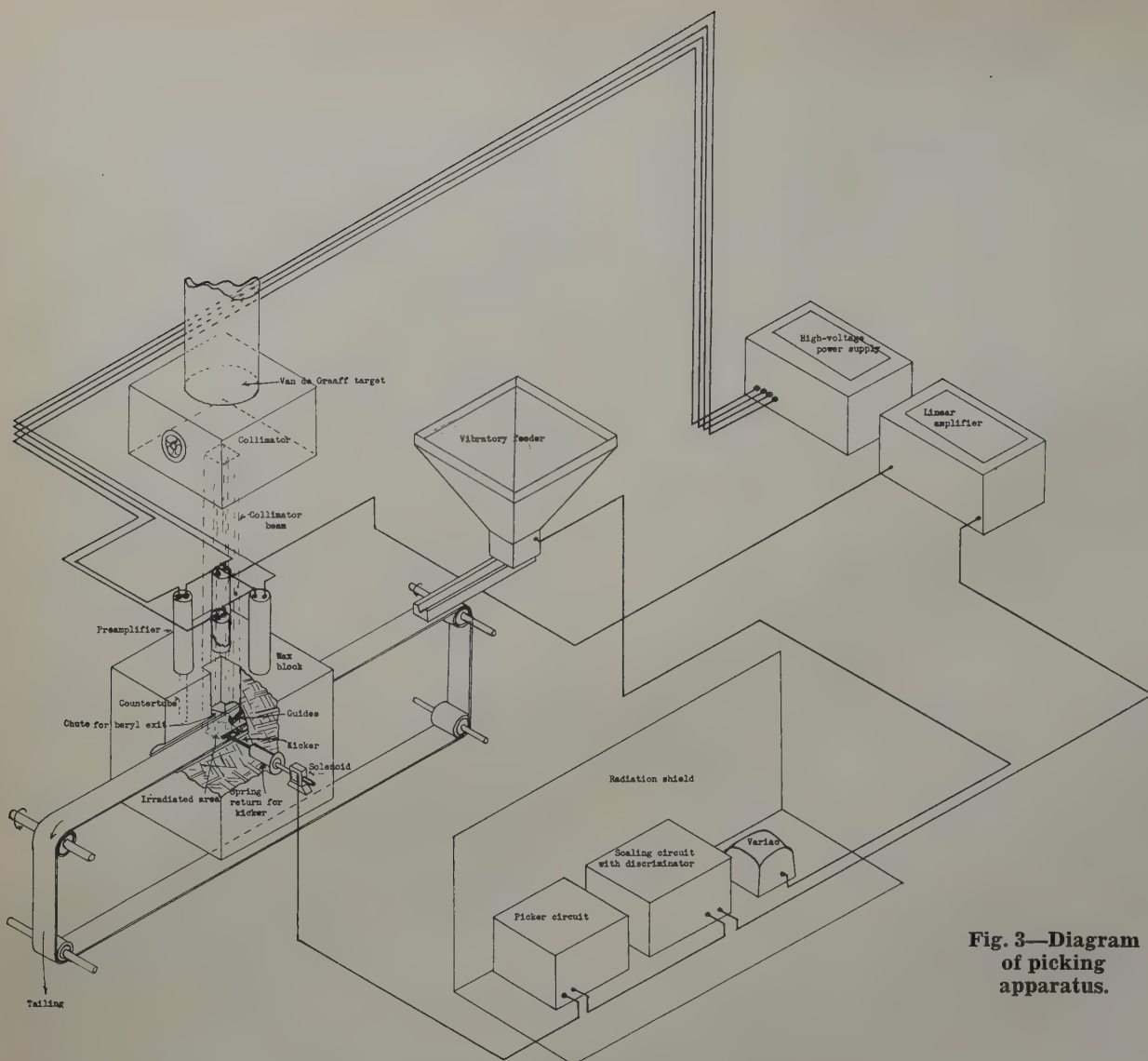


Fig. 3—Diagram of picking apparatus.

Table II. Picking Tests on Mixtures of Minus 3/4-Inch-Plus 1/2-Inch Pieces

Run	Feed, Pieces		Concentrate, Pieces		Tailing, Pieces		Per Cent Picked	
	Beryl	Quartz	Beryl	Quartz	Beryl	Quartz	Beryl	Total
3	71	300	49	50	23	250	68	27
4	45	450	21	77	24	373	47	20
5	50	359	38	75	11	784	78	12
6	Rerun #5 tailing		9	39	2 <sup>a</sup>	745	82	6
7	Rerun part #5 & #6 conc.		30	26	2	64	94	46
8	10	796	9	92	1	704	96	12
9 <sup>b</sup>	10	796	9	91	1	705	90	12

<sup>a</sup> Petrographer says these two pieces were feldspar.

<sup>b</sup> Products from 8 recombined and rerun.

Note: Runs 3 and 4: X ray at 2.0 mev and 100  $\mu$  amps, counter voltage 1,950, amplifier setting 5,000 gain and 15 pdl, belt speed 6 in. per sec, target to belt 90 cm. Runs 5-9: amplifier at 10,000, generator at 2.1 mev.

provements. The results obtained by picking the various mixtures are shown in table II. Apart from mechanical difficulties of feeding and guiding coarse pieces, the results are promising.

**Conclusions:** Preliminary experiments have shown that the specific ( $\gamma, n$ ) reaction of beryllium can be used for its concentration. Radium, purified from elements such as beryllium so that it emits few neutrons, or a Van de Graaff generator, can be used as a source of gamma rays. Probably a pile-irradiated piece of antimony containing Sb<sup>124</sup> could also be used. Boron trifluoride counters, preferably filled to a

pressure of 28 to 50 cm of mercury with the gas enriched with B<sup>10</sup>, can detect efficiently the neutrons evolved provided paraffin or other hydrocarbons are used as moderators. Pulses from the counters can be used to operate a mechanical picker.

With the apparatus employed, pieces of beryl as small as 1 g can be picked with certainty at rates approaching 5 per sec. Before the method can be applied practically, a suitable economic source of intense gamma radiation must be developed as well as a mechanical feeding system which will feed coarse pieces of rock at an even rate.

**Acknowledgments:** The authors are most appreciative of the efforts and interest of Christian La-Pointe, of the Canadian Bureau of Mines, Clark Goodman, Robley D. Evans, Matthew Sands and John Trump, of Massachusetts Institute of Technology, R. W. Dodson, of Brookhaven National Laboratory, R. F. Errington, of Eldorado Mining and Refining Co., James Beckerley and R. F. Van Wye, of the New York Operations Office of the U. S. Atomic Energy Commission, and W. W. Havens, Jr., of Columbia University.



# Concerning the Adsorption of Dodecylamine on Quartz

Using an adsorption-column technique the partition of dodecylamine between quartz and water has been determined at concentrations ranging from 0.5 to 4000 mg per liter. The adsorption varies as the square root of the concentration. Attachment of dodecylamine to quartz is reversible. The amount adsorbed in a flotation operation giving almost complete recovery is under 5 pct of the amount required for a monolayer.

by  
A. M. Gaudin  
and  
F. W. Bloecher, Jr.

THIS paper describes the results of a series of tests carried out to determine the partition of dodecylamine between distilled water and the surface of quartz at various concentrations of dodecylamine acetate in solution. An adsorption-column technique was utilized for this purpose. Ralston and Harwood, of Armour and Co., had synthesized marked dodecylamine acetate for us.<sup>1</sup> In their preparation which we employed, 0.0007 pct of the total carbon was the radioactive isotope, carbon fourteen. Unmarked dodecylamine acetate of high purity was also prepared by them for our use. The presence of carbon fourteen in the dodecylamine enabled us to analyze quantitatively for dodecylamine on the surface of quartz, using a radioactive-tracer technique. The method used was that known as internal gas counting of carbon dioxide, formed by oxidation of the amine. This method has already been described,<sup>2, 3, 4</sup> and the reader is referred to these papers for details of the analytical method and procedures.

A. M. GAUDIN, Member AIME, is Professor of Mineral Engineering, and F. W. BLOECHER, JR., Junior Member AIME, is Research Engineer, Massachusetts Institute of Technology, Cambridge, Mass.

Abstract from Master's Thesis, Richards Mineral Engineering Laboratory, MIT, by F. W. Bloecher; May 20, 1949.

AIME New York Meeting, Feb. 1950.

TP 2813 B. Discussion (2 copies) may be sent to Transactions AIME before May 30, 1950. Manuscript received Nov. 7, 1949.

**Adsorption Apparatus:** To determine the partition of dodecylamine\* between the surface of quartz

\* In this paper the term "dodecylamine" is used to refer to the amine-chain part of the salt and would include the undissociated salt, dodecylamine, dodecylamine hydroxide, or dodecylammonium ion. We are inclined to favor the view that the dodecylammonium ion is the active ingredient, but we do not feel that our data warrant ruling out all other forms of "dodecylamine".

and surrounding liquor, a technique analogous to chromatographic columns was utilized. The experimental setup is shown in fig. 1.

The dodecylamine-acetate solution was made up in a volumetric flask and poured into the glass funnel shown. It was allowed to drip slowly onto the quartz bed, the rate of flow from the funnel reser-

voir being controlled by a hose clamp. The quartz column was contained in a small, fritted-glass filter (size 20 mm, C). The beaded top edge of the filter was cut off to enable the holder and contents to fit into the open end of the 30/45 joint on the combustion tube. This combustion tube is used in the analytical train described in a previous paper.<sup>3</sup> A piece of gum-rubber tubing joins the bottom of the fritted-glass filter to a filter flask, which in turn is connected to a vacuum line, or water aspirator. The rate of flow of amine solution through the bed was

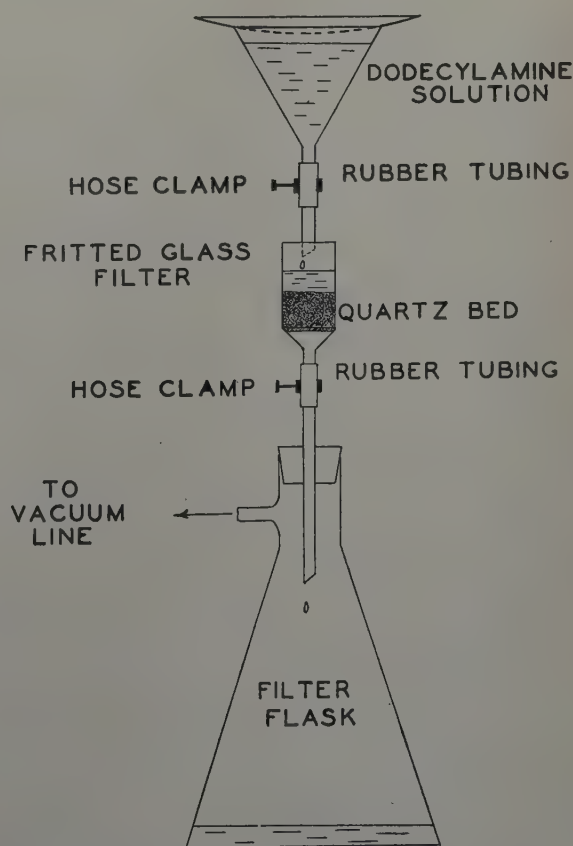


Fig. 1—Schematic sketch of adsorption apparatus.

Table I. Test Description

Test No.	Conc. Amine Acetate, Mg per Liter	Wt. SiO <sub>2</sub> , Grams	pH Solution	pH Effluent	Wt Soln. Retained, G	Total Amount Soln., Cc	Exposure Time	
							Hr	Min
32a	0.53	5.3920	6.10	6.14	0.9670	500	4	25
32b	0.53	5.8811	6.10	6.52	0.9757	500	4	40
34a	1.54	6.3340	6.27	Not recorded	1.2240	500	4	20
34b	1.54	6.8312	6.27		1.1111	500	3	50
35	3.05	6.4685	6.24	Not recorded	1.2457	500	4	30
30a	5.47	7.3651	5.90	6.31	0.9636	500	2	57
30b	5.47	5.6787	5.90	6.27	1.1153	500	3	05
5	11.66	6.9602	5.92	6.17	1.0476	500	2	10
14	32.5	4.3857	6.08	5.92	0.7685	500	2	10
4	62.4	7.9380	5.83	6.22	1.0637	1,000	3	10
2	63.86	7.2017	Not recorded		1.6535	380	1	40
1	64.56	8.1644	Not recorded		0.8536	150	Not recorded	
6	86.0	8.9168	6.28	6.34	1.1917	500	1	30
7	87.0	6.4116	5.93	6.19	0.8747	500	1	40
3a	120.2	5.5492	Not recorded		0.7061	280	2	45
3b	120.2	4.9140	6.2	6.3	0.4922	200	3	00
10a	202.2	5.3339	6.33	6.27	0.6591	490	2	35
10b	202.2	5.1513	6.33	6.38	0.6710	490	3	05
12	443.5	6.0045	6.53	6.45	0.8040	500	3	05
15	632.3	5.4669	6.50	6.48	0.8906	500	2	35
13	662.3	6.4898	6.58	6.44	0.7595	500	2	45
16	993.4	4.2028	6.70	6.70	0.9662	500	4	30
19a	1,394.8	4.2906	6.99	6.88	0.6942	500	3	50
19b	1,394.8	4.3605	6.99	6.68	0.6845	500	4	10
17	1,856.9	4.4849	6.99	6.40	0.5538	500	4	20
18a	3,018.6	4.2837	6.87	6.60	0.5056	500	4	00
18b	3,018.6	4.8977	6.87	6.62	0.6382	500	4	25
22a	3,998.8	0.9988	7.04	6.94	0.2127	250	4	25
22b	3,998.8	1.0245	7.04	7.10	0.3007	250	4	35

controlled by a second hose clamp. The flow was adjusted so that the rate of feed to the bed was equal to the rate of efflux from the bottom, a small pool of liquor covering the quartz bed at all times.

**Preparation of Quartz:** In order to prevent channeling through the quartz bed, it was felt that quartz of fairly uniform size should be used. Therefore, cleaned quartz was fractionated in an infrasizer, and the cone 5 fraction set aside for use.

Massive quartz was first crushed through a jaw and roll crusher, and a 20/65 mesh fraction then separated by screening. Leaching with mixed hydrochloric and nitric acids cleaned the quartz of abraded iron from the crushing operation. Rinsing with an excess of distilled water freed the quartz of dissolved iron and excess acidity. The acid-washed 20/65-mesh fraction was then ground in an Abbe mill with porcelain pebbles and distilled water. The milled product was dried and then fractionated in an infrasizer.

The cone 5 fraction had an average particle size of 21.5 microns (measured as the perpendicular to the long dimension on a microscope stage). The surface area of the quartz was measured by the B.E.T. method of nitrogen adsorption,<sup>6</sup> and by the newer method in which krypton gas is used.<sup>7</sup> Since the quartz was relatively coarse, reproducibility of the determinations using the nitrogen adsorption method was poor, two separate runs giving values of 2430 and 2250 cm<sup>2</sup> per g. However, the krypton method is much better suited for particles in this size range, and in three measurements resulted in an average value of 2320 cm<sup>2</sup> per g. This figure has

been assumed as the correct specific surface of the quartz.\*

\* The measurements made with krypton were experimentally reproducible to within about  $\pm 2$  pct. However, the surface area of the quartz, calculated from the experimental data, depends on the area occupied by each krypton molecule. Assuming a close-packed liquid layer, this area is 14.0 sq Angstroms per molecule. Beebe<sup>8</sup> found, however, that on a "calibrated" sample of anatase, the area was 19.5 sq Angstroms per molecule, indicating that the arrangement of the molecules in the adsorbed monolayer was not independent of the underlying crystal structure. Beebe's value depended upon the prior determination of the anatase surface both by the gas adsorption method of Brunauer and Emmett<sup>9</sup>, and the absolute method of Harkins and Jura (*Jnl. Amer. Chem. Soc.* (1944), 66, 1366.) In the absence of any experimental data giving the surface area occupied by krypton on quartz, we have arbitrarily used an area of 16.8 sq Angstroms per molecule, halfway between the liquid value, 14.0, and Beebe's figure of 19.5.

**Test Procedure:** In carrying out an adsorption test, an excess of dodecylamine-acetate solution was passed through the miniature column. The quartz surface became covered with adsorbed dodecylamine to an extent determined by the experimental conditions. Among the important variables affecting this partition are temperature, concentration of dodecylamine acetate in the solution, the pH, and the presence of foreign ions in the solution. An attempt was made to keep all these factors constant, except that the concentration of the dodecylamine acetate in solution was varied from test to test. The temperature ranged from 20° to 25°C. No attempt was made to adjust the pH, and for almost a ten thousand-fold change in amine concentration, the pH changed only about one unit, climbing from the distilled water value of about 5.9 up to about 7.0 for very high concentrations of dodecylamine acetate (around 4 g per liter). The quartz was stored in a covered glass jar, and all glassware kept scrupulously clean by scouring with hot, chromic-acid



Table II. Radioassay Results of Tests Outlined in Table I\*

Test No.	Conc. Amine Acetate, Mg per Liter	Percentage Strength Amine Acetate <sup>a</sup>	Activity Amine Acetate, Cpm per Mg <sup>b</sup>	Activity of Solution, Cpm per Cc	Actual Activity, Cpm (total) <sup>c</sup>	Activity Less B and Soln., Cpm <sup>d</sup>	Total Wt Amine Acetate Adsorbed, Mg	Mg Amine Acetate per Gram SiO <sub>2</sub>
32a	0.53	100.0	29,300	15.5	826	767	0.02618	0.00486
32b	0.53	100.0	29,300	15.5	1,115	1,056	0.03604	0.00613
34a	1.54	100.0	29,300	45.1	1,785	1,686	0.05754	0.00908
34b	1.54	100.0	29,300	45.1	1,855	1,760	0.06007	0.00879
35	3.05	100.0	29,300	89.4	2,297	2,142	0.07311	0.0113
30a	5.47	100.0	29,300	160.3	3,905	3,708	0.1266	0.0172
30b	5.47	100.0	29,300	160.3	2,973	2,751	0.09389	0.0165
5	11.66	100.0	31,500	367.3	5,758	5,327	0.1691	0.0243
14	32.5	47.50	14,960	486.2	2,905	2,482	0.1659	0.0378
4	62.4	10.07	3,172	197.9	1,667	1,410	0.4445	0.0558
2	63.86	10.07	3,172	202.6	1,596	1,215	0.3830	0.0532
1	64.56	10.07	3,172	204.8	1,608	1,400	0.4373	0.0536
6	86.0	10.07	3,172	272.8	2,162	1,791	0.5646	0.0633
7	87.0	10.07	3,172	276.0	2,007	1,716	0.5410	0.0644
3a	120.2	10.07	3,172	381.3	1,839	1,524	0.4805	0.0866
3b	120.2	10.07	3,172	381.3	1,429	1,195	0.3767	0.0767
10a	202.2	10.07	3,172	641.4	2,439	1,971	0.6214	0.1165
10b	202.2	10.07	3,172	641.4	2,412	1,937	0.6107	0.1185
12	443.5	3.83	1,206	534.9	1,930	1,455	1.206	0.201
15	632.3	3.708	1,135	717.7	2,196	1,467	1.293	0.250
13	662.3	2.655	959.3	635.3	2,185	1,649	1.719	0.265
16	993.4	2.655	777.9	772.8	1,945	1,152	1.481	0.352
19a	1,394.8	2.023	592.7	826.7	2,295	1,685	2.843	0.663
19b	1,394.8	2.023	592.7	826.7	2,293	1,691	2.853	0.654
17	1,856.9	2.023	592.7	1,100.6	2,293	2,267	3.825	0.853
18a	3,018.6	2.023	592.7	1,789.0	6,825	5,885	9.929	2.32
18b	3,018.6	2.023	592.7	1,789.0	7,665	6,487	19.94	2.23
22a	3,998.8	0.9546	279.7	1,118.0	1,318	1,037	3.708	3.71
22b	3,998.8	0.9546	279.7	1,118.0	1,190	811	2.900	2.83

\* Assay for test 15 made in counter 7. For tests 16 through 32 counter 4 was used. For tests 13 and 14 counter 10 was used. All other assays were made in counter 8.

<sup>a</sup> Weight per cent active salt in diluted salt. <sup>b</sup> Cpm = counts per minute.

<sup>c</sup> Corrected for resolving time. <sup>d</sup> B = background activity; Soln. = activity from solution retained in voids.

cleaning fluid and rinsing thoroughly with distilled water.

In detail, the test procedure was as follows:

1. An accurately weighed amount of dodecylamine acetate was dissolved in distilled water in a volumetric flask. The tests were usually run in duplicate, one liter of solution being made up. 500 cc of solution was then passed through each column.
2. Four to ten grams of quartz were weighed out in a tarred beaker, dispersed in distilled water, and transferred to a tarred, fritted-glass filter, using as much distilled water as was required to complete the transfer. Suction was applied to the fritted-glass filter during the transfer.
3. The amine solution was passed through the quartz column, using the setup shown in fig. 1. The level of the liquor in the filter was always kept above the top of the quartz bed. A record was kept of the exposure time of quartz to amine solution.
4. About 3 cc of the stock amine solution was taken out for a pH measurement in a Beckman, Model G, pH meter. The meter was standardized with buffer solution before each determination.
5. Near the end of the run, a few cubic centimeters of effluent were collected for another pH measurement, the meter being standardized again before this determination.
6. At the end of the test, the quartz bed was not sucked dry by the vacuum. Instead, as soon as air started to pass through the bed, the filter and

contents were removed and weighed. The difference between the final weight and the total dry weight of quartz and filter, was equivalent to the volume of solution (in cubic centimeters) retained in the voids of the column. The radioassay results were corrected for the activity due to this solution retained in the bed.

7. The filter and contents were placed in a combustion tube for oxidation.<sup>2</sup> Before combustion, however, the filter and contents were allowed to stand for a day or so at room temperature in order to dry. Attaching the bottom of the fritted-glass filter to a hose from an air line, and inserting the filter, open end down, into a combustion tube, made it possible for one to gently blow the quartz out of the filter into the bulb. The filter was then pulled up out of the tube, and reinserted with the bottom end down. With the bottom end of the filter down, very little gas collected underneath it during combustion of the sample.

**Test Results:** Several dozen experiments were made, and of these 29 tests have been retained as being significant. The test data are too bulky to be presented in a single tabulation, and are, therefore, presented in two tables. In table I are pertinent test data exclusive of radioactivity measurements, arranged not in chronological order, but in order of increasing concentration of dodecylamine salt in the feed liquor. In table II are the radioassay results of the experiments, arranged in the same order as in table I.

The headings of table I are self-explanatory. It may, however, be mentioned that col. 7, headed, "Total Amount Solution, Cc," refers to the total volume of solution passed through the bed in the exposure time listed in col. 8.

In table II, the third column headed, "Percentage Strength," refers to the weight per cent of radioactive dodecylamine acetate present in the diluted salt. The activity of the dodecylamine acetate was cut down to a suitable level for each test by proper dilution with the unmarked salt. By dissolving an accurately weighed amount of unmarked dodecylamine acetate in benzene, together with an accurately weighed amount of radioactive salt, complete mixing of the unmarked and radioactive salts was obtained. By cooling the solution to about 10°C the dodecylamine acetate recrystallized and was sep-

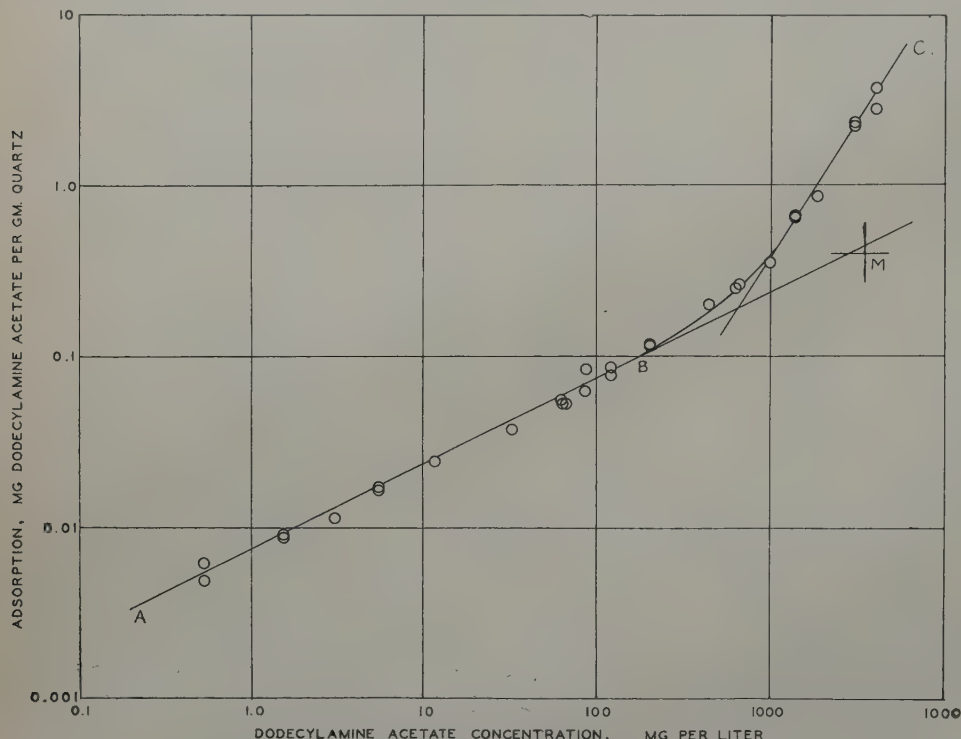
was  $5.47 \times 29,300 \times 0.001 = 160.3$  cpm per cc

Multiplication of the activity per cubic centimeter of solution by the volume of solution retained in the voids of the column gave the activity attributable to the liquid retained in the damp quartz at the end of the test.

The figures reported in col. 6, headed "Actual Activity, Cpm (Total)," are the activities observed by radioassay of the carbon dioxide produced by oxidation of the filter and its mineral contents. These actual activities, in counts per minute, are corrected for resolving time of the circuit.<sup>5</sup> Col. 7 records the net, actual activity; that is, the total actual activity less background and less the activity due to the solution retained in the bed. The total weight of amine adsorbed (reported as milligrams of dodecylamine acetate) was determined by dividing the net actual

activity by the activity per milligram of salt. Dividing the total weight of amine salt adsorbed by the weight of quartz, gave the adsorption per gram of quartz.

**Graphical Presentation of Data:** Fig. 2 is a plot on logarithmic scale of the dodecylamine ace-



**Fig. 2—Adsorption isotherm of dodecylamine on quartz; pH 6 to 7, temperature 20° to 25°C.**

arated from the benzene by filtration. The percentage strength was calculated by dividing the weight of active amine acetate by the total weight of marked and unmarked salts.

The activity of the dodecylamine acetate used in each test was determined by multiplying the actual specific activity\* of the undiluted salt (counts per

\* By actual activity is meant the actual number of decays of C-14 atoms occurring during a given time in the sensitive volume of the counter tube. This actual activity is not observed directly by radioassay. It is determined by taking the observed counting rate, correcting that for resolving time of the instruments (period when the scaling circuit is dead and will not record a pulse from an ionizing event in the counter tube), and subtracting from that corrected rate the background activity. See ref. 2 and 5 for complete discussion.

minute per milligram in a given counter) by the percentage strength divided by 100. For example, the 10.07 pct salt used in test 1 had an actual activity of

$10.07 \times 31,500 \times 0.01 = 3,172$  counts per min per mg, where 31,500 is the actual specific activity (cpm per mg) of the undiluted radioactive salt in counter No. 8.

The activity of the solution was calculated by multiplying the specific activity of the salt by the concentration (milligrams per liter), and dividing by 1000. For example, in test 30, the solution activity

was 5.47 x 29,300 x 0.001 = 160.3 cpm per cc

gram of quartz. The curves of fig. 2 and 3 may be termed adsorption isotherms of dodecylamine on quartz, since the temperature at which the adsorption occurred was held practically constant.

These figures show that the adsorption of dodecylamine increases with the solution concentration, but that the adsorption is not directly proportional to the concentration. Instead, the adsorption increases as the square root of the concentration (slope of line AB equals  $\frac{1}{2}$ ), up to a solution concentration of about 200 mg per liter. Adsorption of dodecylamine then increases more rapidly, eventually more rapidly than the solution concentration. Such a change in behavior would seem to indicate that at least two mechanisms are involved in the adsorption process we have investigated.

To interpret these unexpected results, we have found it useful to consider, on the one hand, the adsorption that would correspond to the formation of a monomolecular layer of dodecylamine at the quartz surface, and, on the other hand, the solution concentration that corresponds to the formation of micelles.

According to the crystallographic orientation of an element of surface, the shape of the area that can be allotted to each cation-adsorption site is a



lozenge or rectangle, of such proportions that it does not differ very radically from being a square and has an average area of 23.4 sq Angstroms.<sup>8</sup> It would seem, therefore, as a first approximation, that a monolayer of adsorbed dodecylamine chains would be oriented on the quartz surface in a ratio of one chain to each cation-adsorption site, providing only that the space required for each amine chain were less than 23.4 sq Angstroms. It happens that the cross-sectional area per chain, for close packing of long-chained hydrocarbons on the surface of water, is equal to 20.5 sq Angstroms,<sup>9</sup> or about 15 pct less than the available surface per adsorption site on the quartz surface. We shall therefore assume that a complete monomolecular coating of adsorbed dodecylamine on quartz requires 23.4 sq Angstroms of surface per adsorbed chain.

Since the surface area of the quartz employed in the experiments was known to be 2320 cm<sup>2</sup> per g, the amount of reagent required to form a monolayer can be calculated. Dividing the total area per gram of quartz, in square Angstroms, by the product of the area required per site, 23.4 sq Angstroms, times the number of molecules per gram mol (Avogadro's number), gives the

**Fig. 3—Adsorption isotherm of dodecylamine on quartz, molar concentrations.**

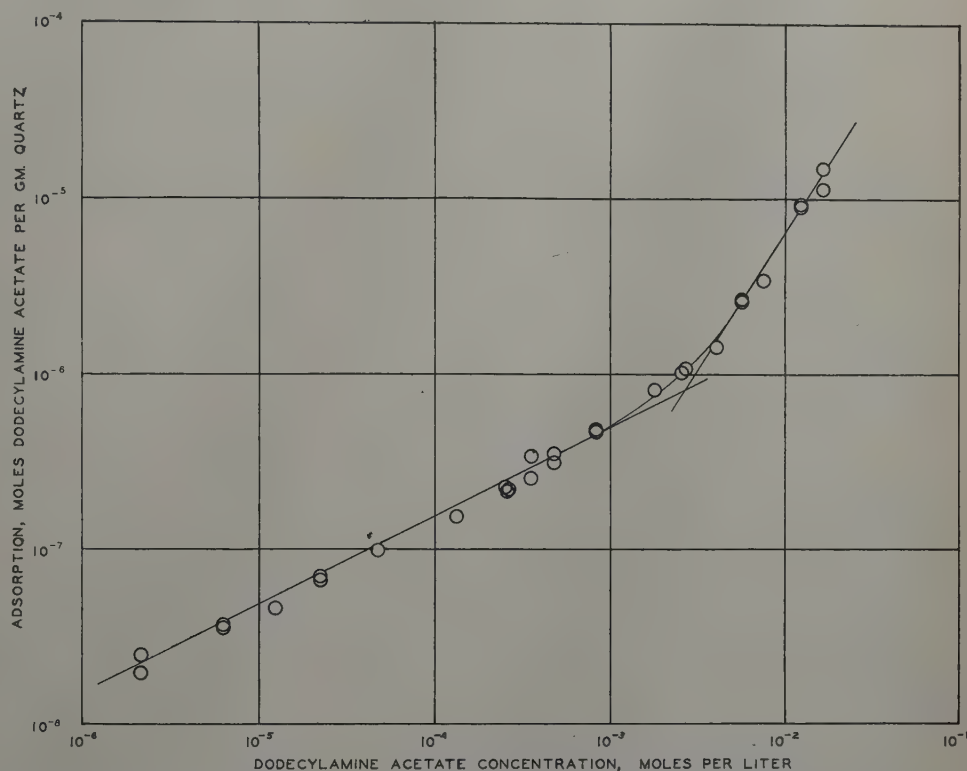
amount of reagent required to form a monolayer, expressed in gram mols per gram of quartz.

$$2320 \times 10^{18} / 23.4 \times 6.02 \times 10^{23} = 1.65 \times 10^{-6} \text{ mols per g quartz}$$

This is equivalent to 0.40 mg dodecylamine acetate per gram of quartz. In fig. 2 this quantity of dodecylamine acetate is represented by the horizontal line passing through point M.

The studies of the late A. W. Ralston and his associates have established that in water solutions of normal, long-chained amines, the dissolved substances exist at low concentrations in a dispersed state, but that an aggregated or micellar condition appears after a certain critical concentration is reached. In the case of n-dodecylamine acetate in water at 30°C, this critical concentration is about 3.5 g per liter.<sup>10</sup> We will accept that as the critical point for micelle formation in the solutions we used. In fig. 2 the vertical line through point M is at this concentration. Point M, therefore, would represent the coexistence of a critical micellar concentration of dodecylamine salt, and of a completed monolayer of adsorbed dodecylamine. It is reasonable to assume that these two events should happen simultaneously, since both indicate inability of the single phase, represented by a homogeneous solution, to contain more than a certain maximum amount of solute.

It is interesting to observe that point M falls close to an extension of the straight-line portion of the adsorption isotherm. This leads to the speculation that the spread between the lines ABM, and ABC, represents the occurrence of polymolecular adsorption before completion of the initial or monomolecular coating. It is certain that with concentrations of dodecylamine acetate in excess of 1 g per liter, the coating is several chains in thickness. It is postulated here, as a result of the consideration of point M, that the coating is not uniformly thick, and that at solution concentrations below that represented by point B, is in places entirely lacking. It may furthermore be indicated that at concentrations of dodecylamine acetate below 100 mg per liter, the coating is entirely monomolecular, and is less than one-fifth complete. For example, at a solution concentration of 1 mg per liter, the coating is about 1.8 pct complete.



**Attainment of Adsorption Equilibrium:** As a rough check to determine how rapidly equilibrium conditions were attained in the adsorption column, tests 1, 2a, 2b, and 4 were run. The solution concentration in each was about the same, around 63 mg per liter, as was the weight of quartz. The data obtained are summarized in table III.

**Table III. Equilibrium Adjustment Data**

Test No.	Conc. Amine Acetate, Mg per Liter	Total Amount Soln., Cc	Exposure Time,		Mg Amine Acetate per G SiO <sub>2</sub>
			Hr	Min	
1	64.56	150	Not recorded		0.0536
2a	63.86	380	1	40	0.0532
2b	63.86	100	0	30	0.0325
4	62.4	1,000	3	10	0.0558

Running only 100 cc of solution through the column in 30 min resulted in adsorption of only 0.0325 mg dodecylamine acetate per gram of quartz. Passing 150 cc through the column (time unrecorded) resulted in almost the same adsorption figure as when one liter of solution was passed through the column in 3 hr, 10 min. It was therefore felt that by

using 500 cc of solution for each test, and an exposure time of at least 2 hr, equilibrium conditions in the miniature adsorption column would be attained.

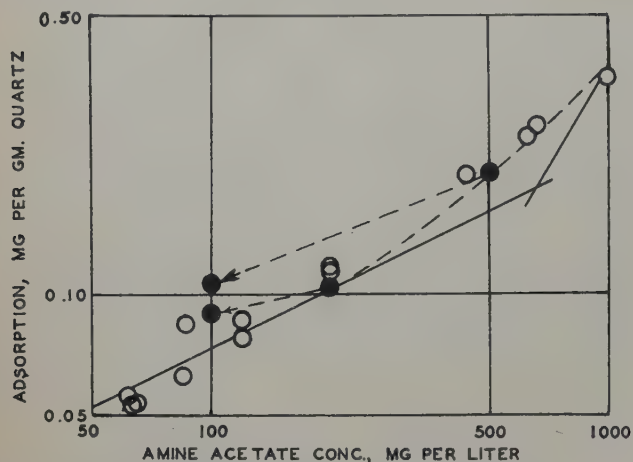


Fig. 4—Graphical illustration of attempt to approach equilibrium from above.

As an additional check, however, an attempt was made to approach equilibrium from above. In one test, the quartz was first coated with dodecylamine at a solution concentration of 506 mg per liter. One liter of 100 mg per liter solution was then passed through the column in 3 hr, 20 min. The adsorption expected from the curve of fig. 2 was 0.074 mg per g of quartz. The actual adsorption was 0.106 mg per g. Evidently the volume of the second solution, and the exposure time of the quartz to solution, were not sufficient to cause complete readjustment of equilibrium.

A second test was run, in which the quartz was first coated with dodecylamine from a solution of 200 mg per liter strength. Then one liter of 100 mg per liter solution was passed through the column in 4 hr, 10 min. Adsorption expected was again 0.074 mg per g, and the actual observed figure was 0.090 mg per g.

A graphical representation of these tests is shown in fig. 4, where a portion of the curve of fig. 2 is reproduced together with the points obtained by approaching equilibrium from above (black circles). It is obvious that dodecylamine goes onto the surface of quartz when equilibrium is approached from below, and comes off when equilibrium is approached from above, but that in neither instance has equilibrium actually been attained. It had been hoped that the exposure time and amount of solution allotted for each adsorption test would be sufficient to insure attainment of equilibrium. The above tests, however, indicate that adsorption equilibrium was approached, but was not quite attained.

**Reversibility of Adsorption:** If the attachment of dodecylamine to the surface of quartz is an adsorption phenomenon, it should be possible to replace adsorbed radioactive dodecylamine with unmarked dodecylamine, or vice versa, at any given solution concentration. Three tests were run to see if this were possible.

In one test the quartz was first coated with unmarked dodecylamine at a solution concentration of 198.8 mg per liter. Then, 500 cc of radioactive dodecylamine-acetate solution (198.6 mg per liter strength) was run through the column in 2 hr, 35 min. The adsorption expected from fig. 2, on the basis of equilibrated exchange, was about 0.116 mg

per g of quartz, and the actual observed adsorption, determined by radioassay, was 0.110 mg per g.

In another test, an attempt was made to replace adsorbed radioactive dodecylamine by unmarked dodecylamine. The surface was first loaded up with radioactive dodecylamine at a solution concentration of 200.2 mg per liter. Then 500 cc of unmarked dodecylamine-acetate solution (200.4 mg per liter strength) was run through the column in 3 hr, 15 min. If complete replacement of the radioactive amine had occurred, there should have been no observed activity (aside from the background count) upon radioassay of the sample. Of a total of 0.118 mg of dodecylamine expected to be adsorbed on all of the quartz, only 0.0042 mg, or 3.6 pct consisted of the original radioactive coating.

A third test was run at a dodecylamine-acetate concentration that is in the range of concentrations encountered in actual flotation practice. The sample was first coated with unmarked amine at a concentration of 4.50 mg per liter, and then one liter of radioactive dodecylamine-acetate solution of the same strength was passed through the column in 3 hr, 25 min. The adsorption expected was 0.0160 mg per g, and the observed adsorption was 0.0152 mg per g of quartz. Assuming the difference to be

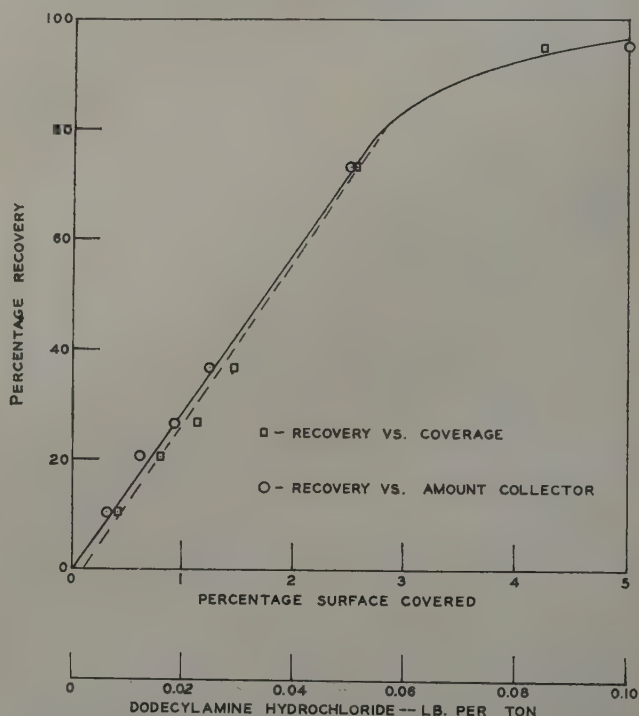


Fig. 5—Flotation recovery of quartz vs. amount of collector added, and vs. percentage available surface occupied by dodecylamine.

Amount of frother constant at 0.08 lb per ton.

made up of unreplaced, unmarked dodecylamine, 5 pct of the original, unmarked dodecylamine coating remained on the surface.

Thus, it seems that the attachment of dodecylamine chains to the surface of quartz is a reversible phenomenon similar to other reversible adsorption phenomena. At any one instant, for a given concentration of dodecylamine acetate in solution, there is a certain total amount of dodecylamine on the quartz surface. The individual dodecylamine chains, however, are not firmly anchored there. They are continually arriving at and departing from the surface. At equilibrium, the rate of adsorption from solution is equal to the rate of desorption, or departure of dodecylamine chains from the surface.



**Correlation of Adsorption and Flotation Data:** In connection with another study, a number of batch flotation tests had been run on the same type of quartz as was used in the adsorption tests. Because of the availability of data on the partition of dodecylamine between quartz surface and liquor, and because of the ability to measure the amount of surface on the quartz by the gas-adsorption method, it now becomes possible to correlate flotation recovery with the density of the adsorbed collector coating. The astonishing fact emerges that to effect even nearly complete recovery the coating is very far from being a complete monolayer.

Of course knowledge of the reagent addition in the flotation tests is expressed directly in terms of pounds of reagent per ton of mineral or in terms of concentration of added reagent per unit volume of liquor. The quantity involved in the partition data is not the quantity of reagent added but the quantity of reagent remaining in solution at the end of the equilibrating process. It is possible to go from data expressed in one way to data expressed in the other form by a mathematical process of successive approximations which are detailed in the thesis by one of the authors.<sup>11</sup>

The results obtained in correlating density of adsorption with flotation recovery are summarized in fig. 5. For example, a recovery of 95 pct is obtained with the surface covered to the extent only of 4 to 5 pct, and a recovery of about 40 pct is obtained with the surface coverage of only about 1.5 pct.

In making the calculations that led to these results, allowance was, of course, made for the fact that the specific surface of the quartz used in the flotation tests was not the same as that of the quartz used in the partition data presented earlier in this paper. In fact it was nearly twice as large, being 4430 instead of 2320 cm<sup>2</sup> per g.

We have assumed that the equilibrium conditions as represented by our adsorption isotherm were also attained in the flotation cell, and that the surface concentration in terms of amount of reagent adsorbed per unit area of quartz at complete surface coverage was proportional to the concentration depicted by the horizontal line through point M in fig. 2.

It is, of course, also implicitly assumed that the reagent ions are standing upright on the surface of the mineral, much as trees in a forest. Clearly, if the reagent ions are assumed to lie flat or at a considerable angle to the vertical, their surface coverage is much greater. Also, if the reagent ions are assumed to be in constant motion, the amount of surface over which they sweep during their vibrations and gyrations about the point of attachment is larger still.

**Summary of Results:** Using radioassay technique and radioactive dodecylamine acetate, an isotherm for the adsorption of dodecylamine from solution on to the surface of quartz has been established for a pH range of 6 to 7, and a temperature of 20° to 25°C. At low concentrations the amount adsorbed per gram of quartz varies directly as the square root of the solution concentration. The solution concentration that would be required to form a monolayer of adsorbed dodecylamine appears to be equal to the concentration at which micelles appear. At submicellar concentrations, however, a polymolecular adsorption coating begins to form before the monolayer has been completed.

The reversibility of the dodecylamine adsorption process was demonstrated by showing that equi-

librium conditions on the surface could be approached from above as well as from below. This reversibility was also demonstrated by replacing adsorbed, unmarked dodecylamine with radioactive dodecylamine, and vice versa.

It was shown that pure quartz, with a specific surface of 4430 cm<sup>2</sup> per g, could be floated almost completely (95 pct recovered), at a pH near 7, with about 5 pct of the total available adsorption sites occupied by dodecylamine chains. This was achieved with 0.1 lb dodecylamine hydrochloride, and 0.08 lb terpeneol per ton of quartz. At that reagent figure, about 60 pct of the collector was on the mineral surface, and 40 pct in the liquor. A curve was presented showing the relationship between flotation recovery and percentage surface covered by collector, together with the relationship of recovery to amount of collector added.

### Acknowledgments

Armour and Company, through their generosity in setting up the Armour Flotation Research Project, have made this research work possible. H. M. Corley, Manager of the Chemical Division, and Dr. Victor Conquest, Director of Research, of Armour and Company, have been most generous with their help and encouragement. Acknowledgment is due the late A. W. Ralston, and his successor, H. J. Harwood, of the Armour research staff, for their work in preparing the reagents used.

W. W. Miller, of Brookhaven National Laboratories, contributed valuable suggestions concerning the use of the carbon-dioxide-filled counters. The authors would also like to express their appreciation of the suggestions and discussions proffered by Professor P. L. de Bruyn of the Department of Metallurgy at M. I. T.

### References

- <sup>1</sup> H. J. Harwood and A. W. Ralston: The Synthesis of Lauric Acid and Dodecylamine Containing Carbon Fourteen. *Jnl. Org. Chem.* (1947) **12**, 740.
- <sup>2</sup> A. M. Gaudin, F. W. Bloecher, C. S. Chang, and P. L. de Bruyn: Radioactive Tracers in Flotation. *Min. and Met.* (1948) **29**, 432.
- <sup>3</sup> A. M. Gaudin and P. L. de Bruyn: Radioactive Tracers in Mineral Engineering Problems and Particularly in Flotation. *Can. Min. and Met. Bull.* (1949) **42**, 331.
- <sup>4</sup> W. W. Miller: High Efficiency Counting of Long-Lived Radioactive Carbon as CO<sub>2</sub>. *Science* (1947) **105**, 123.
- <sup>5</sup> R. D. Evans: "Fundamentals of Radioactivity and Its Instrumentation." Section in *Advances in Biological and Medical Physics*. Vol. I, p. 213-214. (1948). New York. Academic Press.
- <sup>6</sup> A. M. Gaudin and F. W. Bowdish: Surface Measurement by van der Waals Adsorption. *Trans. AIME* (1946) **169**, 95; *Min. Tech.* (May 1944) TP 1666.
- <sup>7</sup> R. A. Beebe, J. B. Beckwith, and J. M. Honig: The Determination of Small Surface Areas by Krypton Adsorption at Low Temperatures. *Jnl. Amer. Chem. Soc.* (1945) **67**, 1554.
- <sup>8</sup> A. M. Gaudin and A. Rizo-Patron: The Mechanism of Activation in Flotation. *Trans. AIME* (1943) **153**, 462.
- <sup>9</sup> N. K. Adam: *The Physics and Chemistry of Surfaces*. p. 50. (1941) London. Oxford University Press.
- <sup>10</sup> A. W. Ralston, C. W. Hoerr, and E. J. Hoffman: Studies on High Molecular Weight Aliphatic Amines and Their Salts. IV. Electrical Conductivities of Aqueous Solutions of the Hydrochlorides and Acetates of Dodecyl- and Octadecylamines. *Jnl. Amer. Chem. Soc.* (1942) **64**, 97.
- <sup>11</sup> F. W. Bloecher, Jr.: Concerning the Adsorption of Dodecylamine on Quartz. Dept. of Met., M. I. T. May 20, 1949.



## Separation of Molybdenite at the Miami Copper Company

by C. H. Curtis

THE valuable mineral content of the current feed to the Miami concentrator is as follows: copper, 0.7 pct total; molybdenum, 0.01. Flotation of this ore yields a sulphide concentrate containing: chalcocite, 44 pct; molybdenite, 0.5; pyrite, 50.0; insol, 5.5.

A combination of potassium ethyl xanthate and pentasol amyl xanthate as collectors, and pine oil as frother, are used in this flotation. Rejection of

C. H. CURTIS, Member AIME, is Chief Metallurgist, Miami Copper Co. & Castle Dome Copper Co., Miami, Ariz.

AIME San Francisco Meeting, February 1949.

TP 2842 B. Discussion (2 copies) may be sent to Transactions AIME before May 31, 1950. Manuscript received April 27, 1949.

pyrite is encouraged by holding the amount of collectors used to the minimum consistent with copper recovery and by operating at high alkalinity (equivalent to 0.35-0.40 lb CaO per ton solution of pH 11.0).

The molybdenum recovery in the sulphide concentrates under the above flotation conditions is approximately 50 pct of that originally present in the ore. Taking into account the acid soluble molybdenum, indicated molybdenite recovery is 75 to 80 pct.

The attempt to separate the molybdenite into an acceptable molybdenum product begins with the bulk sulphide flotation concentrate just described. This concentrate is composed of chalcocite, whose floatability has been promoted to the fullest extent possible for the sake of its recovery from the ore, together with the pyrite which has been activated along with the copper mineral. The problem is to deaden the copper and iron minerals, and to float the molybdenite. Obviously in the accomplishment of this end, conditioning and preparation of the pulp, prior to flotation, plays an all important role.

The first step is thickening to 50 to 60 pct solids, with milk of lime added to the thickener feed to maintain an alkalinity of the pulp equivalent to a pH of 8.5 to 8.8 during its residence in the thickener. The purpose of the thickening is primarily to reduce the volume of pulp for subsequent treatment. However, the relatively prolonged retention of the pulp in the thickener at the desired alkalinity is known to have a favorable depressing effect upon pyrite. There is a limit for this alkalinity above which a depressing effect upon molybdenite occurs.

The thickened pulp (alkalinity: 0.015 lb CaO per ton, pH 8.8), discharges into an agitator, retention time approximately 2 hr, to which additional lime is added to raise the alkalinity to 0.35 to 0.40 lb CaO per ton solution, pH 11.6. This additional lime is required for pyrite depression and can be tolerated without loss of molybdenite because of the limited time of contact in the conditioner tank.

The pulp leaving the lime conditioner passes through two successive steaming tanks, which are mechanically agitated, and into which live steam is admitted directly into the pulp near the bottom of the tanks. The temperature of the pulp is maintained as near boiling as possible. The steaming time is approximately 4 hr. The pulp leaving the last

steamer has an alkalinity of about 0.04 lb CaO per ton solution, pH 8.7. It is believed that oxidation of the copper and iron sulphides occurs during steaming, the resulting sulphates reacting the calcium hydroxide to calcium sulphate and thus reducing the alkalinity. Since the steamer-feed solution is already saturated with calcium sulphate, the calcium sulphate produced during steaming is precipitated. It is believed that this calcium sulphate is precipitated preferentially on copper and iron mineral surfaces thus decreasing their floatability.

Aside from the "lime chemistry" during steaming, pine oil is displaced from the pulp and xanthate decomposed, which has a major effect upon the deadening of the copper and iron sulphides.

Following steaming, the hot pulp is admitted to another conditioning tank wherein it is aerated, primarily for cooling, but incidentally for additional oxidation of the copper and iron sulphides.

The resulting "deadened" pulp is then diluted to 20 pct solids, a specific collector for molybdenite, ordinary stove oil, is added and the separation of the molybdenite by flotation is undertaken at a pH of 8.5 to 8.8 in standard Miami air-flotation machines. B-22 frother is used when necessary. A re-grind of the thickened rougher concentrates is made prior to the first cleaning operation chiefly for rejection of insoluble in subsequent flotation. The cleaner concentrate is then stepped up to 90 pct MoS<sub>2</sub> in an 8-cell Denver flotation machine No. 18. Sodium silicate is added to the cleaner circuit. Its effect is to flocculate molybdenite and stabilize the froth.

In summary, it may be stated:

1. Separation of molybdenite into an acceptable product from sulphide copper concentrates by flotation involves preliminary preparation and conditioning of the pulp, which is of major importance.

2. This preparation and conditioning consists of several successive steps: (A) Thickening to 50 to 60 pct solids at controlled alkalinity to reduce volume of pulp and to contribute to depression of pyrite. (B) Agitation at high-pulp density for limited time with additional lime to provide for depression of pyrite. (C) Steaming at high-pulp density for decomposition of xanthate and xanthate surface films, evolution of pine oil, and oxidation of sulphide minerals other than molybdenite. The latter involves sulphating of lime with probable precipitation of calcium sulphate preferentially on copper and iron minerals. (D) Aeration at high-pulp density for cooling, and for further oxidation of copper and iron sulphide minerals. (E) Dilution of pulp to 20 pct solids and addition of specific collector for molybdenite, common stove oil.

It is hardly necessary to point out that this rather drastic procedure for depression of previously activated copper and iron sulphide minerals, without at the same time depressing molybdenite, is possible due to the inherently high floatability and refractory nature of molybdenite. However, molybdenite is susceptible to depression by excessive lime which must therefore be limited to the amount consistent with satisfactory molybdenite recovery.

The steaming procedure is being carried on at Miami Copper Co. under license agreement with Janney, Nokes, and Johnson, holders of letters patent on the process.



# Coal Preparation for Synthetic Liquid Fuels

by W. L. Crentz,

J. D. Doherty,

and E. E. Donath

The principal washing problems in preparing coal for the synthetic fuel industry may be stated as: (1) ash removal for the Bergius process; and (2) sulphur removal for the Fischer-Tropsch process. A study is made of the effect of ash on oil losses during the elimination of the nonliquefiable portions of the coal in the hydrogenation process. The oil loss increases with ash content and results in increased consumption of coal, hydrogen, and power.

**I**N 1948, the United States used nearly six million barrels of petroleum products every day. Although substitution of synthetic fuels for the natural petroleum product is not here yet, large quantities of special-purpose coal will be required in ever-increasing volume as the domestic production of oil from other sources fails to meet demand and the production of synthetic fuels begins.

W. L. CRENTZ and J. D. DOHERTY, Members AIME, are Coal Technologist and Assistant Chief, Office of Synthetic Liquid Fuels, respectively, U. S. Bureau of Mines, Washington, D. C., and E. E. DONATH is Scientific Consultant, War Department, Quartermaster Corps; formerly Chief Chemist for Research, High Pressure Dept., I. G. Farbenindustrie, Ludwigshafen, Germany.

AIME Columbus Meeting, September 1949.

TP 2832 F. Discussion (2 copies) may be sent to Transactions AIME before April 30, 1950. Manuscript received Sept. 21, 1949.

Papers by authors on the staff of the U. S. Bureau of Mines are not subject to copyright.

Petroleum economists do not agree as to when the shortages of domestic oil will appear; but the increasing per capita consumption of oil, coupled with a constantly growing population drawing on a potential domestic oil supply estimated to be 85 billion barrels, sooner or later will result in demand exceeding domestic supply. When this situation comes about, the excess demand will be either eliminated by curtailment of consumption or the demand met by increasing importation of foreign oil or by the production of synthetic fuels. The first recourse would definitely retard the heretofore unbroken rise in American living standards, while dependence upon a foreign oil supply during a war or other emergency would be dangerous. On the other hand, the enormous coal reserves in the United States, plus

sizable deposits of oil shale, indicate adequate means of augmenting a diminishing domestic oil supply.

Some idea of the impact of an expanding synthetic liquid fuels program on the coal-mining industry may be gathered when it is considered that to supply just half of our present-day petroleum needs synthetically from coal would increase the coal output by about 500 million tons annually, using an 11,900 Btu bituminous coal as the plant feed. If either a subbituminous coal or lignite were used, the quantity required would be materially increased.

**Synthetic Fuel Processes:** The principal synthetic fuel processes using coal as a raw material are the Bergius, or coal-hydrogenation process, and the Fischer-Tropsch, or gas-synthesis process. These two processes not only produce different end products, but the preparation of the coal used in each presents its own individual problem to the coal-preparation engineer. The hydrogenation process brings together a mixture of dry, finely powdered coal, catalyst, and heavy oil to form a paste. Addition of hydrogen under rather high pressure and temperature conditions brings about transformation of the coal to a liquid product, from which a high-grade gasoline and other motor fuels may be obtained. The Fischer-Tropsch process requires the production of synthesis gas, a mixture of carbon monoxide and hydrogen. At the Bureau of Mines Louisiana, Mo., demonstration plant, this gas is being made by reacting dried pulverized coal with carefully controlled volumes of oxygen and superheated steam. After purification and in the presence of the proper catalyst and suitable conditions of pressure and temperature, the hydrogen and carbon monoxide components of the synthesis gas react to form motor fuel. The two processes are not competitive but rather com-

plement each other to form a well-rounded economy of oil self-sufficiency.

**Coal-preparation Problems:** The question then arises: What coal-preparation problems will become paramount in order to discharge a satisfactory product from the washing plant to the synthetic fuel-manufacturing plant? While problems of drying and pulverizing the coal fall within the coal-preparation field, this paper will include only the coal-cleaning phase.

At the risk of over-simplification, the washing problems in preparing coal for the synthetic fuel industry may be stated as: (1) ash removal for the Bergius process; and (2) sulphur removal for the Fischer-Tropsch process. The problem of both ash and sulphur reduction is not only technologic but economic.

It will be the task of the coal-preparation engineer to devise the type of coal-cleaning equipment that will enable him to exercise the control necessary to transform float-and-sink data into operating results. A start has been made in this direction by the construction of washing plants designed to prepare metallurgical coal by low-gravity separation. Unlike the preparation of coal for general industrial use where the degree of preparation is determined mainly by competitive conditions, the preparation of synthetic-fuel coal will have a definite monetary value for each per cent of ash reduction in the coal for hydrogenation and each per cent of sulphur reduction for gas-synthesis coal.

In meeting these demands, the coal industry will be faced with the problem of satisfying a new type of customer. It is basic that the qualifying standards as to what constitutes a satisfactory product are set by the consumer and met by the producer. Only during times of a so-called "sellers' market" will the consumer relax his demands for the type of coal best suited to his needs. It is characteristic of the coal industry, with its large productive capacity and numerous operators each struggling for a share of the market, that the demands of the consumer are paramount. Heretofore the yardstick of industrial consumer acceptability has been based chiefly on calorific value. With the possible exception of buyers for metallurgical use, the large customers of the coal producers have been interested in the product primarily from a steam-raising standpoint. Purchasers of coal for conversion to synthetics will be interested not only in the calorific value but to an even greater degree in the adaptability of the coal to processing. Their questions will involve rate of conversion, and ash and sulphur content (not as a heat-value factor but as an item of expense in chemical processing of the coal). The coal industry must recognize the importance of obtaining this new information about the coal resources, so that, when the quality specifications are stated, the preparation-plant operator will be in a position to meet the requirements.

**Preparation Practice at the Hydrogenation Plant, Billingham, England:** The United States Government in 1943 dispatched a scientific mission to Great Britain to examine the hydrogenation plant of the Imperial Chemical Industries, of Billingham, England. Reports given the American investigators indicated no great difficulty in the preparation of a suitable clean coal, using a sand-flotation process. Raw coal containing 8 to 9 pct ash was crushed to  $\frac{3}{8}$  in. top size, dedusted at about  $\frac{1}{16}$  in., and the  $\frac{3}{8} \times \frac{1}{16}$  in. product fed to a 10 ft diam sand cone.

The float product from this cone is dewatered and desanded over a shaker screen and then fed to a second cone, which makes the final separation at low gravity. Reports vary as to the separation gravity, but float-and-sink data of the raw coal indicate an 82 pct recovery of float coal containing 2.5 pct ash on a specific gravity of 1.35. Using American washing practice as a criterion, the Billingham separation would appear to be formidable; and so it would if the washing characteristics of the British coal were similar to our middle western coals containing the same raw coal ash content. Fortunately for the British, this particular raw coal is adaptable to low-gravity separation much more readily than the medium ash content American coal, the term "low-gravity" being used to indicate separation at a specific gravity substantially lower than is employed in common commercial washing operations. More than 63 pct of their raw coal floated on a 1.30 sp gr which moved the large coal mass at least 0.05 sp gr unit away from the separation gravity, materially cutting down the amount of coal found within  $\pm 0.05$  unit of the 1.35 washing gravity. Many of the American coal beds will produce large amounts of float coal on 1.35 sp gr, but raw coals having an 8 to 9 pct ash content usually produce a float product containing 4 to 5 pct ash in the 1.35 sp gr range, which frequently defies further desanding.

**Effect of Ash in Coal Hydrogenation:** In the Bergius process, the effects of ash in the coal are felt in the following manner:

1. Effect upon the catalyst.
2. Some ash constituents tend to settle in the converters, adversely affecting the space time yield and service factor. Periodic converter shut-down to remove ash accumulation might be necessary in some cases.
3. Decrease in oil yield.

Problem 1, the effect of ash on the catalyst, usually can be corrected by added catalysts or chemical treatment. Problem 2 may have great importance but would vary, depending on the properties of the individual coal. Problem 3, decrease in oil yield because of high ash coal, will be examined and discussed more fully later in the paper. Quantitatively, it has been estimated that the oil loss due to ash amounts to about 9 lb of oil for each per cent of ash in 1 ton of the hydrogenated coal. The residue obtained after recovery of oil from the ash may have some value as a fuel, but it is not expected that the fuel value would materially affect the economic balance of loss of oil against the cost of cleaning the coal. An additional benefit derived from reduction of ash by mechanical cleaning is that washing usually decreases the fusain content of the coal, thereby improving the coal conversion.

Fortunately for the preparation engineer, the coal requirements of the hydrogenation process are divided about equally between coal actually hydrogenated in the process and coal used for secondary purposes like steam raising, production of producer gas, and gasification coal for hydrogen. This affords an outlet for a middling product, rich in bony coal, resulting from preparation of a low-ash coal by low specific-gravity separation. The ash specifications of the coal to be used in the Bergius process can best be set at the lowest ash percentage obtainable with a coal recovery of at least 50 pct of the raw-coal feed.



Washability studies of many of the eastern and midwestern coal beds indicate that they contain varying amounts of coal low enough in ash content to meet the ash requirements of the synthetic fuel industry. This low-ash coal has not been separated out in the past, as the small recoverable yield made the process economically unfeasible. Furthermore, separation of the low-ash coal particles from coal particles of slightly higher ash content required plant control outside the range of conventional equipment.

**Effect of Ash and Sulphur in Gas Synthesis:** In the Fischer-Tropsch (gas-synthesis) process, the quantity of ash in the raw-coal feed is not as critical as in the Bergius process. This can readily be understood because the Fischer-Tropsch process calls for preliminary gasification of the coal leaving the inert high-ash material behind as residue. Nevertheless, the ash matter in the raw coal is not gasified, so that it may be economically advantageous to wash the coal before gasification to increase the volume of synthesis gas per ton of coal charge. Here again, the problem is more economic than technologic, for even though the raw feed is first subjected to mechanical cleaning, either for ash removal or to assure a uniform plant feed, the operation probably would involve well-known washing methods now being used in commercial preparation plants.

Unlike the ash, sulphur in the coal is converted into gaseous products and must be scrubbed from the synthesis gas. Because this is an expensive operation, the bulk of the removable sulphur must be washed from the coal before gasification. Estimates by the Bureau of Mines indicate that the cost of removing sulphur from synthesis gas is \$1.15 per ton of gasified coal when the sulphur content of the charge is 3 pct. Increase in the sulphur content of the gasified coal from 3 to 5 pct would impose an estimated additional cost item of 37 cents per ton of coal for the removal of sulphur from the synthesis gas. Total cost of gas purification, using a 5 pct sulphur-content coal, amounts to \$1.52 per ton of coal. Even the use of low-sulphur coal containing 1 pct sulphur would require a capital investment of almost four million dollars in purification equipment in a gas-synthesis plant producing 10,000 bbl per day and a total of 75 cents per ton of coal gasified, or 34 cents per barrel of oil for the removal of the sulphurous products. Of course, the recovered sulphur may have a market value which can be applied against the cost of removal, but even under the most favorable pricing system the cost of gas purification will in most instances justify cleaning the coal. Thus it is easy to see the economic advantages of cleaning the coal used in the Fischer-Tropsch process to a low-sulphur content if the sulphur can be removed by mechanical cleaning. Examination of coals in Ohio, Indiana, Illinois, and western Kentucky shows that the sulphur content of those available in large quantities is high, in some instances exceeding 5 pct in the raw coal. Even after intensive treatment for sulphur reduction, the sulphur content of the washed coal would be in the range of 1.5 to 3.0 pct. Within these normally high-sulphur areas, there are rather sizable deposits of low-sulphur coal. The low-sulphur and low-ash bed condition of the Illinois No. 6 coal in the Franklin-Williamson district is well-known. Other areas in the region undoubtedly are responsive to sulphur reduction at low-gravity separations.

Sulphur reduction for gas-synthesis coal appears

to be a much more difficult problem than ash reduction for the hydrogenation process. Many of the high-sulphur coal beds contain significant amounts of sulphur in those forms that are not removable by gravity separation. Very fine crushing will reduce the sulphur content so that the float product will contain little more than the organic sulphur. Where the amount of organic sulphur in the coal remains excessively high, the problem of further reduction by chemical means is outside the coal-preparation field. The financial return to the conversion-plant operator through the decrease in cost of expensive gas-purification equipment seems large enough to encourage the coal-preparation industry in developing methods to reduce the sulphur content of the clean coal to a minimum. In the low-gravity range, separations of many American coals have shown that significant sulphur reduction does not occur until the separating gravity is lowered to about 1.30. With these coals, the sulphur content in the 1.35 sp gr float product is approximately as great as would result from a 1.60 sp gr separation. Even at the low separation point of 1.30 sp gr, the yield of float material would be in the range of 50 pct of the raw feed. Unlike the Bergius process, the Fischer-Tropsch does not require sizable quantities of auxiliary fuel. This poses a problem in the disposal of a middling product. About 90 pct of the coal requirements for the gas-synthesis plant is used in producing synthesis gas, therefore the small yield of float coal resulting from the low-gravity separation would have to be justified almost entirely on the basis of the decreased sulphur content, as the plant requirements for a secondary fuel are not sufficient for utilizing the large amount of middling material produced in the washing process.

**Uniformity of Plant Feed:** Perhaps one of the most rigid requirements will be uniformity of the process fuel for both the hydrogenation and gas-synthesis methods. The addition of oxygen and steam to produce synthesis gas undoubtedly will be a closely controlled operation that will necessitate a fuel uniformity seldom imposed by the average fuel user. Likewise, in the hydrogenation process, variations in ash content and petrographic constituents would materially affect and change plant operating conditions. To meet this requirement, it may be necessary to wash certain coals that appear satisfactory in other respects.

**Economics of Low-ash Coal in Hydrogenation:** In coal hydrogenation, an important operating factor due to the ash content of the coal is loss of oil in eliminating the ash and other nonliquefiable portions of the coal after the first stage of the process. The loss increases with ash content of the coal and results in increased consumption of coal, hydrogen, and power.

To determine the economic limits for cleaning coal for hydrogenation due to this one factor, the Coal-to-Oil Demonstration Branch of the Office of Synthetic Liquid Fuels, Bureau of Mines, studied the increased oil yield with decreasing ash content of the coal. A comparison was made of the economic advantages resulting from the use of 2.3 pct ash coal from Rock Springs, Wyo., and a 6.0 pct ash coal from the same mining field in a 30,000 bbl per day hydrogenation plant. It was discovered that using the lower-ash coal produced a savings of 2.8 pct of coal equivalent to hourly savings of 6.5 tons of coal on the moisture and ash-free basis. More important

than this saving in coal consumption is the reduction in capital costs for the 30,000 bbl per day plant by 3 million dollars when using the lower-ash coal. This saving of 3 million dollars is further reflected in the daily plant operating costs and amounts to \$1400. per day or \$58. per hour. With this information, it is then possible to calculate the total savings when the ash content of the hydrogenation coal is reduced from 6.0 pct to 2.3 pct ash and from those calculations to estimate the highest permissible washing cost in cents per ton for each unit ash reduction that would be justified by the decreased coal consumption and the reduction in plant operating costs. The value assigned to the 6.5 tons of moisture- and ash-free coal saved, using the 2.3 pct ash coal instead of the 6.0 pct ash coal, would depend on the cost of coal to the plant on an as-received

and ash-free coal, so allowances must be made for the ash and moisture in the coal as purchased and for the washery loss in cleaning the coal.

Table I shows the maximum allowable cleaning costs in cents per ton of coal as received for each per cent of ash reduction, using coals containing 10, 20, and 40 pct ash plus moisture and a washery recovery of 50 and 70 pct of the raw feed as clean coal. A large proportion of the remaining 50 and 30 pct, respectively, may be recovered as middlings and utilized as powerhouse coal.

These calculations consider the decreased fuel value of this middling product when used in the powerhouse. This adjustment is reflected in the aforementioned hourly savings of 6.5 tons of moisture- and ash-free coal when operating with the 2.3 pct ash content Wyoming coal.

**Table I. Maximum Allowable Washing Costs for Different Coals for Each Unit of Ash Reduction, Cents per Ton of Raw Coal, As-received Basis**

Yield of Washed Coal, Pct	50			70		
	10	20	40	10	20	40
Ash Plus Moisture in Raw Coal As-received, Pct						
Maximum allowable cost of washing for each unit of ash reduction, cents per ton:						
Based on moisture- and ash-free coal valued at \$2.00 per ton (7 cents per million Btu) .....	3.8	3.4	2.5	5.3	4.7	3.5
Based on moisture- and ash-free coal valued at \$4.00 per ton (14 cents per million Btu) .....	4.5	4.0	3.0	6.3	5.6	4.2
Based on moisture- and ash-free coal valued at \$6.00 per ton (21 cents per million Btu) .....	5.1	4.6	3.4	7.5	6.6	5.0

basis. Assuming the value of the ash- and moisture-free coal to be \$2.00, \$4.00, and \$6.00 per ton (this being equivalent to 7, 14, and 21 cents per million Btu) the 6.5 tons of coal per hour saved would result in an hourly dollar savings of \$13., \$26., and

Using a cost figure of 15 cents per ton of coal feed to the washery as the operating cost of the washing plant, it is possible to calculate the minimum reduction in ash content for different coals to justify washing the coal. These data appear in table II.

**Table II. Minimum Reduction in Ash Content to Justify a Washing Cost of 15 Cents per Ton of Raw Coal, Per Cent**

Yield of Washed Coal, Pct	50			70		
	10	20	40	10	20	40
Ash Plus Moisture in Raw Coal As-received, Pct						
Reduction in ash content necessary to justify a washery cost of 15 cents per ton of feed coal:						
Based on moisture- and ash-free coal valued at \$2.00 per ton (7 cents per million Btu) .....	4.0	4.4	6.0	2.8	3.2	4.3
Based on moisture- and ash-free coal valued at \$4.00 per ton (14 cents per million Btu) .....	3.4	3.8	5.0	2.4	2.7	3.6
Based on moisture- and ash-free coal valued at \$6.00 per ton (21 cents per million Btu) .....	2.9	3.2	4.3	2.0	2.3	3.0

\$39., respectively. Adding these figures to the fixed savings in plant operating costs of \$58. per hour, the total hourly savings would become \$71., \$84., and \$97., depending on the value assigned to the coal. Dividing these savings by the 230 tons of moisture- and ash-free coal processed hourly in a 30,000 bbl per day plant reveals a savings of 31, 37, and 42 cents per ton of coal. These reductions in cost occurred through reduction of ash content from 6.0 to 2.3 pct, or 3.7 pct. This figures to a maximum allowable cleaning cost of 8.4, 10, and 11.4 cents per ton of process fuel for each 1 pct reduction in ash content. However, these figures are based on moisture-

These data, showing the beneficial effect of decreased ash content in the hydrogenation coal, reflect the advantages in only one phase of the process. Washing the coal usually decreases its fusain content and might materially improve the coal-to-oil conversion period. In addition, ash constituents tend to settle in the converters, with harmful effect. The influence of the ash content of the coal with reference to these latter two factors will depend on the properties of the individual coal and can only be determined by experimentation. Undoubtedly, when these benefits have been evaluated, the economy of the coal cleaning will be enhanced.



# A Classification and Application of Drill Jibs

For Rock Drill Mounting

by

O. J. Neslage

and

R. W. Jenkins

The need for mechanized drilling to decrease mining costs has resulted in the development of the jumbo from column-and-bar drill carriages to hydraulically controlled jib jumbos. Resultant savings from use of improved jumbos have been high enough usually to retire the machinery investment within a year. The jib jumbo unit has been applied successfully to development and preparation programs and flat-bedded stopes, and an even wider field is forecast for its use.

**M**INE operators today are paying increased attention to the mechanization of drilling operations as an important factor in increasing efficiency and decreasing costs. Recent progress in the design of drill jibs for mounting on drill carriages, or jumbos, has done much to advance the mechanization of mining methods. This progress has introduced new opportunities in the application of "jib jumbos" that are worth serious consideration. An attempt will be made here to classify the various types of jumbos and drill mountings available for level development and flat stope work, to indicate applications, and to measure their effectiveness in relation to their cost. The data here are based on a

O. J. NESLAGE and R. W. JENKINS, Members AIME, are Vice President Commercial Sales and Special Representative, respectively, Joy Manufacturing Co., Pittsburgh, Pa.

AIME Columbus Meeting, September 1949.

TP 2841 A. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received Aug. 26, 1949.

wide variety of field usage, and are offered only to point up, in a general way, the relative efficiency in proportion to cost. It should be remembered that any contemplated jumbo installation must be studied to determine what type of installation is most suitable, considering ground conditions, loading, timbering and haulage equipment, service available, and attitude of manpower.

**Early Drill Carriages:** The advantages of some sort of a drill carriage were recognized at the time rock drills were introduced. In drift or heading operations where a fairly standard drill round and cross section were anticipated, as well as in flat bedded deposits where standard slabbing rounds were used, track and crawler-mounted drills found an early application. These first drill jumbos were little more than drill carriers, designed locally and manufactured with materials at hand. Of comparatively simple construction, function, and low capital cost, they usually consisted of a hinged column which supported one or more drills on arms or universal bars. When the column was pivoted vertically, the drills were moved into position. Screw jack extensions reached from the column to the roof and floor to hold the drill rigid. In some cases, particularly in tunnel jumbos, enough weight was carried that rail clamps or roof jacks were sufficient.

Many of these column and arm-type jumbos, usually equipped with air and water manifolds,

steel racks, and staging for drillers, are still in use today. Some have employed air or hydraulic extensions for securing the columns. They offer the time-saving advantage of having all the necessary equipment together, and they can be moved in and out, set up and torn down as a unit.

**Boom-type Jumbos:** When the advantages of the column-type jumbo were seen, operators began to realize the further gains possible. More time could be saved with mechanical assistance to position the drill vertically. A rack on the column and a pinion on the arm for raising and lowering the drill in the manner of a quarry bar were tried with some success. The most outstanding solution to the vertical handling problem was reached with the boom-type of jumbo. Here two or more machines were mounted

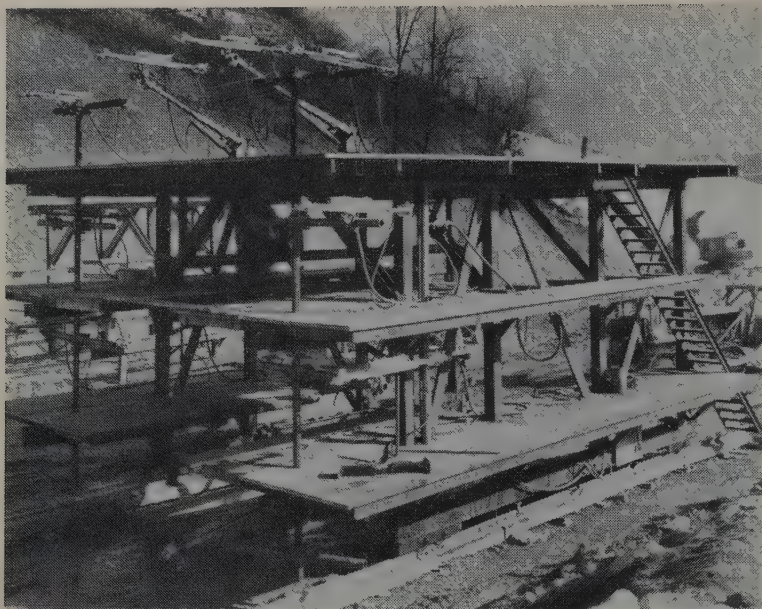


Fig. 1—Self-propelled, rubber tire-mounted "drill-mobile" with two hydrodrill jibs in a flat stope in a Missouri lead-zinc mine. Drills are  $2\frac{7}{8}$ -in. bore sinkers mounted on long feeds.

on a cross arm at the end of a boom that was pivoted at the rear of the track. This was raised and lowered by various mechanical aids. Occasionally air or hydraulic power was used. Nearly all of today's wagon drills use this boom-type principle.

Boom jumbos represent a savings in time and physical effort expended in raising and lowering the drills. The boom must be balanced by the weight of the truck which is secured by rail clamps or roof jacks. The most successful application of this type of jumbo has been in the Tri-State area where multiple long-feed machines are used on caterpillar-mounted, boom-type jumbos for slabbing





**Fig. 2—Jumbo for a western Pennsylvania vehicular tunnel with hydrodrill jib mounting for top center drills to make high center holes easy to reach and to make possible rounding out the arched roof. Drills are 3½-in. drifters on 48-in. automatic feeds.**

operations in large, flat stopes (fig. 1). Since the machines must be moved in and out horizontally on the cross arms by hand, they are best applied where rounds can be laid out with a minimum of lateral drill movement. In headings where the cross section is rectangular and the holes may be laid out in vertical rows, or where the "fanning" of holes can be tolerated, boom-type jumbos can be used effectively (fig. 2). The rounds used are the result of a compromise between the ground conditions and the limitations of the jumbo.

The greatest disadvantage of column and arm, boom-type jumbos lies in their lack of flexibility in arch or horseshoe section headings. Where "fanning" of holes resulted in overbreak and wall shattering and caused safety hazards, excessive timber support, dry packing, or "no pay" concrete filling was required. Moving the machines laterally had to be done by hand, wasting much time and physical effort.



**Fig. 4—Tunnel jumbo mounting 10 hydrodrill jibs with 3-in. on 36-in. feeds on a roadway tunnel job in the Hawaiian Islands.**



**Fig. 3—Special wide mounting for two hydrodrill jibs in a wide drift in a Tennessee zinc mine. One miner drills and loads a 23-hole round in this large heading in a single shift.**

and ease of operation. Vibration, corrosion, and the abrasive action of drill cuttings caused wear and looseness in the screw assembly which was reflected in excess drill maintenance and steel breakage.

The next step in jib development came from the coal mines where the same problems had been met with coal drills, universal tire-mounted cutters and mobile loaders. Problems there had been solved largely by the introduction of hydraulically powered equipment. It was a logical move to hydraulically operated jibs with the jib raised and lowered by a hydraulic cylinder which, for protection, is placed under the boom. Hydraulic pressure is applied by a hand-operated or air or electric-driven pump of simple construction. With this increase of available power, the arms can be made longer to cover greater face widths. The flexibility of the hydraulic system is further utilized by mounting swing cylinders on the jibs for positioning horizontally. All hydraulic cylinders are operated from the same pump by controls conveniently mounted on the jibs.

**Advantages of Hydraulic Control:** The superiority of hydraulic power was further evident from a safety standpoint. The cylinder supply pipes were designed so that if breakage occurred in the external



**Table I. Classification of Jumbos, Other Than Wagon Drills**

Type	Rigidity Secured by	Setting Up and Tearing Down	Positioning Machines	Example
Fixed col. and arm	Weight and anchor to rails	Swing arms ahead and raise on col.	Moving drills and arms by hand	Tunnel jumbos
Fixed col. and arm with mast	Weight and anchor to rails	Add mast and as above	Moving drills and arms by hand	ACM type
Pivoting col.	A. Weight and anchor to rails	Col. tipped vertically and secured		
	B. Screw jacks on cols.	Col. tipped ahead and screw jacks extended	Moving drills and arms by hand	Early Coeur d'Alene type
	C. Air-piston type cols.	As above except air bar extended	(1) Moving drills and arms by hand (2) Rack on col. and pinion on arm	Present Coeur d'Alene type Quarry-bar type
Boom type	Usually use rail clamps or roof jacks	Secure jumbo at face	Position machines vertically by raising or lowering boom. Move in and out on arms by hand.	Southwest type also Tri-State models
Jib type	Weight and hydraulic roof jack	A. Air hand-hydraulic roof jack, brakes on wheels B. Air hydraulic power on jack brake on wheels	Jib is swung by hand. Raised by hydraulic cylinder on boom Air or electric hydraulic raise and swing	Hydraulic jib Power-type hydraulic jib
	Hydraulic jack	Position and tighten jacks	Hand swing on booms. Hand-hydraulic vertical cylinder on cam	Screw-type jib
	Hand-screw jacks	Position and tighten jacks	A. Hand swing, hand screw on booms B. Hand swing, air-power screw on booms	Screw-type jib Screw-type jib

pipings, the jib could not fall so fast that the operator could not clear himself. This is a danger that is present in any screw-supported unit subject to high fatigue stresses from vibration.

Hydraulic jib mountings have been particularly successful in the elimination of overbreak. The ease and rapidity with which a miner may position his holes, allow him to drill rib and back holes parallel to the direction of advance and on the section line. Hydraulic jib units mounted on the top deck of tunnel jumbos to eliminate overbreak on the arch section have been extremely successful (fig. 3, 4, and 5).

The hydraulically operated drill jib provided a flexible yet stable support which effectively reduced the time and effort required to position drills. Since the weight of the drill was no longer an important factor, operators saw that other advantages could be gained by using a machine of sufficient feed length to drill the hole in one run. Numerous steel changes were no longer necessary. The rapid advances being made in the development of tungsten-carbide bits at this time answered the problem of too frequent bit changes. This new bit, along with small gauge, carbon-steel bits, made even greater penetration possible.

**Long Feeds:** The economy of long feeds is evident when one or more steel changes per hole are eliminated. Where a three or four change round can be drilled with only two changes, the savings in time and labor are apparent. The greatest economy is attained when the hole can be drilled without changing steels at all. In many cases, the tungsten-

carbide bit or a small gauge, carbon-steel bit must be used to accomplish this.

The use of longer feeds offers many more advantages. The higher drilling speeds possible with smaller starting gauges are augmented by the elimination of steel changes. Less powder is consumed because it can be more evenly distributed throughout the nearly parallel walled hole. Greater blowing pressure in a smaller hole permits continuous operation, saving time and drill steels when drilling in broken or fitchery ground.

Because of the increased weight of the long feed drill, it must be used with either the jib- or boom-type jumbos. As previously mentioned, long feeds with boom jumbos have been successfully used in slabbing operations. Here, the machine does not have to be moved in and out on the arm. Jib mountings, however, are replacing booms even in slabbing operations.

**Self-propelled Jumbos:** A self-propelled, air-powered, pneumatic tired, dual or triple jib with long feeds for drilling 20-ft rooms is now available. In addition to power jib swing and power elevation, this unit can be equipped with extendable jib arms to maintain square room corners. Its close coupled, individually sprung

wheels make it exceedingly maneuverable in narrow rooms and on rough floor. Other models include a small single jib unit for use in small section drifts in conjunction with slusher loading.



**Fig. 5—Dual hydrodrill jib operated by one man in a 7x8 ft drift in a Montana lead-silver mine.**



**Table II. Comparative Efficiency, Labor Cost and First Cost of Various Type Jumbos Under Assumed Standard<sup>a</sup> Conditions**

Description of Unit	Length of Feed, In.	Setup and Tear Down (Hr)	Drill Time (Hr)	Total Time (Hr)	Eff. <sup>b</sup> of Drilling, Pct	Adv. Ft per Man Hour	Labor Cost per Ft Adv.	Saving Over A per Ft	Saving <sup>c</sup> in 1 Year	Approx. Cost, Dollars
"A" Col. and arm less carriage	30	1.5	6.2	7.7	30	0.90	2.22			800
"B" Col. and arm on carriage		1.0	6.2	7.2	30	0.97	2.06	0.16	400	1,100
"C" Boom on carriage		0.75	4.2	4.95	40	1.4	1.43	0.79	1,970	1,500
"D" Hydrodrill jib		0.5	3.4	3.9	50	1.8	1.11	1.11	2,800	1,800
"E" Same with long feed	90	0.5	2.9	3.4	65	2.0	1.00	1.22	3,050	2,300
"F" Long feed and carbide bits		0.5	2.5	3.0	75	2.3	0.87	1.35	3,370	2,500

<sup>a</sup> Assumed conditions  
1 man crew, 8x6 ft  
Heading 24-7 ft holes  
(168 ft drilling)  
1.5 fpm drill speed  
Labor \$2.00 per hr.  
Single drill used in  
all cases.

30 in. penetration,  
45 in. penetration,  
90 in. penetration,  
<sup>b</sup> Efficiency of drilling. Equal to:  
Drill footage in ft (168 ft)

$\% = \frac{\text{Drill speed (1.5 fpm)} \times \text{Drill time (min)}}{\text{Drill footage in ft (168 ft)}} \times 100$

<sup>c</sup> 1 year's program  
assumed to be  
2500 linear ft.

**Classification:** A classification of the various types of jumbos is shown in table I. Each type of jumbo is characterized, as described above, by the way in which rigidity is secured, how it is set up and torn down, and how the machines are mounted and positioned. An example is given for each jumbo type.

Table II is a chart which shows the comparative efficiency, as well as approximate labor and equipment costs, of various jumbo types. It shows, too, possible feed lengths, drilling times, and relative savings as based on more or less standard conditions as noted on the chart. It is necessary to present these figures in a general way, since there are a great number of variables between installations such as ground hardness, bit size, drill size, etc. All figures are derived from actual field conditions. Drilling efficiency here is assumed to be the ratio between "full throttle ahead" time to the overall drilling time including setup and drill-positioning time.

**Savings with Jumbo Usage:** Like other items of capital equipment, the initial price of refinements must be reflected in lowered operating costs in order to maintain a constant return on this investment. The increases in operating efficiency obtained with the various jumbos are excellent examples of improved returns as the capital investment is increased. Table II compares the various drilling units on bases of advance per man hour, drilling labor, cost per foot, capital price, and savings as compared to the lowest priced unit. An outstanding example is the case of the jib-jumbo units. Here, labor savings alone retire the capital investment in a year's program of 2500 linear ft.

Field experience in the use of jib-type units has verified these theoretical assumptions. A large western copper mine reduced contract labor costs from \$2.75 and \$3.35 per foot for 8x6 ft and 10x8 ft sections to \$1.70 and \$1.90 respectively. Another operator has maintained cost of jib-jumbo drifts at \$5.00 per foot overall while the distance to the face has increased to 3000 ft. Ordinarily the price would have been increased to \$6.50. Direct labor savings on jib-jumbo applications have run from \$1.50 to \$5.00 per foot. Advances per man shift of 4.67 ft in double headings and 3.6 ft in single headings of 9x7 ft cross section have been reported.

**Applications of Jib Jumbos:** All these savings are

of course dependent on effectively using the time saved. Basically, if a machine is capable of saving a certain amount of money per foot of heading, the more it is used the more it will save. Utilizing this principle, many large operations have laid out their development and preparation programs to keep the jumbo constantly working on several headings rather than standing idle while mucking. One example of this program in a western mine is driving five headings, making three rounds on one shift and two on the other, using a dual jib jumbo and two shovel loaders. Weekly footage averaged between 150 to 180 ft in 10x9 ft headings. Three rounds could have been made on each shift except for delay in waiting for clean faces.

Another example of cost reduction by keeping a jib jumbo constantly at work is in bedded deposits where special drill crews on mobile drill units move from one heading to another. Loading and blasting are done by another crew. In this way, drill footage per machine shift has been doubled with no increase in labor.

Frequently, mine development must be done in single headings where the jumbo is idle during the mucking portion of the round cycle. Under these conditions, the higher drilling rate of the jumbo may be used to complete a longer round during the same cycle: the recent interest in burn cut rounds has developed techniques applicable to nearly all ground conditions for pulling to depths of 10 to 15 ft. Where ventilation and other safety precautions permit blasting during shifts, the jib jumbo is an effective means of increasing the number of rounds per shift in a single heading.

In closing, the hydraulic jib type is the most advanced of drill mountings and is the first unit to justify high production loading and transportation equipment for mine heading work. Its mechanical ability to support drills of all feed lengths and to ease the handling has been reflected in increased output per machine shift. To utilize properly this increased output, a thorough examination of every aspect of the work contemplated is necessary. Proper organization with planned service and cycling will increase advance and lower costs to a point where today's concept of cost on development work may be completely changed.



# Production and Marketing of Garnet Abrasive Sands

From Emerald Creek, Benewah County, Idaho

**Occurrence:** small crystals in alluvial sands from the eroding Belt Series mica schists. **Flowsheet:** dragline, trommel screen, jigs, drier, crusher, screens. **Value:** ground, \$2.50 per cu yd, garnet sand gross \$60. per ton. **Costs:** production and sales \$48. per ton. **Uses:** blasting sand in aircraft plants, monumental works, foundries. **Efficiency comparison:** superior to silica sand. **Future:** possibilities only touched upon, research continuing.

by John S. Crandall

**T**HE mineral garnet, while ordinarily considered a semiprecious gem stone or a second-grade industrial gem, has also proved itself in the field of industrial abrasives. Its use is well known as a sand-paper grain, and as a sandblasting sand its qualities are rapidly becoming recognized in more and more industries.

Production of garnet as an abrasive is confined chiefly to two areas in the United States, North Creek, N. Y., where the Barton Mines Corp. operates, and Emerald Creek, Benewah County, Idaho, where

*JOHN S. CRANDALL is Manager, Idaho Garnet Abrasive Co., Spokane, Wash.*

*Third Annual Northwest Industrial Minerals Conference sponsored by the Columbia Section in cooperation with the Oregon and the North Pacific Sections, AIME, May 1949.*

*TP 2844 H. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received May 26, 1949.*

the Garnet Mines, Inc., and the Idaho Garnet Abrasive Co. with mills at Fernwood, Idaho, operate.

This paper is confined to a discussion of production and marketing of garnet sand by Idaho Garnet Abrasive Co.

**Occurrence:** Garnets in the Emerald Creek area occur as disseminated crystals in beds of micaceous schists of the Belt Series, which in this section are estimated to be close to 4000 ft thick. The schists are high in alumina and silica with iron, manganese, and magnesium. Subjection of the original sediments to high temperatures and pressures caused metamorphism to take place with the resultant recrystallization of high alumina-silica minerals such as garnet, mainly spessartite and almandite varieties, cyanite, sillimanite, chlorite, actinolite, tourmaline, biotite, and muscovite, with minor amounts of ilmenite and magnetite. Quartz is also present in considerable amounts.

Fast erosion of the soft mica schists on exposure to weathering has created extensive alluvial deposits containing up to 10 pct garnet having a maximum grain size of 3/16 in. These alluvial sands and gravels are now being treated for the recovery of garnet sands.

**Treatment:** Overburden of 1 to 4 ft must be stripped to expose the garnetiferous gravels. This operation and the subsequent feeding of the gravels to a trommel-screen washing plant are performed by a 3/4 yd dragline. The trommel-screen openings are 3/16 in., thus allowing a separation and concentration based on grain size, since over 95 pct of total free garnets are minus 3/16 in. All plus 3/16-in. material is wasted at this point. The minus 3/16-in.

material is further concentrated in a sand-drag classifier, where the slimes and silts are washed out and wasted. The sand product from the classifier varies in garnet content from 20 to 60 pct according to the particular section of ground being worked.

This sand product is trucked to a jig plant where two sized fractions are made in a trommel-screen. The minus 3/16-in. plus 10-mesh portion is fed to a Pan-American two cell 42-in. jig. The minus 10-mesh portion is treated in a Bendelari three cell 42-in. jig. The jig concentrates are combined to form a 98 pct garnet sand. The jig tailings contain 3 to 5 pct garnet which is mainly flat crystals and chips which will not settle into the jig hutch. Subsequent treatment of these tailings in a scavenger jig followed by drying and electromagnetic separation will, according to tests, reduce the garnet losses in the tailings to something around 1 pct.

Jig treatment of this feed approaches ideal as the major portion of the garnet crystals are the natural dodecahedrons and so are, in general, close to spherical. The specific gravity of pure garnet is 4.2, while the next heaviest mineral in the feed is cyanite with a specific gravity of 3.6, then quartz with specific gravity of 2.6. The garnet concentrate is practically free of quartz. The predominant impurity is cyanite which amounts to about 1.5 pct. The rod-like crystals of cyanite appear to up-end in the jig and go into the hutch with the garnets. Some ilmenite and magnetite appear in the concentrate but in very minor amounts. Subsequent washing in a sand-drag classifier removes fine silts and iron oxides.

The gravel feed to the washing plant will average 8 pct recoverable garnet content. Concentration ratio in this plant runs about 2.5 to 1. Washing-plant concentrate as fed to the jigs will average 45 pct garnet by weight. Concentration ratio of jigging runs about 2.2 to 1.

The garnet concentrate is dried in a rotary oil-fired drier and then fed to vibrating screens in closed circuit with crushing rolls.

Practically any grit from 10-mesh down to 150-mesh grain size may be graded to specifications in two 3-deck vibrating screens. The present production, however, is approximately 75 pct No. 36, 15 pct No. 60, and the balance No. 80 and No. 100. Metal-screen cloth is used for sizes down to 36 mesh. From 36 mesh and finer, silk-screen cloth is used since it has less tendency to blind.

All garnet sand is bagged in 100 lb self-sealing, sleeve-type paper bags. Practically all shipments are made in carload lots. Car loading is convenient since the plant is in Fernwood on the tracks of a branch line of the Milwaukee railroad. Truck shipments can and are made occasionally.

**Values and Costs:** The garnet content of the ground now being worked will average approximately \$2.50 recoverable per yard, including overburden. The finished garnet sand loaded in the cars has a gross sale value of \$60.00 per ton.

Production costs are all figured on garnet-sand product rather than on feed to any of the plants, and are as follows: stripping, washing, and trucking from the mine on Emerald Creek eight miles to the jig plant at Fernwood, \$13.00 per ton of garnet sand. Treatment in the jig plant, \$5.00 per ton of garnet. Treatment in the screening plant, bagging, and loading in cars, \$10.00. A total of these costs is \$28.00 per ton of finished product. This figure takes into account a 10 pct loss of sand due to the unavoidable generation of minus 100-mesh sand in crushing, which at the present time is being wasted because of lack of market.

A royalty of \$3.00 per ton is paid on all garnet sand shipped from the plant thus bringing the total production cost to \$31.00. Overhead, road maintenance, etc., along with a 20 pct sales discount bring the actual total production and sales cost to \$48.00 per ton of finished garnet sand.

It is expected that mill improvements and changes made this spring will materially cut the production costs.

The screening plant has a present capacity of one to two tons per hour of operation, varying according to fineness of grits being produced.

**Marketing:** The Emerald Creek garnet has a hardness of 7.0 to 7.5 and has a typical gem-garnet color. A very few of the larger stones found in the area have been cut into ring and jewelry settings of considerable beauty and durability.

Tests and actual use of garnet sands in sand blasting show that the finer grits have a greater cutting efficiency than coarser sized, probably because of the greater number of cutting edges per unit of the sand. Crushed sand also has greater efficiency than the uncrushed sand of the same mesh sizes, probably because the cutting edges are actually sharper. The most efficient blasting sand appears to be a mixture of crushed fine and coarse grains, the maximum particle size being determined by the work to be performed.

Up to the present time all of the garnet sand produced in the Fernwood plant has been used as a sand-blasting abrasive. The principal users have been aircraft industries, both manufacturing and maintaining, U.S. Navy, U.S. Air Force, ship builders, monumental works, bronze, brass, and magnesium foundries. Garnet sand has also been used extensively to clean spark plugs in service stations, and for many other miscellaneous metal-cleaning jobs. Some glass designers have found the sand satisfactory as a material for etching and frosting.

Some experimental work has been done with marked success in glass and lens grinding where it is used as a secondary abrasive before polishing. Garnet sand with a fineness of minus 270 mesh has been tried as a denture polish in several dental laboratories and shows great possibilities. All of these latter uses require very fine material which must be very carefully and exactly graded to avoid scratching caused by oversize particles.

At present all material is graded in vibrating screens and with silk-screen cloth. The development of an appreciable market in the mesh sizes below 100 mesh will necessitate the use of air classification

with the possibility of hydraulic classification in the very fine sizes, such as minus 325 mesh down to 700 mesh. Air classification would start at 65 mesh and would consist of a series of cyclones arranged in number and size according to the various grain-grading requirements of the market.

To date the market possibilities of Idaho garnets have just been scratched. Experimental work is being carried on by the Idaho Garnet Abrasive Co. and also by many users of the material. The following is a summary of the findings of one of such experiments performed by the Ryan Aeronautical Co. as a comparison of garnet sand and silica sand:

Direct cost: Garnet grit	0.00536 cents per part
Silica sand	0.00515 cents per part

The following advantages are not shown as direct material costs:

1. Smaller storage area required for garnet grit for the same number of working days.
2. Garnet grit will not draw moisture and may be used when wet.
3. Operator's vision is not obscured by dust.
4. The health hazard of silicosis is eliminated (where not cutting silica rock).
5. Replenishment of grit during operation is lowered 43 pct resulting in longer operating time.
6. Hauling charges and removal of residue is lowered approximately 50 pct.
7. 600 pounds of garnet grit is equal to 1400 pounds of silica in working efficiency.

An excerpt from a letter from the Glenn L. Martin-Nebraska Co. concerning comparative tests in their plant is as follows:

"Primary tests have shown this material to be about 2 or 3 to 1 ratio in usage, but we intend to run a more severe test and watch it very accurately with the laboratory in charge."

The results of these tests led to a 600 ton order for garnet sand.

The Battelle Memorial Institute, Columbus, Ohio, made a fairly complete comparative test on Idaho garnet sand in 1947 using grit sizes No. 24 and No. 60 against silica sand of the same mesh grades. The following is the conclusion made from these tests:

The most outstanding advantage shown for the garnet abrasive over silica sand is its greater cutting speed. The average cutting speed of the coarse garnet is about 30 pct greater than that of the coarse silica, and that of the finer garnet is about 50 pct faster than the finer silica.

The garnet also shows a marked advantage over silica in that more abrasive action is obtained per pound of abrasive consumed. This is chiefly the result of faster cutting with equivalent breakdown rates. This greater efficiency of the garnet abrasive is an important consideration with regard to cleaning costs for the two types of abrasive materials. Not to be overlooked is the possible reduction of the silicosis hazard with the use of garnet.

**Conclusion:** The production of garnet abrasive sand from the Emerald Creek deposits has been increasing with growing demands of the market. Better and more efficient methods of production and refinement are constantly being investigated and tried. Also the possibilities of broader markets are being explored.

With a high quality product such as these garnet sands, there will ultimately develop another stable and profitable production of nonmetallic industrial minerals from the Pacific Northwest.



# Nonmetalliferous Mineral Resources in Arkansas

by W. B. Mather

Arkansas' nonmetalliferous deposits are of many different types. The principal types briefly described in this paper are: clays, shales and slates; silica deposits; limestone and dolomite; barite; gypsum; celestite; nepheline syenite; greensand; phosphate; vermiculite; fuller's earth; bentonite; diamonds; sand and gravel. The future development of these deposits is expected to effect profoundly the industrial development of the state.

ARKANSAS is not only a geologist's paradise, but it is also particularly fortunate in possessing a wide variety of mineral resources.

The future mineral production and mineral industry of Arkansas, like the other states in Mid-America, will probably depend to a large degree on efforts to increase the exploitation of the nonmetalliferous deposits of the state.

The term nonmetalliferous, as used in this paper, embraces all types of mineral deposits, except fuels,

---

W. B. MATHER, Member AIME, is Chairman, Division of Mineral Technology, Southwest Research Institute, San Antonio, Texas.

AIME San Francisco Meeting, February 1949.

TP 2845 H. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received Aug. 8, 1949.

---

water, and those deposits from which, after beneficiation, metals are ultimately derived. For example, bauxite is the ore of aluminum, but a considerable portion of the bauxite mined eventually is employed in the production of nonmetallic alumina, an important oxide in the abrasive and ceramic industries. The bulk of the production, however, becomes metallic aluminum and therefore this type of mineral resource is considered metalliferous and is omitted from this paper.

Arkansas' nonmetalliferous deposits are of many different types, too numerous to discuss in detail in a single paper; therefore, only their salient features are presented. Many types of deposits such as limestone, glass sand, novaculite, shale, and sandstone are exposed over wide areas, providing inexhaustible reserves. Other types of deposits, such as barite and tripoli, contain large reserves which certainly are not unlimited. Still other types of deposits, such as celestite, kaolin, and vermiculite, appear to occur in small isolated deposits containing relatively small reserves.

Of course, all of Arkansas' nonmetalliferous deposits probably have not as yet been recognized, nor have all of the known deposits been prospected, tested, and evaluated. Various types of deposits are too low grade or too limited in extent to be profit-

ably produced by today's standards. In the case of other deposits, the known uses are few in number, thus acting as a retarder to their development. Due to anticipated technologic advancements, the future demands of industry are not expected always to be the same as they are today, and therefore idle deposits or those of current limited production may readily become the important revenue producers of the future.

The types of nonmetalliferous deposits discussed here include: clays, shales, and slates; silica deposits including glass sand, novaculite, tripoli, sandstone, chert, and quartz crystals; limestone and dolomite; barite; gypsum; celestite; nepheline syenite; greensand; phosphate; vermiculite; fuller's earth; bentonite; diamonds; sand, and gravel. The location of the various types of deposits are presented on the accompanying map (fig. 1). Symbols indicate the general area of occurrence. Many deposits, such as limestone, shale, novaculite, and so on, are represented by formations which could only be properly shown on a geological map. In these instances only active deposits or those of exceptional merit are indicated by symbols on the map. A general northeast-southwest line divides the interior highland from the Gulf Coastal and Mississippi Alluvial Plains of south and east Arkansas.

The data presented in this paper are only a summary of a portion of a more comprehensive survey and study of the natural resources of the entire state.<sup>1</sup> The study was sponsored and underwritten by the Arkansas Power and Light Co., Little Rock, as a public service to the state and its residents. Permission to present some of the data and conclusions resulting from the survey is gratefully acknowledged.

Table I, prepared by the Arkansas Geological Survey from state severance tax records, presents the annual production secured from Arkansas nonmetalliferous mineral deposits.

## Clays, Shales, and Slates

**Clays:** It is realized that each method of classifying clays is prone to criticism. In this paper, Arkansas' clays of potential value are arbitrarily

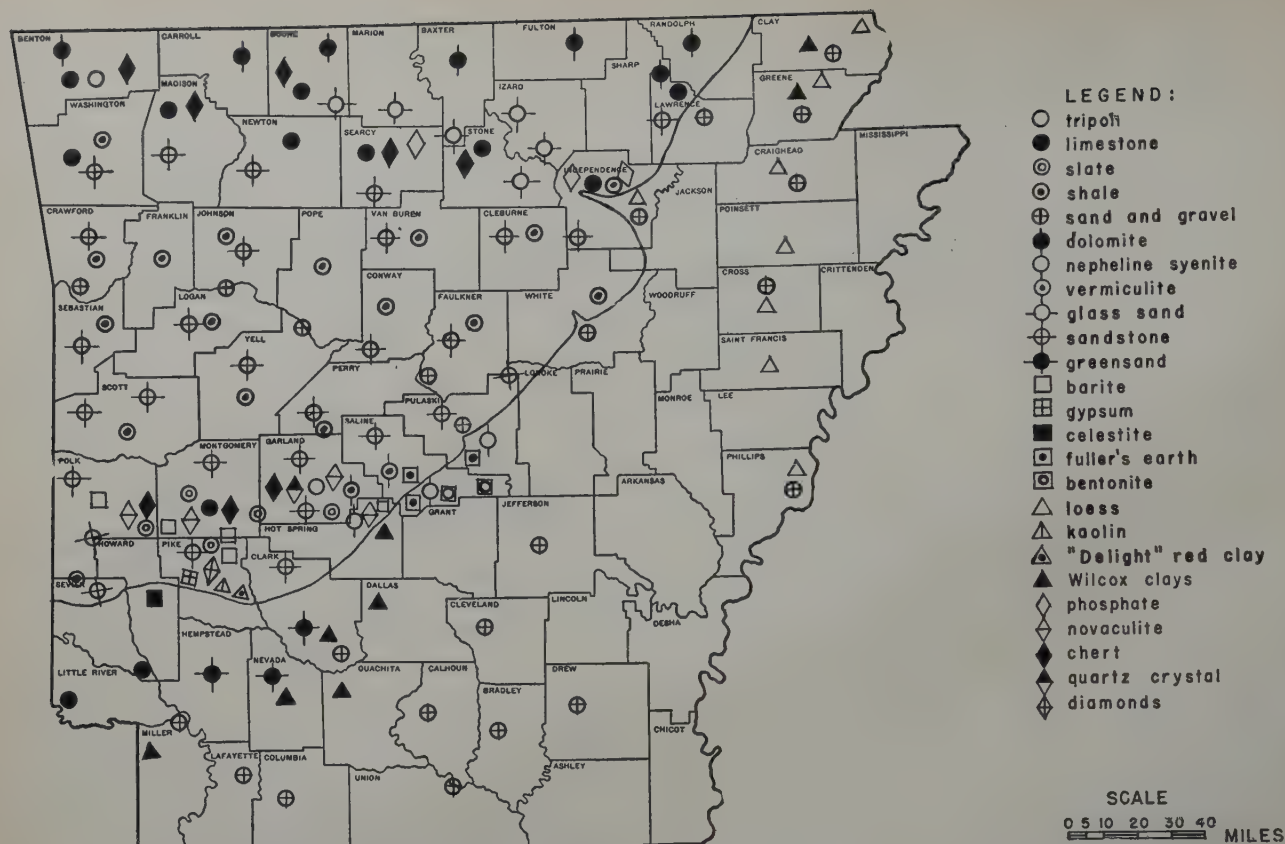


Fig. 1—Map of Arkansas with location of nonmetalliferous deposits.

NW-SE line separates interior highland and Gulf Coastal-Mississippi Plains.

divided on the basis of their position in the geologic column as follows:

1. Tokio clays of Cretaceous age exposed along the rim of the Gulf Coastal Plain in southwest Arkansas.

2. Wilcox clays of Tertiary age exposed along or a short distance from the edge of the Gulf Coastal Plain in south Arkansas and on Crowley's Ridge of east Arkansas. This group could also be extended to include Midway, Claiborne, and Jackson clays, but their value in the field of ceramics or industry in general is at present unknown.

3. Pleistocene loess, which forms a thick capping on Crowley's Ridge, is located in east Arkansas. It also occurs sporadically along the edge of the Mis-

issippi Alluvial Plain of east Arkansas. Residual and transported clays of Recent age occur throughout the state.

The following brief discussion presents the salient features of the clays in each of these groups:

1. Two clays, kaolin and "Delight" red, belonging to the Tokio formation of Cretaceous age and located in Pike County, are believed to be the most important members of the first group of clays.

a. The kaolin (fig. 2), which is chalky white to cream-colored, distinctly bedded, flat-lying, 18 in. to 6 ft thick, has been described by Herold.<sup>2</sup> It is exposed in isolated deposits of limited extent on or near hilltops between the towns of Murfreesboro and Delight. Individual deposits contain up to 10,-

000 tons. Future prospecting should reveal new deposits now hidden by overburden. Its present use, as an additive in the manufacture of refractories, is sporadic. It has been shown to be a superior raw material for the manufacture of vitreous whiteware or any of the common uses of kaolin, such as a white filler in paper and rubber.

b. The "Delight" clay is a red, fine-grained, plastic, nonrefractory, transported clay, exposed in the vicinity of the town of Delight. The deposit is about 12 ft thick and is underlain by sand and overlain by 3 ft of gray clay. This clay has very high shrinkage and warping tendencies which are detrimental to its use by ceramic manufacturing industries, except in the case of terra sigillatas. Where color, plasticity,

Table I. Annual Production from Nonmetalliferous Mineral Deposits

Mineral	1947		1948	
	Quantity	Value	Quantity	Value
Barite, sh. tons	376,861	\$2,411,910	362,403	\$2,681,782
Cement, bbls	252,209	421,482	1,150,000	1,978,000
Chert roofing granules, sh. tons	20,762	160,690	27,686	214,290
Clay products <sup>a</sup>		1,437,839		1,620,599
Raw clay, sh. tons	249,639		332,090	
Glass sand, sh. tons	188,652	371,644	140,485	276,754
Gypsum, sh. tons	45,742	101,089	67,543	149,270
Limestone, <sup>b</sup> sh. tons	518,254	381,030	349,940	594,898
Novaculite (abrasive), lbs	1,350,880	135,088	368,000	36,800
Sand and gravel, sh. tons	3,356,355	2,882,566	3,188,252	2,773,779
Stone				
Dimension stone				
Limestone, sh. tons	880	40,550	3,707	170,818
Crushed stone				
Novaculite ballast, sh. tons	103,901	96,628	125,056	116,302
Sandstone, sh. tons	683,470	635,626	865,301	804,730
Syenite roofing granules, <sup>c</sup> sh. tons	107,289	1,582,508	158,756	2,341,621
Tripoli, sh. tons	900	17,091	1,800	34,182

<sup>a</sup> Includes brick, tile and other heavy clay products

<sup>b</sup> Chiefly agricultural and chemical grade limestone but includes some construction stone

<sup>c</sup> Artificially colored



**Table II. Chemical Analyses of "Delight" Clay**

	Delight Red <sup>a</sup>	Kaolin <sup>2</sup>
SiO <sub>2</sub>	60.90	44.04
Al <sub>2</sub> O <sub>3</sub>	19.08	39.25
Fe <sub>2</sub> O <sub>3</sub>	7.20	0.68
CaO	0.20	0.26
MgO	1.41	0.21
Alk		0.13
TiO <sub>2</sub>	1.13	
H <sub>2</sub> O (140°)	1.60	
L.O.I. (1000°)		15.30

<sup>a</sup> Chemical analyses, unless otherwise specified, were made in our institute laboratories

and green strength are important, it may be used as an additive to ceramic bodies.

The chemical nature of these clays is revealed by the analyses given in table II.

The fusion point of the "Delight" red is between 2200°F and 2300°F, while that of the kaolin is cone 35 (3245°F). The latter is higher than most of the other kaolins in the country.



**Fig. 2—Twin Knobs kaolin deposit near Murfreesboro, Ark.**

2. The Wilcox clays of Tertiary age, recently described by Funnell,<sup>3</sup> may be subdivided into three principal types:

a. The first consists of ball-type clays, which normally exhibit the properties of white burning, refractoriness and plasticity; they are exposed in the vicinity of the town of Malvern near the east end of Magnet Cove in Hot Spring County and along the north portion of Crowley's Ridge. Present operators in the Malvern area have been mining clay from approximately a 300-acre tract. The thickness of the ball clay varies from 16 to 31 ft. Samples from several other undeveloped deposits in the area have also been tested. In addition to the present use of the clay as a blend in the manufacture of refractory brick and construction tile, it may be employed in the manufacture of chinaware, whiteware, and pottery bodies. The analyses in table III indicate the

**Table III. Chemical Analyses of Clays in Malvern Area**

	Acme	Malvern	
		Upper Gray	Lower Leafmold
SiO <sub>2</sub>	61.30	58.06	59.52
Al <sub>2</sub> O <sub>3</sub>	24.94	27.55	24.95
Fe <sub>2</sub> O <sub>3</sub>	0.46	0.61	0.52
CaO	0.24	0.18	0.26
MgO	0.22	0.41	0.37
Alk	0.83		1.09
TiO <sub>2</sub>	2.34	1.28	1.29
H <sub>2</sub> O (140°)	0.62	1.08	1.00
L.O.I. (1000°)	8.12	8.76	10.97

chemical composition of samples from two locations in the Malvern area.

This clay fuses between 2984°F and 3092°F.

b. The second type consists of sagger clays that normally exhibit refractoriness and plasticity. Arkansas sagger clays which are siliceous, refractory, and white burning are best exposed near Texarkana in Miller County and near Chidester in Ouachita County.

The following section is exposed in the Texarkana deposit (fig. 3):

4 to 6 ft—overburden

1 to 15 ft—white sagger clay

1 to 15 ft—yellow wad clay

base—lignitic blue clay and shale (Midway formation) suitable for manufacture of bloated lightweight aggregate

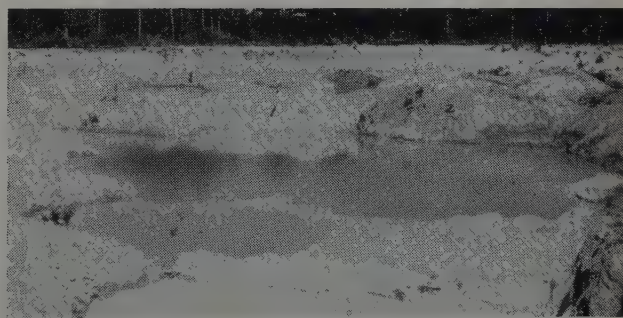
The Ouachita County deposit consists of alternating beds of white sagger clay, pure white sand, lignitic clay and is overlain by 18 in. to 2 ft of overburden. The white sagger clay beds vary from 4 to 8 ft thick (three beds).

These clays are suitable for the manufacture of sagger, pottery, flower pots, wall, floor and drain tile, stoneware and terra cotta, in addition to their present use as a blend in the manufacture of vitrified sewer pipe and white face brick.

The analyses in table IV present the chemical composition of these clays.

**Table IV. Chemical Analyses of Sagger Clay**

	Oneill Chidester	Dickey Texarkana
SiO <sub>2</sub>	69.64	69.16
Al <sub>2</sub> O <sub>3</sub>	18.28	19.00
Fe <sub>2</sub> O <sub>3</sub>	0.43	0.56
CaO	0.22	0.12
MgO	0.43	0.45
Alk	2.18	1.59
TiO <sub>2</sub>	1.33	0.88
H <sub>2</sub> O (110°)	1.80	1.36
L.O.I. (1000°)	5.45	5.98



**Fig. 3—Wilcox sagger and wad clay. Dickey clay pit north of Texarkana, Ark.**

1. White sagger clay. 2. Yellow wad clay.

The fusion point lies between 2900°F and 3000°F.

c. The third type of Wilcox clay is a wad clay (actually low-grade fire clay) with plasticity its principal property. Arkansas wad clays are best exposed in the Texarkana deposit previously described, where 1 to 15 ft of yellow wad clay underlies the sagger clay. This material is used presently as a blend in the manufacture of vitrified sewer pipe but could also be employed as a wad between saggars or as a sealant for patchwork in kilns.

Other Tertiary clays belonging to the Midway, Claiborne, and Jackson formations, exposed in the Gulf Coastal Region and the Mississippi Alluvial



Plain of south and east Arkansas, should be included here; they are only mentioned, however, because sufficient time could not be allotted to their study. The dark Midway clay underlying the Texarkana Wilcox clays, for example, is a suitable raw material for the manufacture of bloated lightweight aggregate. Jackson clay, in the vicinity of Monticello has been employed as a bleaching clay, and elsewhere as raw material in the manufacture of brick and tile. Generalizations regarding these clays, however, cannot be made until they have been studied in more detail.

3. Loess (reaching a thickness of over 100 ft) caps Crowley's Ridge in northeast and east Arkansas. Small sporadic occurrences of loess are also located along the edge of the Mississippi Alluvial Plain of east Arkansas. During the early years of the present century loess was employed in the manufacture of brick at many locations along the Ridge—at Paragould, Blytheville, Forrest City and Helena, etc. Today the best opportunities offered for the development of these deposits lie in their employment as a raw material for the manufacture of bloated lightweight aggregate.

Residual and transported clays of Recent or Quaternary age derived from relatively recent erosion and sedimentation are found throughout the state. Many of these deposits are too small and too variable in composition to be of any known value. Red "tallow" clays, for example, developed through the weathering of Paleozoic limestones and dolomites, occur in small and often thin, sandy or cherty, erratic deposits. The "buckshot" clay of Quaternary age of central, south, and east Arkansas is extensive in area but highly variable in composition; in the early years of the present century, it was occasionally employed in the manufacture of brick. Thick beds of Quaternary red clay, along the Red River in Hempstead County of southwest Arkansas, have proved to be an excellent raw material for the manufacture of bloated lightweight aggregate. Pink colored clays, derived from the weathering of manganese-rich rocks and perhaps suitable as a pigment, occur in small deposits in the Batesville area. A detailed study of these and similar types of clays is a matter for future consideration.

Since the bauxite deposits were not studied during this survey, the accompanying kaolinitic clays were only examined in a cursory manner, and therefore are not discussed in this paper.

**Shales:** Paleozoic shales occupy a prominent position in the stratigraphic column of Arkansas. They range in stratigraphic position from phosphatic Cason of Ordovician age and black Chattanooga shales of Devonian-Mississippian age to the thick Atoka formation of Pennsylvanian age. The thickness of the various shale-bearing formations range from less than a foot to over 6000 ft (Stanley and Atoka). They are interstratified with sandstone, limestone, coal and novaculite beds and lenses. They occupy a large portion of the northwestern half of the state, in the Ozark Mountains, Arkansas River Valley, and in the Ouachita Mountain region. Their attitude ranges from vertical to horizontal, and their composition may vary from arenaceous and calcareous to phosphatic and carbonaceous. Various horizons have been used in the manufacture of brick and tile. Certain horizons in the Atoka formation are presently employed as a blend in the manufacture of vitrified sewer pipe. Other horizons, espe-

cially in the Moorefield and Atoka, have been found suitable for the manufacture of bloated lightweight aggregate. The phosphatic Cason and black carbonaceous Chattanooga, Fayetteville, and similar shales should be prospected as a possible source of uranium.

**Slate:** Shales of Ordovician, Silurian, Devonian, and Mississippian age, in the Ouachita Mountain region of central and west central Arkansas, have been converted into slates through the metamorphism which accompanied the mountain building of the area. These, of course, are highly contorted and are interbedded with sandstone and novaculite. They have been utilized in the manufacture of roofing shingles and at one locality (near Caddo Gap) are being used in the manufacture of roofing granules. Slates and shales of Saline County, locally converted into soapstone, were formerly mined for use as a lubricant. These deposits are available for any industry requiring a low grade form of talc, as in the production of foundry facing, insecticides, and possibly in the manufacture of roofing, paper, and paints, as well as in ceramics. Highly carbonaceous horizons should be checked to determine if they contain uranium.

### Silica

Arkansas' silica deposits are divided into six types; glass sand, novaculite, tripoli, sandstone, chert, and quartz crystals. They have all been of some economic importance in the past, but it is anticipated that the greatest future development will probably be confined to the first three types of deposits.

**Glass Sand:** The St. Peter sandstone of Ordovician age is exposed over a wide area of the Ozark Mountain region of north central and northwest Arkansas.<sup>4</sup> It is a high-grade raw material similar to that of the St. Peter formation of central-east Missouri and the Oil Creek sand of east-central Oklahoma, both of which are major sources of sand for the glass industry of Mid-America. The only active glass-sand operation in Arkansas is located on the Missouri Pacific Railroad at Guion in Izard County. This operation is producing 10,000 to 20,000 tons per month by adit from the upper 55 ft of a 160-ft section of the St. Peter. The material is washed and sized at the mine. It is consumed by Arkansas' glass industry as well as that of adjoining states.

There are other formations in north Arkansas such as the King's River sandstone and the Calico Rock sandstone or basal member of the Everton formation, as well as unconsolidated sands of the Gulf Coastal Plain, that may provide suitable raw material for the glass industry, or any other requiring a high-grade raw material. One of the principal factors in the development of these deposits, in addition to grade, is accessibility to rail transportation. The quantity of St. Peter sandstone, easily accessible to railroad facilities, is inexhaustible; therefore, St. Peter sandstone will probably continue to be developed to the exclusion of the other potential glass-sand deposits of the state.

**Novaculite:** Arkansas novaculite, of Devonian age, is a cryptocrystalline, hard, flinty, white to black, siliceous rock. The tenacity of the rock apparently is not due to the cementing of the fine, angular quartz grains but rather to the interlocking of the grains. Novaculite crops out throughout the Ouachita Mountain area of central Arkansas, ex-



tending from Malvern west to near the Oklahoma boundary, and reaches a maximum thickness of about 950 ft. It is generally composed of three lithologic units: the lowest is massive, white novaculite; the middle consists of interstratified layers of black shale and thin-bedded dark novaculite; while the upper section consists principally of a massive, gray to white novaculite which in part is calcareous. The formation was highly folded during the development of the Ouachita Mountains, and therefore usually occurs in every attitude except horizontal. It is composed almost entirely of silica, as indicated by the chemical analyses<sup>5</sup> of table V.

**Table V. Chemical Analyses of Novaculite**

	White Novaculite, Hot Springs	Fine Ouachita	Ten Mile Quarry	
			White Ouachita	Black Ouachita
SiO <sub>2</sub>	99.45	99.46	99.12	99.27
Al <sub>2</sub> O <sub>3</sub>		0.13	0.48	0.23
Fe <sub>2</sub> O <sub>3</sub>	0.26	0.06	0.02	0.03
CaO	0.12	0.04	0.12	0.08
MgO	Trace	0.08	0.06	0.08
K <sub>2</sub> O	0.19	0.16	0.14	0.10
Na <sub>2</sub> O	0.54	0.10	0.24	0.11
L.O.I.	0.06	0.14	0.22	0.60

Novaculite has been employed for a number of years as a raw material for the manufacture of whetstones and oilstones. For this purpose there are two classes of commercial material available in these deposits, the "Arkansas Stone" and "Ouachita Stone." Each class contains two types of stone, hard and soft. The Arkansas Stone is a very fine-grained, hard, compact rock which breaks with an uneven to conchoidal fracture, while the Ouachita Stone resembles unglazed porcelain and is relatively more porous. Pore space in the latter constitutes about 5.0 pct of the total bulk, as compared to 0.17 pct for the Arkansas Stone.

Novaculite is presently quarried at two locations in the Magnet Cove area as a raw material for the manufacture of silica brick and as a railroad ballast (Rock Island Railroad).

Loosely-consolidated Cretaceous conglomerates in Pike County contain thick accumulations of well-rounded novaculite pebbles. These have been found as suitable for use in pebble mills as the Danish pebbles currently imported at a price of 16 cents per pound.

**Tripoli:** Tripoli is the term usually applied to a cryptocrystalline white material composed almost entirely of silica. It is soft when mined, but cakes and case-hardens on exposure.

Arkansas has two types of tripoli, or at least tripoli of two different origins: (1) in Benton County of northwest Arkansas, tripoli reaches a thickness of 40 ft and is derived from chert; (2) in Hot Springs, Pike, Garland, and Montgomery<sup>6</sup> counties of central Arkansas, tripoli beds attain a thickness of 100 ft and are derived from novaculite.

The origin of tripoli is a matter of conjecture, but it is generally presumed to have been formed either by the action of ground water and normal weathering, or to have been deposited in its present form. Both these theories are inadequate in explaining Arkansas' deposits.

Tripoli, derived from chert, contains as high as 99.7 pct silica and as low as 0.018 pct iron oxide, while that derived from novaculite varies from 98.3 to 99.5 pct silica and from 0.19 to 0.40 pct iron oxide.

The open-pit mining of tripoli in the past has led to considerable contamination due to rain wash, etc.

Future open-pit operators should try to avoid this source of contamination or, wherever possible, underground mining methods should be employed.

At present tripoli is mined northeast of Hot Springs (fig. 4) and processed at a plant in the city of Hot Springs. Two other mining and milling operations are expected to commence production in the immediate future in an area a few miles east of Rogers in northwest Arkansas.

The type of beneficiation process depends upon the market in which the tripoli is to be utilized. High-grade raw material may be secured in many instances by selective mining, while in other cases chemical treatment for the reduction of iron oxide is imperative. Where low and medium-grade raw material is required, simple drying, grinding, and sizing may be employed.

High-grade tripoli may be used by industry wherever a white color, fine particle size, high degree of



**Fig. 4—Tripoli deposit, Malvern Mineral Co., northeast of Hot Springs, Ark.**

purity and relative chemical inertness are of prime consideration. At present tripoli is used as an abrasive in scouring soaps and powders, metal polishes and in various metal-buffing compounds. It is widely employed in oil-well drilling mud and has also been used as a filler in paints, plastics, and paper.

Arkansas tripoli deposits contain large reserves but they are not unlimited. The drilling of one deposit in Benton County has indicated an estimated reserve of between 1,300,000 and 1,800,000 short tons of tripoli.<sup>7</sup>

**Sandstone:** Under this heading all types of sandstone deposits are included, exclusive of sandstone employed by the glass-sand industry, which has been previously discussed.

Sandstone, of course, is a common constituent of the Paleozoic rocks in the northwest half of the state. It ranges in age from Ordovician to Pennsylvanian; in thickness, from beds of less than an inch thick to formations several thousand feet thick (Jackfork); in attitude, from vertical to horizontal; and in composition, from nearly 100 pct quartz grains to micaceous, calcareous, ferruginous, and argillaceous sandstones.

Partially consolidated to consolidated Cretaceous sandstones are exposed along the north and west edge of the Gulf Coastal Plain. They also vary in composition, often being marly. They are usually nearly horizontal in attitude.

Arkansas sandstones are employed entirely in building and road construction; however, the St. Peter sandstone of north Arkansas, in addition to its employment as a glass sand, has been used as a foundry sand.



**Chert:** Chert, more commonly known as flint, is a cryptocrystalline, hard siliceous rock, exhibiting a conchoidal fracture and usually is composed almost entirely of silica. It ranges in color from white (Boone) to black (Bigfork). It may also be reddish in color, due to iron staining, and black, due to manganese and iron. The two thickest and therefore most prominent cherts of Arkansas are: the gray to black Bigfork chert of Ordovician age, which is believed to reach a total maximum thickness of 700 ft, and occurs in the Ouachita Mountain area of central and west central Arkansas; and white cherty horizons in the Boone formation of Mississippian



Fig. 5—Limestone quarry, Batesville, Ark. St. Clair limestone block in foreground. Pink Fernvale limestone in quarry face.

age, which occasionally reach a continuous thickness of 50 to 60 ft in north Arkansas. Chert also occurs sporadically in many other Paleozoic formations.

The Boone chert currently is used as a road metal and could also be employed as a railroad ballast, while the Bigfork is used in the manufacture of roofing granules.

**Quartz Crystals:** Quartz crystals, of hydrothermal origin, have been mined for years from veins and cavities in sandstone formations of the highly folded Ouachita Mountain area of central and west central Arkansas,<sup>9</sup> especially in the area immediately north and west of Hot Springs. The crystals in the various deposits were investigated during World War II for their piezoelectric properties to determine their suitability for frequency control in radio receivers and transmitters. For the most part however the Arkansas quartz crystals are not adaptable for this purpose and therefore find their chief market as mineral collector's items, museum specimens, and also after cutting they are marketed by jewelry stores in Hot Springs under the trade name of "Hot Springs diamonds."

#### Limestones and Dolomites

**Limestone:** North and central Arkansas are underlain by Paleozoic formations, therefore, limestones are abundant, varying from Powell, Smithville, Everton, Joachim, Plattin, Kimmswick, and Fernvale of Ordovician age, through St. Clair of Silurian age to Boone and Pitkin of Mississippian age. The variety of limestone permits a wide variety of uses which are dependent on grade, color, texture, etc., ranging from the production of burned lime to road and building construction.

Noncherty portions of the Boone have been most widely used as the raw material for the manufacture of burned lime, although others, especially the

Plattin, Fernvale, Kimmswick, St. Clair and Pitkin, with a calcium carbonate content of over 98.00 pct and in many cases in excess of 99.00 pct, are as suitable for this purpose or any other requiring a high-grade limestone.

In a state that is predominantly agricultural, large quantities of agricultural lime are required and all of these formations contain a sufficiently high-lime content to permit their use for this purpose.

Plattin, Kimmswick, Fernvale, and St. Clair in the Batesville area (fig. 5) are quarried for use as interior and exterior marble.

Siliceous black limestones of the Fayetteville and Pitkin formations have been widely utilized as a decorative stone. The "black marble" of the Fayetteville formation is a siliceous (silica content varies from 10 to 25 pct) limestone which derives its black color from included carbonaceous matter. The individual limestone beds, which vary in thickness from 1 in. to over 3 ft, are interstratified with black carbonaceous shale. The Pitkin limestone also contains carbonaceous zones that are called "black marble." These are usually of higher calcium carbonate content than those in the Fayetteville with correspondingly lower silica content (from 3 to 12 pct silica). "Black marble," in addition to its present use as a polished decorative stone, could be used successfully in terrazzo.

A sizeable mass of coarsely crystalline contact-metamorphic calcite crops out in the Magnet Cove area. It has been quarried for use as an "aglime."

Upper Cretaceous chalks occur along the north edge of the Gulf Coastal Plain in southwest Arkansas. Chalk of the Annona formation is believed to offer the best opportunities for development, as it reaches a thickness of over 150 ft and a central 55-ft section (fig. 6) contains 90.0 pct calcium carbonate. The farm land in the Gulf Coastal Plain of south Arkansas, as well as adjoining states, is "sour," requiring the addition of appreciable quantities of "aglime." These deposits of Annona chalk, which are suitably located along rail lines, should be producing "aglime" for the whole area. Arkansas' only cement plant, which is located at Saratoga, utilizes a portion of the Annona chalk as a raw material.

Impure limestones or marls of Tertiary age crop out along the Mississippi Embayment in Pulaski and Saline counties, but their low calcium carbonate content (47 to 76 pct) more or less limits their potential use to the manufacture of rock wool.

**Dolomite:** Dolomite in Arkansas is restricted principally to outcrops of Jefferson City and Cotter formations along or adjacent to the Missouri boundary in northwest and north central Arkansas. These dolomites are often very high grade and at one location are produced as a raw material for the Oklahoma glass industry. In certain localities they could be employed as a source of burned magnesia and almost everywhere as a source of "aglime."

#### Barite

Since 1945 Arkansas has been the nation's principal barite producer. The output is employed almost entirely in the manufacture of drilling mud. Arkansas barite occurs as a replacement deposit at the base of the highly folded Stanley shale, immediately overlying the Arkansas novaculite. Barite apparently replaced carbonate-rich zones in the shale, leaving thin intercalated sandstone beds unaltered. It also has been found as a replacement deposit in



the upper portion of the novaculite formation, occasionally replacing the novaculite.

Two companies, National Lead Co. and Magnet Cove Barium Corp., are mining barite from a synclinal structure northeast of Magnet Cove. The baritic zone is 35 to 45 ft thick and mine dirt averages about 67 pct  $\text{BaSO}_4$ .<sup>9</sup> Both open-pit and underground mining methods are employed. The ore is beneficiated by flotation, producing a 92 to 95 pct product. The estimated reserve as of January 1949 in this deposit was 8,400,000 short tons.

Similar type barite deposits have been prospected in Montgomery, Polk and Pike counties.<sup>10</sup> The Montgomery County barite deposits, which are located some 40 miles west of Hot Springs, have been described by McElwaine.<sup>11</sup> The Gap Mountain deposit is composed of four lenticular barite bodies, 300 to 1200 ft in length, 20 to 30 ft thick, with a vertical to steep dip. The Fancy Hill deposit is composed of six lenticular deposits, 300 to 1800 ft long, 30 ft thick, with nearly vertical dip. The Sulphur Mountain deposit, which is located along the opposite limb of the syncline to the Fancy Hill, has not been as extensively prospected, but it is known to have a minimum thickness of 26 ft, and a minimum length of 250 ft and a dip of 50°. The U. S. Bureau of Mines has drilled and sampled these deposits.<sup>12</sup>

The estimated barite reserves for the entire state of Arkansas, as of January 1949, were 15,000,000 short tons of +40 pct ore.<sup>13</sup>

Structural and stratigraphic conditions exhibited by the known barite occurrences extend from Magnet Cove to Mena near the Oklahoma border; therefore the possibilities of finding new deposits in this 100-mile stretch appear favorable.

### Gypsum

Gypsum occurs in south Pike County in the DeQueen member of the Trinity formation (Lower Cretaceous). The gypsum is white to pinkish in color, fine-grained and massive. Veins of satin spar are abundant. The principal bed is relatively flat-lying and averages about 7 ft in thickness with a maximum thickness of 12 ft. It usually contains a 1-ft bed of black shale 3 ft from the base.

The gypsum was formerly mined by underground methods, but today open-pit mining methods are employed, requiring the removal of some 30 ft of overburden. Present production from the one active operation at Highland averages about four cars per day. It is consumed entirely by the cement industry. A much larger production could be secured if the present operators were permitted to extend their market.

### Celestite

Celestite or strontium sulphate was mined in Howard County during the last war and, like gypsum, occurs in the lower part of the DeQueen member of the Trinity formation. The celestite is found in pink, to gray, coarse-grained masses which are easily broken. It occurs in a flat-lying bed and varies in thickness from 1 in. to 1 ft. It appears to underlie the top soil of treeless portions of otherwise heavily forested areas, the soil being unable to support trees. The thin character of the known beds limits the use of the material (pyrotechnics) to a period of national emergency.

### Nepheline Syenite

Nepheline syenite crops out in the Little Rock, Benton, and Magnet Cove areas of central Arkansas.

The lateritic weathering of this rock produced bauxite in the two former areas. It is currently employed in the manufacture of roofing granules at the Little Rock plant of the Minnesota Mining and Manufacturing Co. It has also been used as a building stone and as an aggregate in construction work.

The Arkansas Geological Survey,<sup>14</sup> in conjunction with the Rolla Branch of the U. S. Bureau of Mines, has been conducting laboratory tests with nepheline syenite on the separation of the nepheline and feldspar from the iron-bearing minerals. The separation of these components was successful, but the cost of the process is too high to permit the present employment of the material as a flux by the ceramic industry. It is hoped, however, that through future



Fig. 6—Annona chalk at White Cliffs, Ark.

technologic developments, Arkansas nepheline syenite may some day become an important raw material in the ceramic industry.

### Greensand

Greensand or glauconitic-bearing sand and marl crop out in thick beds of upper Cretaceous age (Nacatoch Sand) in Hempstead, Nevada and Clark counties. East of the town of Washington, in Hempstead County, the greensand bed is more than 50 ft thick. The potash content, which varies from 1 to 4 pct, is too low to permit the production of this material by either the water softener or fertilizer industry. The deposits, however, require further prospecting and evaluation.

### Phosphate

A nodular phosphatic zone in the Cason shale, reaching a maximum thickness of 6 ft, extends across several counties of north Arkansas. Sampling reveals a phosphate content of about 30 pct  $\text{P}_2\text{O}_5$ . These deposits have not been able to compete on the open market with Tennessee and Florida phosphate; however, future prospecting may discover thicker and richer phosphatic beds in this shale. In addition, future tests on the application of ground rock phosphate directly to the soil may provide a large local market for this material. The knowledge that phosphatic shales occasionally possess a uranium content should lead to a thorough prospecting of these Cason shale exposures.

### Vermiculite

Vermiculite is derived from the weathering of ouachitite sills and dikes in Hot Springs and Garland counties of central Arkansas. The depth of the weathering is unknown and therefore the extent of the deposits is a matter of conjecture. The ouachitite, originally consisting of biotite and augite pheno-



crysts in a glassy to fine-grained ground mass, has been decomposed through weathering to a clay containing altered biotite crystals. The latter, which have been converted to vermiculite, reach a maximum width of 1 in. The sills provide the best prospect of future production of this insulating material, if they are of sufficient thickness and if weathering has been sufficiently extensive in area.

### Fuller's Earth

Fuller's earth or bleaching clay, which is used in the decolorizing of oils, occurs in several central and southern Arkansas counties. It has been reported in Saline, Drew, Pulaski, Searcy, Ouachita, Grant, and Dallas counties. It occurs as a transported clay, as for example the Tertiary deposits in the Jackson formation near Monticello, Drew County, and as a residual clay, formed by the weathering of basic igneous dikes, as at Olsen Switch in Saline County.<sup>15</sup> The latter has been the site of most of Arkansas' production, which was sporadic from 1901 to about 1940. Between 1904 and 1907 Arkansas ranked second nationally in the production of fuller's earth and third from 1909 to 1911, inclusive.

### Bentonite

Bentonitic clays, derived from the weathering of volcanic ash, have been reported in Saline, Hot Springs and Grant counties. True bentonite possesses the property of absorbing several times its volume of water. To date, this type of material has not been found in the state and, although the deposits are locally called bentonite, they actually belong to the class of clays known as metabentonites or nonswelling bentonites. While the former, due to its absorptive properties, is employed in drilling muds and the grouting of dams and drainage ditches, the latter is often employed as a lubricant and, because of its base exchange properties, is used in the water-softening industry. Two of the four deposits in Saline County and one in Hot Springs County produced a total of 3988 short tons between 1933 and 1938.

### Sand and Gravel

**Sand:** Sand, suitable for all types of construction, occurs in inexhaustible quantities throughout most of the state. In the area underlain by Paleozoic formations it is derived from the weathering of sandstones. In the remainder of the state, or in the Gulf Coastal and Mississippi Alluvial Plain, Tertiary and Recent sand deposits are abundant. In several localities, as at Fort Smith, Little Rock, Dardanelle, and Newport, it is secured from river channels. The principal consumer is the construction industry, although it has been reported that some production in the past has been utilized by the glass industry.

**Gravel:** Gravel, like sand in the northwest half of the state, is produced by the disintegration of Paleozoic rocks. In the Gulf Coastal Plain of south Arkansas and Mississippi Alluvial Plain of east Arkansas it occurs in Cretaceous conglomerates and in Quaternary terrace deposits. The Quaternary gravel deposits underlying the loess deposits of Crowley's Ridge, in the latter area, are probably the most uniform, and in the past have been the most widely used deposits in the state.

### Diamonds

A discussion of Arkansas' nonmetalliferous mineral resources would not be complete if the Arkansas

diamond deposits were omitted. These are the only known diamond deposits on the continent. They have been inactive for some time but at present are being reopened. In the past, the diamonds were mined from the residuum derived from the weathering of three volcanic plugs near the town of Murfreesboro, Pike County. Although 8 to 10 pct of the yield are reported to have been of gem quality, the deposits were never successfully exploited as a source of gem stones. It may be a matter of interest to note that the largest diamond found in these deposits, known as the "Uncle Sam," weighed 40.23 carats and was cut to a 14.3 carat gem stone. If the deposits are to be reopened, it appears that it will be necessary to concentrate on the production of industrial stones rather than those of gem variety.

### Conclusions

This concludes a brief résumé of Arkansas principal nonmetalliferous mineral resources. The industrial potential, lying dormant in these deposits, is believed to be extensive. The wide variety of deposits predicates their future employment by a wide variety of industries. The development and production of nonmetalliferous deposits are not wrapped in the legend and romance of metalliferous deposits and their production and beneficiation usually produce a relatively low-priced raw material; however, the often unlimited reserves and relative ease of processing provide a long life, relatively low initial investment and the employment of fewer highly trained technical personnel for anyone contemplating entering this field.

### References

- <sup>1</sup> G. W. Ward, W. B. Mather, J. E. Funnell and Noel H. Wood: *Natural Resources of the State of Arkansas*. Ark. Power and Light Co., Little Rock. (1949).
- <sup>2</sup> Paul G. Herold and George R. Heyl: *Kaolin Deposits of Southern Pike County, Arkansas*. Ark. Geol. Survey. *Bull.* 7 (1942).
- <sup>3</sup> J. E. Funnell: *Recent Studies of Wilcox Group Clays in Arkansas*. Accepted for publication in the *Bull. Amer. Cer. Soc.*
- <sup>4</sup> Albert W. Giles: *St. Peter and Older Ordovician Sandstones of Northern Arkansas*. Ark. Geol. Survey. *Bull.* 4 (1930).
- <sup>5</sup> Staff, Ark. Geol. Survey: *Mineral Resources of Montgomery, Garland, Saline and Pulaski Counties, Arkansas*. County Mineral Rept. No. 3 (1942) 67.
- <sup>6</sup> Robert B. McElwaine: *Two Tripoli Deposits in Pike and Montgomery Counties, Arkansas*. Private Publication, Little Rock. (Sept. 20, 1948).
- <sup>7</sup> J. Norman Payne: Univ. of Ark. Personal communication.
- <sup>8</sup> Hugh D. Miser: *Quartz Veins in the Ouachita Mountains of Arkansas and Oklahoma (Their Relations to Structure, Metamorphism, and Metalliferous Deposits)*. *Econ. Geol.* (1943) **38**, 91-118.
- <sup>9</sup> T. A. Jones: *Barite Deposits in the Ouachita Mountains, Montgomery, Polk and Pike Counties, Arkansas*. U.S. Bur. Mines. *R.I.* 4348 (1948) 1.
- <sup>10</sup> *Op cit.*
- <sup>11</sup> R. E. McElwaine: *Exploration of Barite Deposits in Montgomery County, Arkansas*. U.S. Bur. Mines. *R.I.* 3971 (1946).
- <sup>12</sup> *Op cit.*
- <sup>13</sup> H. B. Foxhall, State Geologist: Personal communication.
- <sup>14</sup> W. E. Crockett and H. B. Foxhall: *Preliminary Report of Occurrence and Properties of Nepheline Syenite in Arkansas*. *Cer. Soc. Bull.* (1948) **27** (2) 64-67.
- <sup>15</sup> *Op cit.*: 56-59.



# The Burt Filter

by W. G. Woolf and A. Y. Bethune

Filtration of hot (60°C) supersaturated zinc sulphate solution (sp gr 1.540) from slimy leach residues at the electrolytic zinc plant of Sullivan Mining Co., Kellogg, Idaho, is described. Separation of liquids and solids in the finished leach is by Burt filters modified to meet conditions of high-acid, high-density process for electrolytic zinc. Operating characteristics of the Burt filter on this pulp are presented.

THE hydrometallurgy of special high-grade zinc as practiced by the Sullivan Mining Co. at its electrolytic zinc plant, Kellogg, Idaho, involves an important filtration step immediately following the leaching process. By means of the filtration the heavy zinc sulphate solution is separated from the residual products which remain after the zinc calcine has been dissolved in the sulphuric acid electrolyte. Because this plant uses the so-called high-acid, high-density process<sup>1</sup> for the production of

<sup>1</sup> W. G. Woolf and E. R. Crutcher: Electrolytic Zinc Plant, Sullivan Mining Company. *Trans. AIME* (1936) 121.

superpurity zinc metal the above mentioned filtration must be accomplished in the face of certain difficult conditions inherent in the process.

W. G. WOOLF and A. Y. BETHUNE, Members AIME, are Superintendent and Assistant Superintendent, respectively, of Sullivan Mining Co., Electrolytic Zinc Plant, Kellogg, Idaho.

AIME San Francisco Meeting, February 1949.

TP 2770 B. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received Nov. 15, 1948; revision received Oct. 18, 1949.

First, the strength of the electrolyte (270g H<sub>2</sub>SO<sub>4</sub> per liter) results in a saturated zinc sulphate solution, having a specific gravity of 1.510 to 1.540, which must be kept warm during filtration because of its property of "seeding out" small crystals if allowed to drop much below 60°C. Second, the action of the "high" acid on zinc calcine under the temperature conditions of the leach (80° to 102°C), although favorable to good zinc extraction, causes a considerable quantity of iron to be dissolved (8 to 18 g per liter) along with variable quantities of alumina and silica, depending on the grade and type of original zinc concentrates roasted. These three, iron, alumina, and silica, are almost completely precipitated during the neutralization of the leach (only a few milligrams per liter of each remain in solution), so that the resulting pulp, instead of being a granular, sand-like product having a particle-size distribution dependent on the fineness of the zinc calcines leached, is in reality a slimy, chemical precipitate whose filtration characteristics constantly change depending on the amounts of iron silica, and other impurities, which are dissolved and reprecipitated. Third, the combination of supersaturated solution of high specific gravity plus a dense, semi-gelatinous residue creates a difficult washing problem requiring a positive displacement wash to liberate the zinc sulphate entrapped in the pulp.

In a closed-cycle hydrometallurgical operation, such as practiced in this plant, the extent of washing is determined by the volume limitations imposed on the intermediate wash waters by the amount of "fresh" (or process) water which may be added. The volume of fresh water used for make-up purposes is limited to the amount which is lost

during the closed cycle by evaporation in the leach, sulphate content of the calcines leached, moisture content of the residue, and spillage.

The Burt filter as modified and improved by the Sullivan Mining Co. has successfully met and overcome these difficulties under a variety of zinc plant operating conditions since 1928. It might have many interesting applications to metallurgical fields other than that of electrolytic zinc, and its possible usefulness to hydrometallurgists in general warrants its description and discussion.

The Burt filter is so named from its inventor who originated it in Mexico for pulp filtration in the cyanide process for gold and silver ores. While retaining the basic principle of Burt's earlier revolving pressure-type filter with internal filtration media, a number of modifications and improvements have been made in Sullivan Mining Co.'s installation.

The Burt filter may be classified as a batch-type pressure filter in contradistinction to either the conventional vacuum-type filter, which depends on atmospheric pressure to force solution through a cloth medium, or to the filter-press, which employs whatever pressure is imparted by the pump delivering the liquid being filtered. The Burt consists essentially of a hollow steel cylinder about 40 ft long, 5 ft in diameter, resting horizontally, and capable of rotation about its long axis. It is supported on one end by a hollow trunnion and near the other end by a riding-ring and roller combination. The cylinder is lined with filter units each fastened against the inside of the shell and parallel to the long axis so as to form a hollow cavity into which pulp may be charged. A specific amount of pulp is admitted to the filter and a unique valving arrangement prevents the loss of pulp while air pressure forces the solution through a canvas medium to the discharge port of each filter unit. The residue is left on the surface of the canvas inside the cavity. The remainder of the filter cycle is concerned with washing the residue free of zinc sulphate, discharging it from the Burt, and preparing the filter for the next charge.

A more detailed description of Burt filter construction, a typical filter cycle, and its operating characteristics when employed on material encountered in this plant will be given in that order.

**Description of the Filter:** Fig. 1 shows a side elevation view of a filter with riveted shell construction. Since this drawing was made shells have been fabricated by welding, instead of riveting, with complete success. Shells are lagged on the outside to retain heat. Fig. 1 shows a side elevation and plan view of a Burt filter in operating position. The ½-in. steel shells are lined with 3/16-in. copper sheet as protection against the corrosive action of the solution (containing about 500 mg Cu per liter) on iron, and the copper is given a thin protective coating of plastic-base paint. Fig. 2 is a view from the discharge end of the filter, with head removed, before filter units are fastened to the periphery. It shows



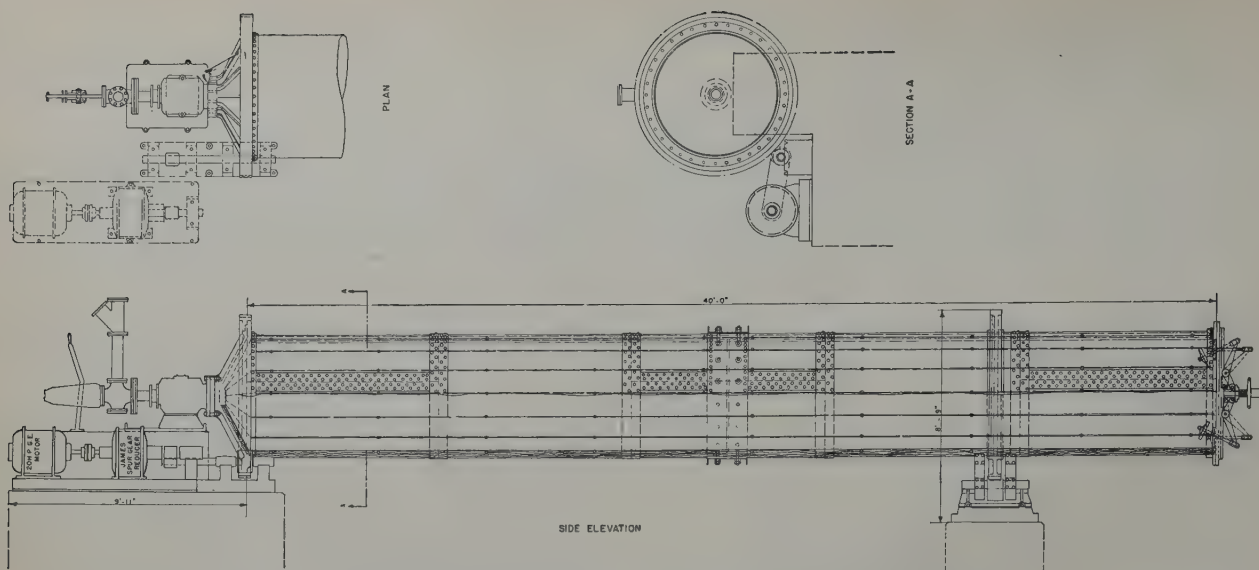


Fig. 1—Side elevation and plan view of filter. Riveted construction.

the copper lining in place and the interior of the filter.

The individual filter units, called "boards," are made of wood. They consist of a solid core overlain with two grooved surfaces, each 12 in. wide, which collect the filtered solution and channel it toward the discharge port located at one end. The assembled

board is 19 ft 7½ in. long, triangular in cross section, with a 33° internal angle between the grooved surfaces. The surface grooves are ¼ in. deep by ¼ in. wide. The entire board is covered with a specially manufactured canvas "sock" or bag of "palmatwill" weave, the open end of which is folded over, then sewn tightly with linen thread to prevent leakage of any solids through to the grooved surfaces.

Fig. 3 shows the individual filter boards before and after covering with canvas. Fig. 4 gives various views of a board and shows the manner of its attachment to the inside of a filter.

The third side of each board acts as the base of the filter unit and is fitted at midpoint with a bolt which protrudes through the shell and holds the unit firmly against the copper lining. A hollow solution discharge nipple at one end of the board secures the unit at the center of the filter, while specially designed wing-washers or lugs anchor it at the opposite end. Fig. 5 shows a filter with all units installed, and the wing-washers holding them in place.

Thirty-two units, or boards, constitute a full set for one Burt filter. Each unit has an effective filtration area, calculated from the dimensions of the two grooved surfaces on which the canvas rests, of 37.046 sq ft. Total filtration area is 1185.5 sq ft.

The feed inlet valve mechanism and the supporting trunnion at the head-end of the filter is shown in fig. 6. The essential features are the hollow charge pipe leading to the interior of the filter, the "flapper-valve" which seats on the end of the pipe, inside the filter, the hollow rod which activates the flapper, and the stuffing chamber. Through this hollow charge pipe all pulp and all subsequent washes are admitted to the filter. They are prevented from escaping under pressure by the flapper-valve which seats against the pipe when pulled shut by the hollow rod. Through the hollow rod, high-pressure air is applied to the filter. The stuffing chamber allows movement of the rod back and forth when necessary, and permits rotation of the filter without loss of air pressure. Various admittances to the filter, and even the application of vacuum, are controlled by individual valves on the separate lines. Fig. 7 gives a general view of the head end of a filter showing the separate pipelines and valves for the various washes, air vacuum, and steam.

The mechanism at the discharge end of the filter

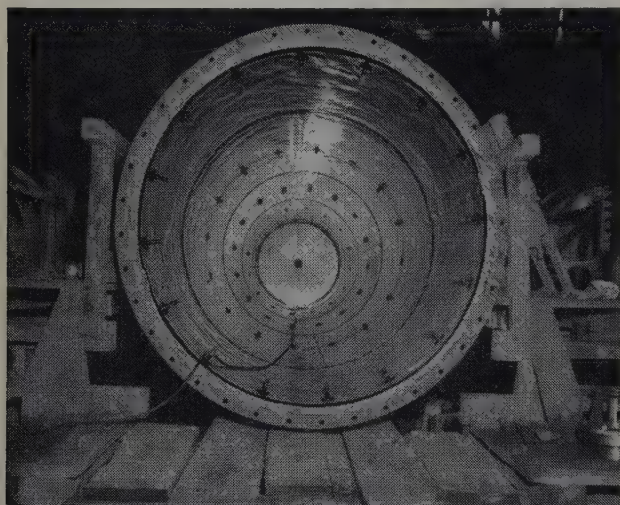


Fig. 2—Interior of filter from discharge end, before installation of filter units.

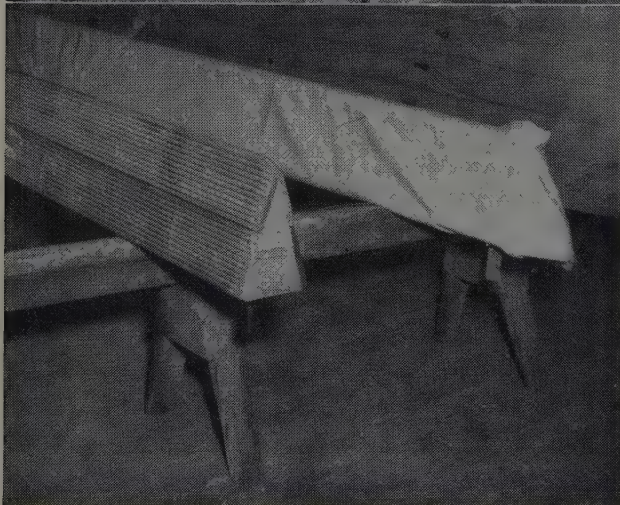


Fig. 3—Showing discharge nipple and individual filter boards before and after covering with canvas.



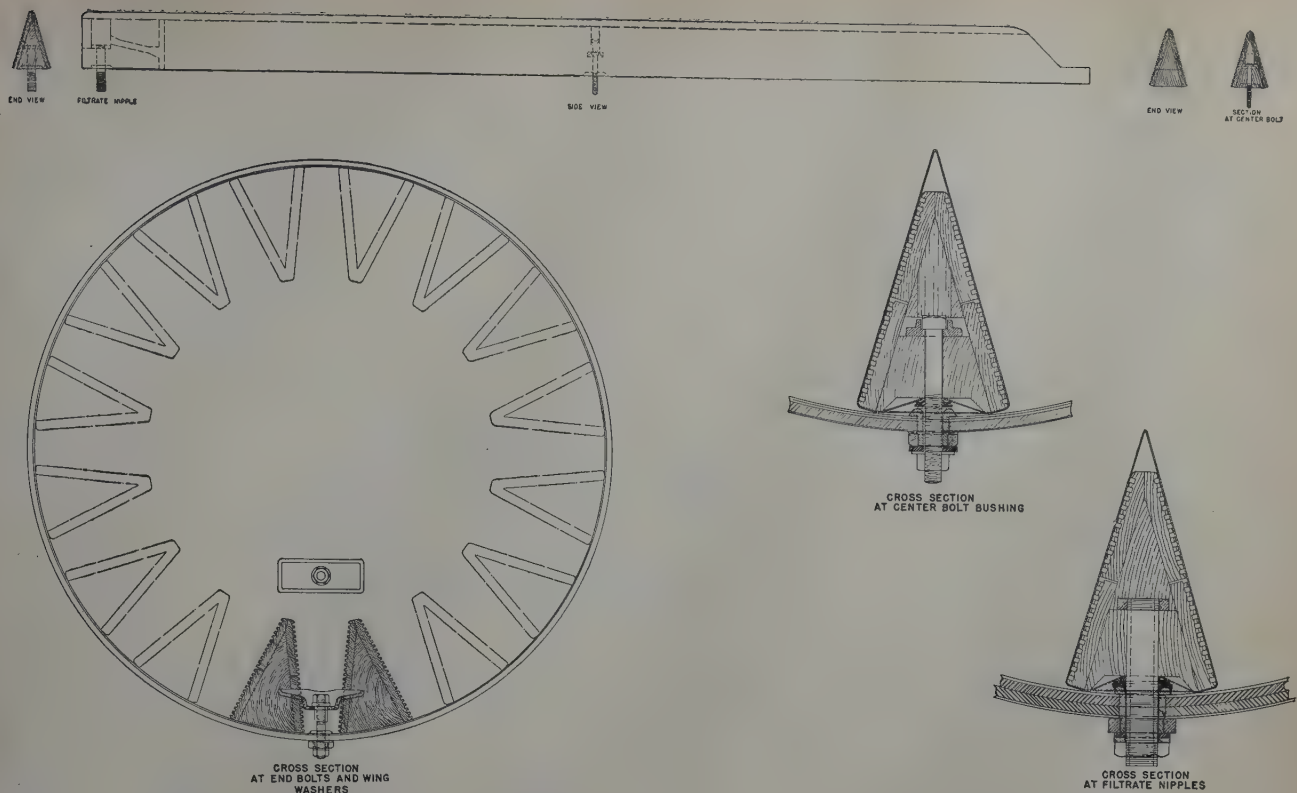


Fig. 4—Various views of filter board and manner of attachment to inside of shell.

is shown in fig. 8. The discharge ports are activated by a simple screw device, manually operated. The ports seat on the inside of the filter and are faced with live rubber. Turning the wheel clockwise closes the ports; counterclockwise opens them.

An ordinary hand valve placed near the outer edge of the discharge head acts as a relief valve to eliminate back pressure when the filter is loading, and as an indicator to show when it is fully charged.

**Filter Cycle:** Before describing a complete filter cycle it must be pointed out that the time consumed per cycle, and the results obtained, will depend almost entirely on the character of the pulp in question. Different mechanical setups may affect overall cycle times somewhat, especially in regard to charging and discharging, but, in the main, the ease or difficulty with which certain pulps filter and wash will determine the entire operation.

A complete cycle will involve the following: (1) charging (with leach pulp); (2) filtration (production of "Burt" solution); (3) first wash (with "weak" wash); (4) second wash (with "weak" wash); (5) third wash (with "mull" water)\*; (6)

\* "Mull" water is the term applied to the most dilute solution used on the filters. It consists essentially of process water (well water, condensate, any "fresh" water) plus small quantities of zinc sulphate leached from the zinc plant residue in the thickener operation. Any volume loss in the mull water circuit is made up by adding process water.

repulping (with mull water, plus vacuum); (7) discharging (production of zinc plant residue); (8) flushing (with mull water); and (9) blowing closed filter (with air).

A typical cycle is as follows:

(1) The filter, blown dry, is set for maximum loading by spotting the relief valve on the discharge head to upper dead-center position. The flapper valve is opened, and leach pulp is charged to the filter, by gravity flow from a storage tank, through a separate line suitably valved for that purpose. The operator opens the relief valve and waits until the cessation of escaping air and the appearance of

leach pulp indicates the filter is fully loaded. Time of loading is about 15 min. Theoretically the filter has a capacity of 17.29 vt\* (4144 gal or 553.9 cu ft).

\* The volume unit, (abbreviated vt), used in this paper refers to one volume ton, which is the volume occupied by 1 ton (2000 lb avdp) of water at 4°C. It is equal to 239.69 U.S. gal. The specific gravities are referred to water. The heavy solutions are usually tested at 60°C.

All valves are then closed and air pressure of 40 lb per sq in. is applied. The filter is caused to rotate. Rotation continues throughout the entire cycle, until the filter is again spotted for discharging.

(2) Filtration proceeds for about 45 min or 1 hr with Burt solution at a sp gr = 1.540 (60°C) being discharged at the center of the filter, collected in a sump located under the filter and pumped to storage tanks. When the major portion of the Burt solution has been filtered away from the residue, the cake, adhering to the filter cloth, "cracks." This allows the escape of warm, moisture-laden vapor, through the solution discharge nipples thereby creating a

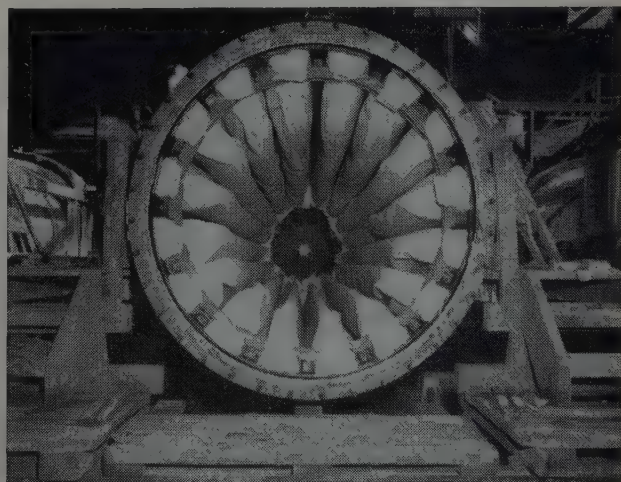


Fig. 5—View from discharge end showing filter units installed and held in place by wing-washers.



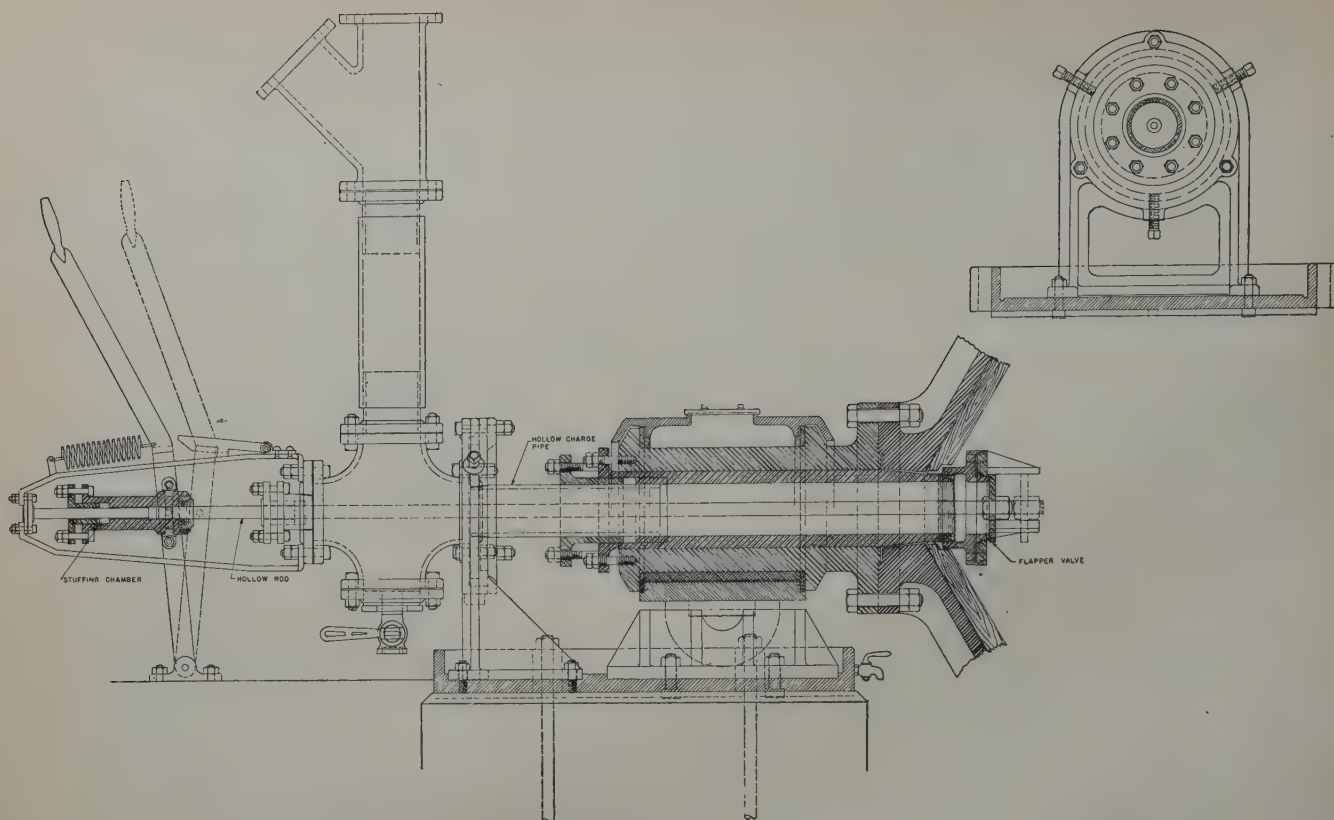


Fig. 6—Feed inlet valve mechanism and supporting trunnion at head end of filter.

miniature cloud at the center of the filter. The appearance of the cloud is so unmistakable that it is used to signify the end of filtration, and/or the end of any washing step. It is termed, locally, as a "blow."

(3) At the end of the first blow, the air is turned off and washing of the residue in the filter is begun. The recovery of the Burt solution still held in the residue is accomplished by applying successive washes each one of which is more dilute than the former. These washes, which begin as fresh water, are advanced from weaker to stronger washes as they gradually become enriched by repeated usage on the filters. Finally they become sufficiently concentrated to be disposed of in the leach.

The first wash so added is termed "weak wash." When the pressure within the filter drops below 5 lb per sq in., approximately 4 vt ( $\pm 950$  gal) having a specific gravity of about 1.170 are forced into the filter maintaining the pressure of 5 lb. Valves are closed, air pressure to 40 lb per sq in. is applied, and the wash forced through the cake, consuming about 30 to 40 min. This wash will displace sufficient Burt solution entrained in the cake to raise the average specific gravity of the effluent to  $+1.300$  ( $60^{\circ}\text{C}$ ). The filtrate is discharged to a sump and then to a tank reserved for "strong wash." This material is advanced directly to the leach where evaporation completes its concentration to  $\pm 1.540$  sp gr ( $60^{\circ}\text{C}$ ). Again the vapor cloud denotes that the wash has gone through; the second "blow" has been reached.

(4) The second wash added is "weak wash," similar in specific gravity to the first. Approximately 5 vt ( $\pm 1200$  gal) are used. This is recirculated upon itself (pumped back to the "weak wash" tank) and consumes 20 to 25 min. The object of this step is to wash the residue down as near to the specific gravity of the weak wash as possible. Usually this objective is attained within 20 or 30 "points" of gravity. For example, if the weak wash is applied at sp gr = 1.170, the last several gallons of filter discharge just prior to the third blow will be sp gr 1.180 to 1.200.

(5) "Mull water," having a specific gravity of about 1.020 to 1.040, and in sufficient quantities to make up for the volume of weak wash advanced, is added as the final wash. About 4 vt will usually wash the residue so completely that the final solution discharge at the 4th blow will not exceed 1.050 to 1.080. All mull water used in this step is advanced to weak wash. Time consumed is 15 to 20 min.

(6) After the final blow, air is allowed to remain on for a few moments to force all possible effluent to weak wash, then the air is turned off and pres-

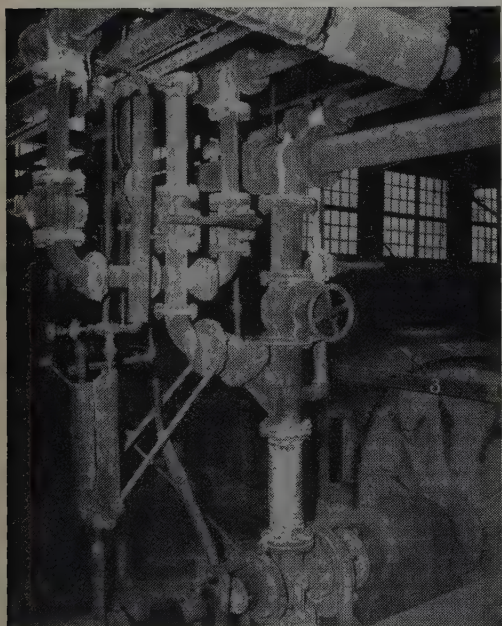


Fig. 7—General view of feed end of a filter, showing separation valves for rate pipelines various washes, air, vacuum and steam.



sure allowed to drop to zero. Five volume tons ( $\pm 1200$  gal) of mull water are added, the flapper valve left open, and vacuum is applied for 15 min. During this time the cake will be pulled from the canvas and mulled with water to form a thick slurry, which is usually free from all lumps and capable of smooth flow by the end of the vacuum period.

(7) Vacuum is stopped and the discharge ports are now opened to discharge the slurry to an open launder leading to conventional thickeners.

(8) Because the filter is level it will not completely discharge itself; it must be assisted by a flushing operation using about 5 vt of mull water. This final addition of mull water, applied with the discharge ports still open, sluices the residue from the interior of the filter and cleans the filter units. Rotation is stopped, one of the open ports spotted at lower dead-center, and the filter allowed to drain. The discharging, flushing, and draining takes about 40 min, including the 15-min vacuum period.

(9) When sufficiently drained, the filter is again closed, air is applied and rotation resumed. After 5 min the filter is blown dry and may be spotted at full-charge position ready to start the next cycle. If idle for any reason, the filter is kept warm by admitting low-pressure steam.

**Operating Characteristics:** Rotation of the Burt filter is necessary to its successful operation. By rotation the residue is not only evenly distributed over the filter units but the rotational movement keeps a quantity of pulp always moving over the surface of the cake which is thus continually sealed, whether it be a filtration or washing step, until all but a negligible amount of liquid has been discharged. This self-sealing action results in washing by displacement, which is more effective in removing entrained solution than washing by dilution.

Undue speed is not necessary. For zinc plant work, 4 rpm has been found satisfactory. Filters at this plant are equipped with controllers which permit set speeds of 2, 3, 4, and 6 rpm.

Filters are individually driven, each with a 20 hp motor, speed reducer, drive pinion, and master gear. Although full power load is maintained for short periods, consumption averages 15 hp per filter.

A small amount of cloudy filtrate will always appear when a filter is first charged and rotation begins. This is because some of the particles to be filtered are of lesser dimensions than the opening in the weave of the canvas bags. The cloudy condition persists for only a few minutes, however, due to the interstices of the cloth rapidly becoming partially blocked with larger particles, and from then on clear Burt solution is produced. Cloudy effluent does not appear in any subsequent washing steps, even when wash water is being applied to the filter.

Zinc plant data show that approximately 86 pct of the zinc sulphate available in the leach pulp charged to the filter is produced directly as Burt solution ready for further processing. The balance is trapped in the cake and must be recovered by washing and eventual recycling via the leach.

Metallurgical statistical figures for the year 1947 indicate that finished leach pulp, as delivered, contained 96.38 pct by volume of heavy zinc sulphate solution (sp gr = 1.540 at 60°C) and 3.62 pct by volume dry, solid residue (sp gr = 3.2). Per gallon of finished leach pulp the weight relationship, as delivered to the filters, was:

Burt solution	12.3861 lb per gal—	92.566 pct
Dry, solid residue	0.9948 lb per gal—	7.434 pct
Leach pulp	13.3809 lb per gal—	100.00 pct

Filters were invariably charged to capacity, so taking 4144 gal of pulp as norm, the total charge is 55,450 lb per cycle, consisting of 51,328 lb Burt solution (3836 gal or 16.0 vt) and 4122 lb dry residue.

In actual practice the total charge will be slightly larger than this due to the small amount of filtration which takes place while the filter is loading, and before the air is turned on. This additional volume will depend mainly on the filterability of the pulp.

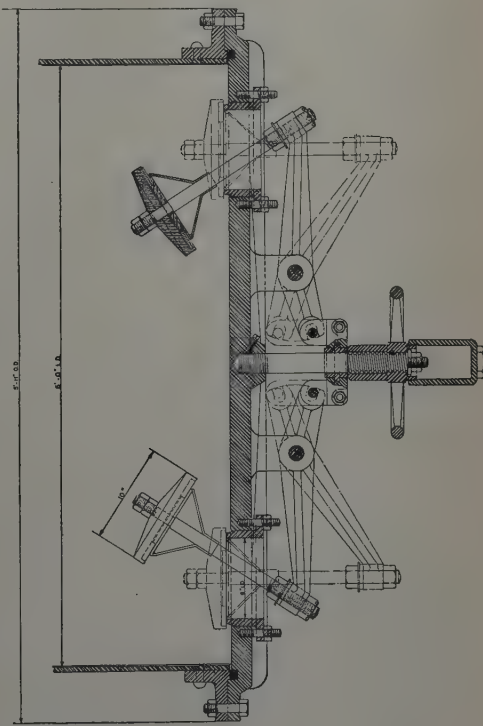
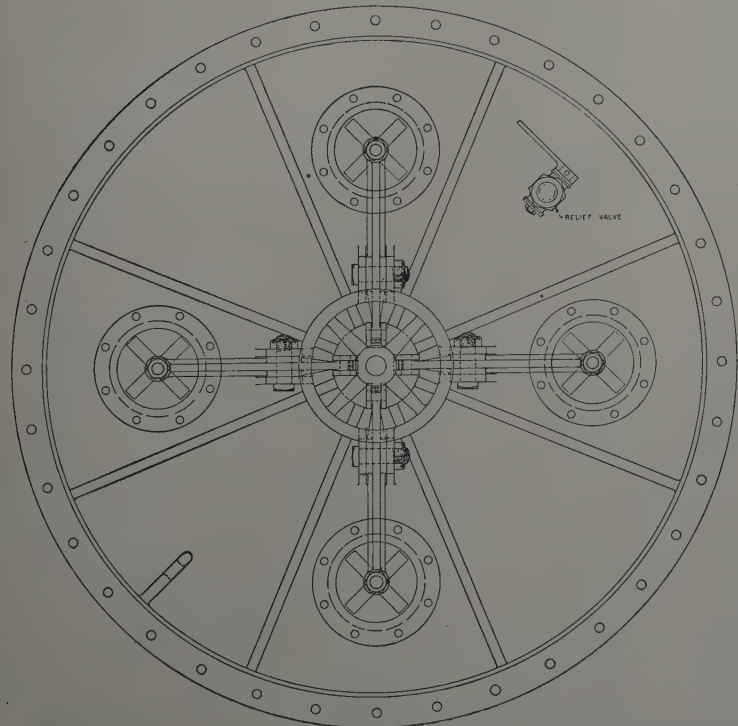


Fig. 8—Mechanism for discharging residue from filter.

It may be as low as 0.1 vt or as high as 1.0 vt. Some experimental work has been done in overcharging; that is, forcing the pulp into the filter under pressure with relief valve closed. This is entirely practicable as far as operation of the filter is concerned.

The wash-waters are kept separate in individual circuits which consist of sumps, storage tanks, pumps, and pipelines. Each circuit, however, whether it be strong wash (sp gr  $\pm 1.300$ ), weak-wash (sp gr  $\pm 1.150$ ) or mull-water (sp gr  $\pm 1.030$ ), is common to the entire installation of Burt filters.

Four parallel launders set at right angles to the long axis of the filters and directly under them, are provided for the filtrate as it spills from the discharge nipples into the collecting pan of each filter. A "swing-launders," anchored at one end but capable of being swung manually, by the filter operator, directs the filtrate from the collecting pan into whatever sump the solution should be placed. From the sumps it is pumped to storage tanks to be available for addition to leach or for delivery to whatever filter needs wash of that gravity. Live steam, at low pressure ( $\pm 10$  lb) is supplied through an open-end pipe under the solution surface in the storage tanks to keep the wash waters at  $50^{\circ}$  to  $60^{\circ}\text{C}$ .

The extent to which washing in Burt filters may be practiced, in any plant, will be governed by economic considerations. The principal factors affecting the amount of washing are: (1) Need for obtaining a clean residue. (2) Ease with which wash-waters may be converted into usable products, that is, the need for necessary volume control. (3) Time allowed, per total cycle, for the washing process.

In zinc plant metallurgy, where complete recovery of all entrained Burt solution is highly desirable, operations are directed toward obtaining the maximum wash consistent with a total cycle time not exceeding  $3\frac{1}{2}$  hr. As all wash waters are eventually disposed of in the leach, one method employed to wash the filters more thoroughly is to recirculate strong-wash whenever time permits.

Wash waters are delivered to the filters under a timing system. Pumps are first calibrated; then simple charts are established for the operators to follow. Signal timers indicating elapsed time, and equipped with warning lights, are installed at the head end of each filter to minimize errors. This simple system, followed successfully for a number of years, allows the department to keep the volumes of wash-waters continuously in balance by means of only three measurements of tanks per 24 hr, one set of measurements per 8-hr shift. Wash water effluents from the filters are directed toward their respective circuits at the beginning of every "blow" period.

Leach pulp volume		103.75 vt
Burt solution available in pulp (96.38 pct)		100.00 vt
<hr/>		
		Phase of Cycle
Burt solution produced		86.38 vt 1st blow—filtration
Advanced to strong wash	10.84 vt	2nd blow—weak wash to strong wash
Collected in weak wash	1.95 vt	3rd blow—weak wash recirculation
Advanced to weak wash	0.35 vt	4th blow—mull water to weak wash
Burt solution washed		13.14 vt
Total Burt solution recovered		99.52 vt
Lost in residue		0.48 vt vacuum filter discharge
		100.00 vt

Zinc-plant metallurgical data for 1947 indicate the relationships (see table in col. 1) with regard to washing and recoveries within the Burt filters when filtering the type of pulp described in this paper.

Although the Burt filter performs a metallurgical task in a satisfactory manner it does not discharge a finished cake comparable to that of either a filter-press or vacuum drum or leaf filter. The Burt filter discharge is in the form of a slurry. If this slurry is to be pumped or laundered to some distant point, then the Burt has an inherent advantage. However if a dried or semidried product is wanted then the slurry must be rehandled. Zinc-plant practice is to settle in conventional Dorr thickeners, filter on Oliver vacuum drum-type filters, and dry to  $\pm 20$  pct in rotary driers. The filtrate from the Oliver is returned to the mull-water circuit.

Filter-cloth life will vary from 25 to 60 days depending on the type of pulp being fed. Mechanical failure rarely causes more than 30 pct of filter-unit stoppage in the life-cycle of a set of bags. Actual ripping or tearing of the canvas allows mud to be discharged from the solution nipples, which is stopped by driving a wood plug into the nipple. Any filter unit thus plugged represents a decrease of 3.125 pct of the total filter area available, but this decrease is accepted for the life of the bags rather than to stop the filter, for replacement of the unit. Blinding of the bag by gypsum, a natural ingredient in zinc plant residue, is considered the major cause of short bag life. Other compounds may contribute, of course, but poor filtration capacity, due to blinding, fixes the practical life of a set of bags. When the total cycle time exceeds the permissible limit the filter is shut down and a freshly covered set of filter units installed. Core boards have a useful life of about 18 months, grooved surfaces about 12.

Experience has shown that one man can operate five Burt filters if the cycles are not shorter than three hr. The manipulation necessary to put through about fourteen cycles per 8-hr period constitutes a good shift's work. Over a period of one year filters will average six cycles per day, counting down-time for mechanical repairs and resacking of boards.

A crew of four men is ordinarily employed to renew a filter. Elapsed time for removing head, pulling old units, installing resacked units, closing filter, and checking for correct operation is 8 hr. In a continuous operation involving nine filters this same four-man crew resacks the filter units, assembles the filter boards from specially manufactured stock, and makes any necessary repairs thereto.

Sufficient air for nine Burt filters is supplied by a 500 cu ft per min compressor driven by a 75 hp motor at 300 rpm. One duplex vacuum pump, driven by a 30 hp motor at 300 rpm, having a displacement of 800 cu ft per min at 16 in. vacuum, services all filters and permits four filters to be on the vacuum line at once. Another similar vacuum pump is required by the Oliver filters. For zinc-plant work, pipelines are of copper, and pumps are of stainless steel. Wash-water tanks are unlined, of wood-stave construction. Sumps are of wood.

Burt filters could be operated with less individual attention if conditions warranted the installation of additional equipment. For example, wash-waters could be premeasured in separate tanks for automatic delivery to filters when required, and filtrates could be directed to proper circuits by automatic shifting of the swing-launders.



# Effect of $\text{BaCl}_2$ and Other Activators on Soap Flotation of Quartz

Chemical conditions for flotation and nonflotation of quartz with oleic acid as collector and barium, calcium, aluminum, iron, and tin as activators were studied using a simple vacuum-flotation technique in glass-stoppered graduates. The detailed study of barium activation led to an interpretation based on ideal Langmuirian chemisorption.

by

R. Schuhmann, Jr.,

and

Brahm Prakash

**F**LOTATION of quartz is of practical importance as something to be avoided in soap-floating many types of ores. Clean, unactivated quartz is not floated with fatty acids and soaps, such as oleic acid and sodium oleate, in the quantities normally used for flotation. However, data in the literature indicate that almost any multivalent cation will activate quartz if given an opportunity. Thus, a common problem is to prevent activation of quartz by the various inorganic cations inevitably present in flotation pulps.

---

R. SCHUHMAN, JR., Member AIME, is Associate Professor, Department of Metallurgy, Massachusetts Institute of Technology, Cambridge, Mass., and BRAHM PRAKASH, Student Associate AIME, was a Government of India Scholar, MIT.

This paper is based on a thesis submitted by B. Prakash in partial fulfillment of the requirements for the degree of Doctor of Science at MIT.

AIME New York Meeting, Feb. 1950.

TP 2826 B. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received July 26, 1949.

---

Wark and his coworkers<sup>1</sup> have demonstrated the reversibility of the chemical reactions and adsorptions involved in the activation, depression, and collection of the common sulphide minerals. The procedure in much of their work was to bring a mineral surface to equilibrium with solutions of known pH, collector concentration, and activator concentration, and then to test the floatability of the mineral by contact-angle measurement. From the data, graphs were constructed with pH and reagent concentrations as coordinates. These graphs show fields of flotation and fields of nonflotation, separated by narrow transition regions whose locations are shown by so-called contact curves. From the shapes and locations of the contact curves, which roughly separate fields of flotation from fields of nonflotation, a quantitative understanding of the interaction of the reagents with each other and with the minerals often can be deduced.

The study of quartz flotation to be described in this paper follows in broad lines the approach of Wark and coworkers. That is, pH, activator concentration, and collector concentration were varied to find equilibrium conditions of flotation and non-

flotation, and the results are presented graphically by means of contact curves. However, instead of testing for floatability by measuring the contact angle on a polished surface, a simple vacuum flotation technique was developed and used. Purified oleic acid was the collector and terpeneol the frother. Barium activation was studied in some detail, and exploratory studies were made of activation with calcium, aluminum, ferric iron, and stannic tin.

## Preparation of Materials

**Quartz:** Large lumps of high-grade vein quartz were crushed dry in a cone crusher and rolls. The -20, +28-mesh portion was screened out and used in the subsequent steps. This material was passed through a high-intensity magnetic separator to discard iron, then leached twice with hot concentrated HCl and washed repeatedly with distilled water. The cleaned sand was then wet ground with porcelain balls in a porcelain pebble mill, deslimed repeatedly by settling and decantation to discard -800-mesh material, and again washed with hot HCl followed by distilled water. The resulting stock of quartz was stored under water. Chemical analysis gave 99.8 pct  $\text{SiO}_2$ .

Table I gives the size analysis of the quartz used for flotation tests. Calculations from these data, using shape factors given by Gaudin and Hukki<sup>2</sup> indicate a specific surface of about 500  $\text{cm}^2$  per g.

Blank flotation tests in distilled water, and in water with added frother, showed the prepared quartz to be completely nonfloatable and thus indicated the absence of organic contamination.

**Oleic Acid:** The preparation of oleic acid was based on fractional vacuum distillation of methyl oleate<sup>3</sup>, followed by regeneration of oleic acid, and finally fractional crystallization of oleic acid from acetone solutions at low temperatures.<sup>4</sup> The pure oleic acid was stored in a refrigerator. The iodine number of the oleic acid was found to be 90.0 (theoretical 89.93).

Oleic acid was used in the form of a dilute water solution of sodium oleate, after preliminary flotation tests showed no effects of form of addition and order of addition of reagents when an adequate conditioning time (that is, 30 min) was provided.

**Other Reagents:** Sodium hydroxide solutions low in carbonate were prepared by first making 1:1

Table I. Size Analysis of Cleaned Quartz

Screen Size, Microns		Weight, Pct
—295	+208	10.4
—208	+147	21.3
—147	+104	18.5
—104	+74	28.0
—74	+52	12.2
—52	+37	8.5
—37	+18 <sup>a</sup>	1.1
	—18 <sup>a</sup>	0.0
Total		100.0

<sup>a</sup> Sedimentation size, calculated from Stokes' law.

aqueous solutions and centrifuging to separate the carbonate and other insoluble material. All metal salts used were of analytical reagent grade.

### Experimental Procedures

**Vacuum Flotation:** The flotation cell consists of a standard, glass-stoppered graduated cylinder of 100-ml capacity. A large number of these graduates are kept on hand so that a dozen or so tests can be in progress at the same time. For lecture demonstration purposes a 1-liter graduate has been used with substantially the same technique as that used with the 100-ml cylinders.

About 1.2 g ( $\pm 10$  pct) of quartz is scooped out of the stock (under water) and transferred to the cylinder with clean, air-saturated distilled water. Reagents are added and the volume brought to about 100 ml. The glass stopper is inserted and the pulp then agitated for 30 min by tumbling end over end at 50 rpm. For this purpose a pair of brackets holding up to 12 100-ml cylinders was built onto the slow-speed shaft of a 1/20 hp gearmotor.

After the 30-min conditioning period, the graduate is removed from the agitator and the glass stopper replaced by a rubber stopper with a glass tube and stopcock, connected by suction tubing to a trap and to a water-aspirator vacuum pump. After applying the vacuum for a short time, flotation or nonflotation of the mineral is readily observed. If the mineral is at all floatable, the bubbles nucleate and grow on the mineral bed in the bottom of the graduate and then break loose carrying up a load of mineral. This process continues until all floatable mineral is lifted from the bottom. If the mineral is nonfloatable, a longer period of application of the vacuum is required before bubbles even start to form, and the bubbles will not carry a mineral load to the surface. Qualitatively, a considerable gradation in floatability can be observed as chemical conditions are changed gradually from conditions of nonflotation to conditions of complete flotation.

Use of air-saturated water to make up the pulps is preferred practice but is not essential since with sufficient vacuum the boiling of the water itself furnishes the bubbles for flotation. Care must be taken to use clean air in saturating the water. The best procedure is to draw laboratory air through a cleaning train and then through a closed gas-washing bottle containing the water to be saturated. A water aspirator is convenient for this purpose. The cleaning train consists of (1) an ascarite absorber, (2) concentrated NaOH wash bottle, (3) water wash bottle, and (4) trap. Compressed air from the laboratory supply line passed through this train retained enough organic contamination to cause mineral flotation without a collector.

**Comparison of Vacuum Flotation and Captive-bubble Techniques:** Previous studies of relationships between floatability and reagent concentrations have been based largely on the use of the captive-bubble technique of measuring contact angles. The main advantages of the vacuum-flotation technique used in this work over the captive-bubble technique are: (1) the apparatus is simpler, cheaper, and available in any laboratory; (2) less skill and special techniques are required; (3) the test is itself a flotation test, so that no assumption is made of a relationship between floatability and the contact angle on a prepared, plane surface; (4) the sensitivity is greater, as contact angles well under  $10^\circ$  are sufficient for flotation but are not detected by the captive-bubble method; (5) the mineral surface area is greater than that tested in a polished surface, thus giving reduced contamination hazard and reduced sampling error; and (6) the method permits rapid exploration if a large number of graduates and a suitable mechanical agitator are provided.

**Determination of Critical pH:** The principal variables in this work were pH, activator concentration, and collector concentration. The usual procedure was to make up a pulp (as already described) in the 100-ml graduate with given quantities of activator and collector and with 2-mg terpeneol per liter to act as frother. With this pulp a series of flotation tests was carried out at varying pH. Sodium hydroxide (carbonate-free) and hydrochloric acid were used for pH control, and a full conditioning period was provided after each pH change to insure equilibrium. The pH was measured with a Beckman Model H, line-operated pH meter. The Beckman special high-pH electrode was used at high pH values.

When barium is used as activator, no flotation occurs with a given activator-collector combination below a certain critical pH. As the pH is raised above this critical value, quartz floatability increases rapidly so that complete and rapid flotation is obtained at pH values from 0.5 to 1.5 units above the lowest pH at which slight flotation can be detected. The quartz remains floatable as the pH is increased further. With activators other than barium, the variation of quartz floatability with pH may be more complex. Not only is there a "lower critical pH" below which no flotation occurs, but also there may be an "upper critical pH" above which there is no flotation. A few cases of more complex behavior have been observed, which involve two pH ranges in which flotation occurs.

In the vicinity of the critical pH, the change from incipient flotation to complete and rapid flotation takes place over a pH range of about one pH unit, and if the contact curve is oblique to the pH axis the critical range may extend several units. Also, it is to be expected that a change in particle size would shift the critical range somewhat, since to float as well as fine particles, coarse particles must be more air-avid. Thus, the critical pH is not a point of abrupt change from flotation to nonflotation, and this feature must be kept in mind in interpreting and using the data.

Critical pH values were checked by approach from both sides. Except for some inconclusive flotation tests with iron as activator, it was found that the value of the critical pH was the same whether approached from the high or low-pH side. Also order of addition of reagents had no effect on the



results. Thus, the chemical reactions and adsorptions involved are reversible, and the flotation tests were carried out under conditions close to chemical equilibrium.

### Barium Activation of Quartz

**Experimental:** Critical pH determinations were made over the following ranges of reagent additions:

	Mg per Liter	Millieq. per Liter
Oleic acid	5-500	0.018-1.8
BaCl <sub>2</sub> · 2H <sub>2</sub> O	3-300	0.025-2.5

These concentrations cover the usual range of practical reagent concentrations. Thus 5 mg per liter = 5 ppm = 0.01 lb per ton of water, which is equivalent to 0.04 lb per ton of ore in a pulp 20 pct solids. For the very dilute pulps (about 1 pct solids) used in this work, the quantities of reagents abstracted by the mineral can account only for a very small fraction of the total reagent addition, since a reagent addition of only 1 or 2 mg per liter of pulp would be sufficient to form a monomolecular film on the quartz.

**Fig. 1—Effects of BaCl<sub>2</sub> and oleic acid on critical pH for quartz flotation.**

For given activator addition, flotation occurs under conditions below and to right of curve for that quantity of activator.

Curve No.	BaCl <sub>2</sub> ·2H <sub>2</sub> O Mg per Liter
1	3
2	30
3	50
4	90
5	160
6	300

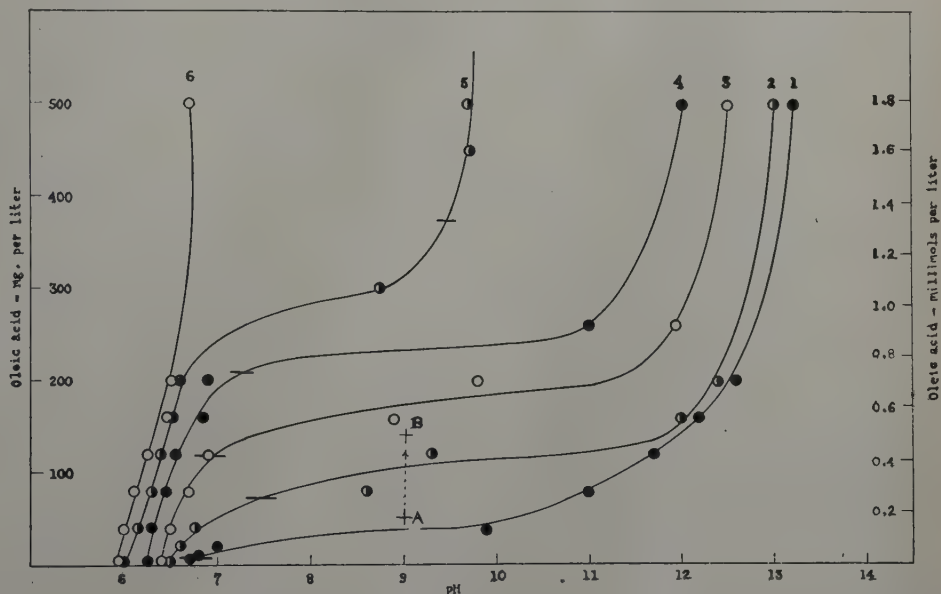
Figs. 1 and 2 give the experimental results plotted in two different ways. In fig. 1 the quantity of oleic acid is the ordinate and pH the abscissa, and each curve then shows the variation of critical pH with oleic acid addition for a fixed quantity of activator. In fig. 2, the ordinate is quantity of barium chloride so that each curve shows the variation of critical pH with barium chloride addition for a fixed quantity of collector. When the conditions represented by any one curve in either figure are considered, flotation occurs at conditions in the area to the right of the curve, and no flotation occurs under conditions to the left. The transition region from slight flotation to good flotation may be regarded as a band about 1 pH unit wide just to the right of the contact curve.

The contact curves in fig. 1 form a regular family of curves. With no activator added, no flotation occurs at any pH so no curve is given for 0 addition of BaCl<sub>2</sub>·2H<sub>2</sub>O. Each of the curves can be divided into three parts, starting from the bottom left: (I) oleic acid addition below stoichiometric requirements to form BaOle<sub>2</sub>, (II) transition region with curve substantially horizontal, and (III) oleic acid addition above stoichiometric requirements to form BaOle<sub>2</sub>. The condition of stoichiometric equivalence of barium and oleate additions to form the normal oleate is indicated on each curve by a heavy dash. For curves 1 to 4 the dash falls on the hori-

zontal parts of the curves within experimental error, for curve 5 the dash is a little above the horizontal part of the curve, and for curve 6 the oleic acid additions were not extended to the equivalence point.

When barium is in excess of that required to form barium oleate (part I of each curve), the critical pH falls in the range 6 to 7. No flotation occurs below the critical pH. Flotation occurs at all pH values above critical, including the pH of a normal sodium hydroxide solution. For the stoichiometric condition of excess barium, critical pH is relatively insensitive to reagent quantities.

When oleate is definitely in excess of that required to form barium oleate (part III of each curve), the critical pH is well into the alkaline range so that flotation is possible only at a high pH. The critical pH is lowered and the pH range of flotation becomes greater as the quantity of activator is increased. For example, with 3 mg BaCl<sub>2</sub> ·



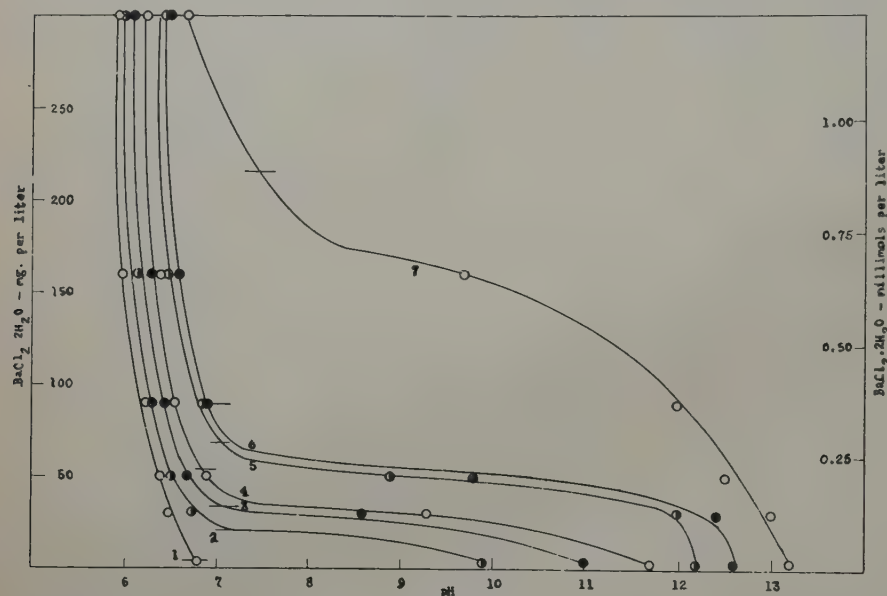
2H<sub>2</sub>O per liter and an excess of oleic acid the quartz floats only in strongly alkaline pulps above pH 13, but with 160 mg BaCl<sub>2</sub>·2H<sub>2</sub>O per liter and an excess of oleic acid the quartz floats at pH values as low as 11.

These results are in qualitative agreement with the experimental data of Gaudin and Rizo-Patron,<sup>5</sup> who found at a given pH (namely, 10.6) that most effective flotation was obtained with 1:1 mol ratio of oleate to barium and that flotation recovery fell off to approach zero as the 2:1 stoichiometric ratio was approached. However, their hypothesis, that these limiting ratios are the result of the fact that a barium ion adsorbed on quartz can hold only one oleate, is insufficient to account for our data since in our experiments only a small fraction of the total barium and oleate added can be accounted for on the quartz surface.

Action of oleic acid as a depressing agent can be seen by considering a given process in terms of the data in fig. 1. If a pulp is made up to pH 9 and contains 30 mg BaCl<sub>2</sub>·2H<sub>2</sub>O per liter and 50 mg oleic acid per liter, the pH and collector concentration are represented by point A. Point A is below and to the right of curve 2 for 30 mg BaCl<sub>2</sub>·2H<sub>2</sub>O per liter so that the quartz is floatable. If now we are unsatisfied with the way quartz floats under these

conditions and try to improve its flotation by adding more collector, we will move up toward B. We find that the more the collector, the worse the flotation, and in fact after crossing curve 2 at a collector concentration of about 100 mg per liter the quartz is depressed altogether by adding oleic acid while keeping pH and activator additions constant. After reaching this stage we can make the quartz float again either by adding more activator or by raising the pH to recross the curve.

Usually our interest is in depressing quartz after it has been activated unintentionally. The data in fig. 1 indicate several possible methods of depressing barium-activated quartz, the choice depending on stoichiometric relations in the pulp, as follows:



Initial Condition in Pulp  
Stoichiometric excess of  
Ba<sup>++</sup> over oleate

Procedure to Depress Quartz  
Drop pH to below 6

or

Add excess oleate

Stoichiometric excess of  
oleate over Ba<sup>++</sup>

Lower pH until quartz  
flotation stops; the  
greater the Ba concentration the lower the  
critical pH.

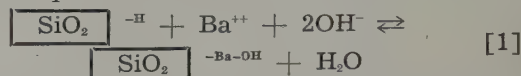
If these measures cannot be applied, other reagents may be necessary to deactivate the quartz by complexing or precipitating the barium.

Fig. 2 shows the same data as fig. 1 and is subject to the same explanation. Fig. 2 also shows clearly that increasing the barium chloride always lowers the critical pH and thus always increases the range in which quartz floats. Also, it can be seen that with any substantial excess of Ba over stoichiometric requirements for BaOl<sub>2</sub> (namely, above the flat portions of the curves), the critical pH is in the range 6 to 7 even for a 100-fold variation in oleic acid. (Curve 6 in fig. 1 also shows this.)

**Activation, Collection, and Precipitation Reactions:** Our data are consistent with the view that the quartz flotation is activator-controlled and not collector-controlled, provided only that there is enough oleate in the system to film the quartz when it does become activated. Two phenomena which would be expected in a collector-controlled system are absent: (a) There is no upper critical pH, corresponding to displacement of Ol<sup>-</sup> by OH<sup>-</sup> and depression as the pH is raised; (b) Even with a large

excess of barium, several times more than required to precipitate the oleate as insoluble soap, no difficulty was experienced in collecting quartz, in fact, flotation was best under these circumstances.

The following equation may be written to represent the activation reaction at one locus of ruptured bond on the quartz surface:



This equation is electrically balanced and represents the observed stoichiometry of base exchange in general, though perhaps not the true reaction mechanism. Stoichiometrically no distinction can be made between an increase of 2H<sup>+</sup> in the solution per Ba<sup>++</sup> adsorbed and the decrease of 2OH<sup>-</sup> per Ba<sup>++</sup> adsorbed by Eq 1. The abstraction tests of Gaudin and Rizo-Patron<sup>5</sup> indi-

**Fig. 2—Effects of BaCl<sub>2</sub> and oleic acid on critical pH for quartz flotation.**

For given collector addition, flotation occurs under conditions above and to right of curve for that quantity of collector.

Curve No.	Oleic Acid, Mg per Liter
1	5
2	40
3	80
4	120
5	160
6	200
7	500

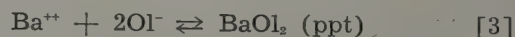
cated the reversibility of this reaction, as they found no abstraction of Ba<sup>++</sup> by quartz in acid pulps and high abstraction in alkaline pulps.

After activation by Eq 1, collection can occur as follows:



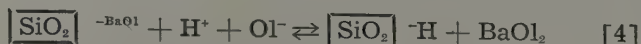
Our data indicate that this reaction proceeds strongly to the right, as the quartz was not depressed by increasing OH<sup>-</sup> and decreasing Ol<sup>-</sup> within the range of conditions studied. That is, the pH was increased to 13.4 and the Ol<sup>-</sup> reduced to a very small concentration by adding an excess of Ba<sup>++</sup> and under these extreme conditions the quartz still floated.

The formation of a barium-soap precipitate (independent of the presence or absence of quartz) is represented by:



This reaction of course also proceeds strongly to the right, as the barium soap is known to be very insoluble.

The depressing effects of lowering pH and increasing Ol<sup>-</sup> are accounted for by the following reaction:

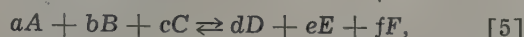


This equation shows floatable quartz, filmed with oriented, partial monomolecular BaOl<sup>+</sup>, returned to its original nonfloatable condition by replacing the BaOl<sup>+</sup> with H<sup>+</sup> and forming a separate BaOl<sub>2</sub> precipitate which does not help the quartz to float. Since both H<sup>+</sup> and Ol<sup>-</sup> are reactants on the left-hand side of the equation, it is clear that decreasing pH



and increasing  $Ol$  favor the formation of an insoluble barium oleate in preference to a  $BaOl$  film on the quartz, in agreement with our experimental results. A semiquantitative explanation of the experimental results, based largely on Eq 4, is developed in the subsequent sections.

**Adsorption Equilibria and the Equilibrium Constant:** For any chemical change at constant temperature, represented by



equilibrium conditions can be expressed in terms of an equilibrium constant  $k$ , defined as follows:

$$k = \frac{(a_D)^d (a_E)^e (a_F)^f}{(a_A)^a (a_B)^b (a_C)^c} \quad [6]$$

in which  $a_A$ ,  $a_B$ , etc., represent the activities of the respective reactants relative to some arbitrarily fixed standard state for each reactant. This is a general relationship based on the second law of thermodynamics which applies whether the reactants are pure solids, liquids, or gases; or in liquid or solid solutions; or in surface films, physically or chemically adsorbed; or in colloidal dispersions. However, the equilibrium constant defined as above has not been used generally for adsorption equilibria for lack of activity measurements on substances in adsorbed films and because simple relations between activity and composition in adsorbed films (corresponding, for example, to Raoult's law for ideal solutions, Henry's law for dilute solutions, and the simple activity-partial pressure relation for perfect gases) have not been established.

Thus, to give the equilibrium constant quantitative significance for reactions in which one or more of the reactants is in the system as an adsorbed film, we must develop some measure of activity for substances in adsorbed films. Moreover, judging from the mass of experience with solutions, it may be helpful to define an *ideal* system for which activity is a simple function of the composition of the surface in which adsorption occurs. The deviations of actual systems from ideal behavior can be evaluated in terms of activity coefficients. If the ideal system is chosen properly, it may be possible to take the activity coefficient as unity for many practical systems.

**Langmuirian Adsorption:** Langmuir's ideal case of chemical adsorption on solid surfaces may be described as follows:<sup>8</sup> Adsorption occurs on specific atomic sites, which may be called "elementary spaces." These elementary spaces are all alike and equivalent. At any given instant, some of the spaces are occupied by units of the adsorbate (namely, atoms, molecules, ions) while some are empty. Kinetically, at a given instant, the adsorbed units may be "evaporating" or leaving some occupied sites and other units may be "condensing" on empty sites. At equilibrium the rate of evaporation from occupied sites must equal the rate of condensation on empty sites. For the most ideal system, each elementary space behaves independently of the others so that the force of attachment and the probable time of residence of an adsorbate unit on a given elementary space are not affected by the presence or absence of adsorbate on adjacent elementary spaces.

For the ideal case of Langmuirian adsorption,

simple kinetic considerations\* lead to the following

\* Rate of evaporation is proportional to number of occupied sites  $N_A$ . Rate of condensation is proportional to concentration  $c_A$  and to number of empty sites  $N_e$ . Equating these rates as an equilibrium condition leads to Eq 7.

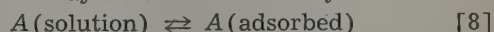
equilibrium relation between mols adsorbed per unit area  $N_A$ , mols empty spaces per unit area  $N_e$ , and concentration in solution  $c_A$ :

$$\frac{N_A}{N_e} = k_A c_A \quad [7]$$

† If there is only one species of adsorbate, this equation takes the more familiar form  $\frac{\theta}{(1-\theta)c} = k_A$  in which  $\theta$  is the fraction of elementary spaces occupied by adsorbate at equilibrium.

in which, at a given temperature,  $k_A$  is an adsorption constant characteristic of the adsorbate-adsorbent combination under consideration. Since Langmuir first derived a relation of this type and demonstrated its validity experimentally for several cases of adsorption of gases on solids, subsequent experimenters have found it valid for other adsorptions both from gases and from solutions. In general, it has proved most useful for adsorptions sometimes called chemisorptions in which the forces between adsorbate and adsorbent are specific and of the same order of magnitude as chemical bonds within a crystalline solid.

The adsorption of a given molecular species  $A$  from solution may be written formally:



For this chemical change, the equilibrium constant is

$$k = \frac{a_A(\text{adsorbed})}{a_A(\text{solution})} \quad [9]$$

The equilibrium constant becomes the same as  $k_A$  in Eq 7 if the customary assumption  $a_A = c_A$  for dilute or ideal solutions is made and, if further, it is assumed that the activity of adsorbate  $a_A$  (adsorbed) is equal to the ratio of mols adsorbed  $N_A$  to mols empty spaces  $N_e$ .

In processes of ion adsorption, the solid surface can be regarded as having two kinds of elementary spaces, positive sites in which negative ions adsorb and negative sites in which positive ions adsorb. In an ideal case these may be considered independently, so that the activity of an adsorbed ion can be expressed in terms of the ratio of the number of the adsorbed ions to the number of empty spaces of the appropriate sign. If one multivalent ion occupies  $n$  adjacent sites, kinetic reasoning leads to the ratio  $\frac{N_A}{(N_e)^n}$  as the appropriate activity function.

On the basis of the considerations in the preceding paragraphs, therefore, we propose that the activities of chemisorbed substances be compared with activities for the ideal case of Langmuirian adsorption, for which the activity of the adsorbed substance  $a_{\text{ideal}}$  is proportional to the ratio  $\frac{N}{N_e}$  of the number of adsorbed ions to the number of empty sites. If the adsorbed substance is an ion of charge  $n$  and occupies  $n$  single-charged sites on the surface,  $a$  should be taken as proportional to  $\frac{N}{(N_e)^n}$ .

Deviations of a real system from ideality can be measured by means of an activity coefficient ( $\gamma$ ), so that  $a_{\text{non-ideal}}$  may be considered as proportional

$$\text{to } \gamma \frac{N}{N_e} \text{ (or } \gamma \frac{N}{(N_e)^n} \text{)}.$$

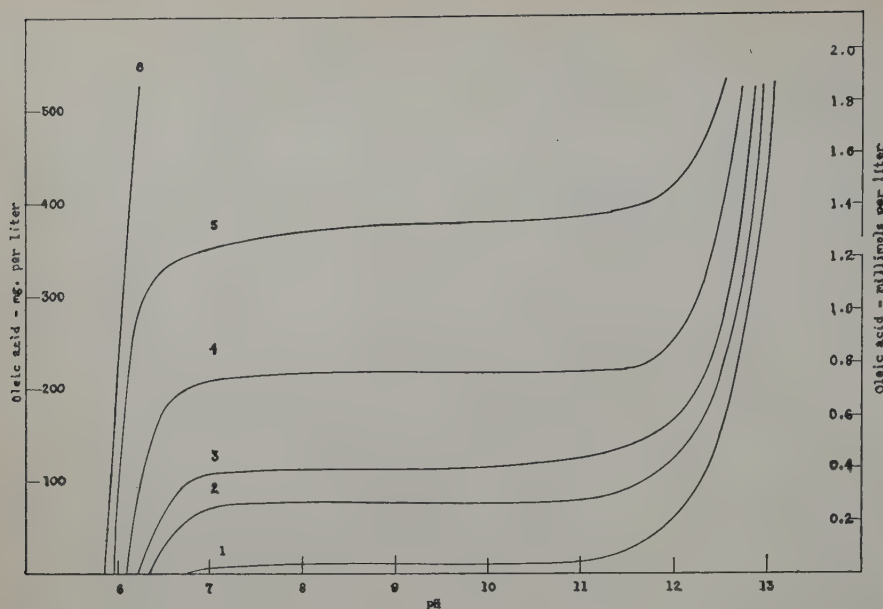


Fig. 3—Theoretical contact curves for quartz flotation with  $\text{BaCl}_2$  and oleic acid.

Compare with experimental curves in fig. 1.

Curve No.	$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$ Mg per Liter
1	3
2	30
3	50
4	90
5	160
6	300

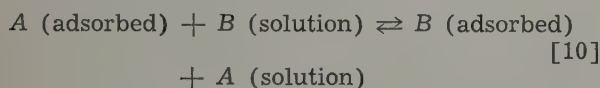
$$k_3 = \frac{1}{(\text{Ba}^{++}) (\text{Ol}^-)^2}$$

$$k_4 = \frac{(N_H)}{(N_{\text{BaOl}}) (\text{H}^+) (\text{Ol}^-)}$$

The equilibrium constant  $k_1$  deals with the adsorption of  $\text{Ba}^{++}$  on quartz in the absence of oleate, for which we have no data. Since, as already discussed, reaction 2 goes strongly to the right under all conditions tested we can say only that  $k_2$  is very

**Equilibrium Constants for Reactions Involving Adsorbed Reactants:** Earlier it was pointed out that the equilibrium conditions for a reaction involving a surface could be expressed in terms of a mass-action constant if a suitable measure of the activity of substances in the surface was available. Accordingly, the activity concept for adsorbed substances developed in the preceding section affords a basis for applying the mass-action law to reactions involving one or more adsorbed reactants.

A simple exchange adsorption may be represented by:



If we assume that the adsorptions of  $A$  and  $B$  are ideal Langmuirian adsorptions and that the solutions follow Henry's law of dilute solutions, the mass-action constant for this reaction becomes simply:

$$K = \frac{C_A N_B}{C_B N_A} \quad [11]$$

in which  $C_A$ ,  $C_B$  are the concentrations in solution and  $N_A$ ,  $N_B$  are the surface concentrations (namely, mols adsorbate per unit area). The number of empty spaces  $N_s$  cancels out and does not appear in the equilibrium constant.

Data for ion exchange on various commercial adsorbents, such as the ion-exchange media used for water softening, are well correlated by mass-action constants of the form of Eq 11. In fact, workers in this field have also derived such mass-action expressions on a theoretical basis similar to that of the foregoing discussion.<sup>7</sup>

**Application of Mass Action to Reactions in Quartz Flotation:** For reactions 1 to 4 in the quartz-flotation system, the mass-action constants can be written as follows, assuming ideal Langmuirian behavior of adsorbates:

$$k_1 = \frac{(N_{\text{BaOH}})}{(N_H) (\text{Ba}^{++}) (\text{OH}^-)^2}$$

$$k_2 = \frac{(N_{\text{BaOl}}) (\text{OH}^-)}{(N_{\text{BaOH}}) (\text{Ol}^-)}$$

large and that no appreciable quantity of adsorbed  $\text{BaOH}$  is present in the systems with oleate added. In other words, the elementary spaces available for ion adsorption are occupied either by  $\text{H}$  or by  $\text{BaOl}$  or they are empty. The stoichiometry of ion exchange adsorptions and the usual preservation of electrical balance indicate that the proportion of empty spaces is very small, so that in our systems for practical purposes,  $N_H + N_{\text{BaOl}}$  should account for substantially all the elementary spaces. Thus we are left with the equilibrium constants  $k_3$  and  $k_4$ , to account for our experimental data.

The contact curves in fig. 1 represent the borderline conditions between floatability and nonfloatability, and thus for a surface covered with  $-\text{H}$  and  $-\text{BaOl}$  groups correspond to a limiting value of  $\frac{N_{\text{BaOl}}}{N_H}$  below which the surface is too hydrophilic for flotation. Thus, re-arranging the expression given above for  $k_4$ ,

$$(\text{H}^+) (\text{Ol}^-) = \frac{N_H}{N_{\text{BaOl}} k_4} = \text{constant along contact curve.}$$

Taking a constant value of  $1.6 \times 10^{-16}$  for  $\frac{N_H}{N_{\text{BaOl}} k_4}$  along the contact curve and a value of  $6.5 \times 10^{22}$  for  $k_3^*$ , the family of theoretical contact curves

\* These values are for concentrations expressed as mols per liter.

shown in fig. 3 was calculated to correspond to the experimental curves given in fig. 1. The two numerical constants were chosen by trial and error to give the best agreement between experiment and theory.

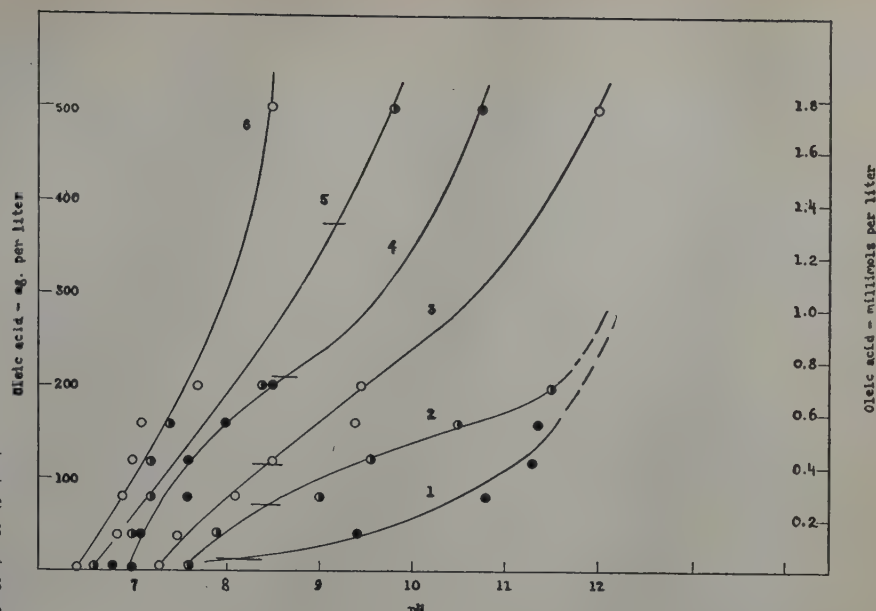
Calculations for the curves were based on the following, in addition to the equilibrium constants already given:  $(\text{H}^+) = 10^{-\text{pH}}$ . With oleate added in stoichiometric excess over barium:  $(\text{Ol}^-) = \text{mols HOl added per liter} - 2 \times (\text{mols BaCl}_2 \text{ added per liter})$ . With barium in stoichiometric excess:  $(\text{Ba}^{++}) = \text{mols BaCl}_2 \text{ added per liter} - \frac{1}{2} \times (\text{mols HOl added per liter})$ . The last two relations represent the fact that the quantities of reagents which react to form the  $\text{BaOl}_2$  precipitate must be subtracted from the added quantities in calculating actual ion concentrations.



**Fig. 4—Effects of  $\text{CaCl}_2$  and oleic acid on critical pH for quartz flotation.**

For given activator addition, flotation occurs under conditions below and to right of curve for that quantity of activator.

Curve No.	$\text{CaCl}_2$ Mg per Liter
1	2.3
2	13.6
3	22.7
4	40.9
5	72.6
6	136.2



Comparison of the experimental curves in fig. 1 with the theoretical curves in fig. 3 shows a remarkable similarity, especially when it is kept in mind that the positions and shapes of the theoretical curves were fixed by choice of only two arbitrary constants for the whole family of curves. We believe the correspondence close enough to represent strong support of the validity of the chemical equations given earlier in accounting for the behavior of barium-activated quartz.

Closer comparison of experimental and theoretical curves, however, shows serious deviations from the proposed ideal behavior when the oleic acid is present in stoichiometric excess over the barium; that is, in the upper right-hand legs of the curves in figs. 1 and 3. The deviations might be described as follows: In the theoretical development, it was assumed (1) that a stoichiometric precipitate of  $\text{BaOl}_2$  is formed and has a definite solubility product and (2) that this precipitate itself does not affect the floatability of quartz. These assumptions lead to the conclusion reflected in fig. 3 that critical pH should be independent of the amount of  $\text{BaOl}_2$  precipitate and independent of the amounts of  $\text{Ba}^{++}$  and  $\text{HOl}$  added separately but reacting in solution to form the precipitate. The experimental data show a definite variation of critical pH with calculated quantity of  $\text{BaOl}_2$ , indicating that for alkaline pulps with excess soap assumption 1 or 2 above, or both might be in error. As will be seen, this deviation becomes even more marked with calcium activation.

### Calcium Activation of Quartz

Vacuum-flotation tests of calcium activation were made over the following ranges of reagent additions:

	Mg per Liter	Millieq. per Liter
$\text{CaCl}_2$	2 - 136	0.04 - 2.5
Oleic acid	5 - 500	0.018 - 1.8

The contact curves are given in fig. 4. These curves are broadly similar to those for barium-activated quartz, in that the critical pH is markedly increased by increasing the quantity of oleic acid. Also, within the pH range 8 to 10 the quartz is floatable if the molal ratio  $\text{Ol}:\text{Ca}$  is less than about 2:1 (indicated by heavy dash across contact curves), but is not floatable if the oleate is definitely in excess of this ratio. Thus, as with Ba, there are conditions under which Ca-activated quartz can be depressed by adding excess oleic acid. As with Ba, we believe the oleic acid deactivates the quartz, putting the Ca into the form of the insoluble soap, by a reaction analogous to reaction 4. We have observed related

phenomena in soap flotation of tin ores in moderately hard water. That is, first addition of collector produced a high-silica, nonselective froth and subsequent conditioning with more collector decreased the quartz content of the float.

Calcium, unlike barium, exhibited an upper critical pH above which the quartz did not float. Because of difficulties in measuring the very high pH values, no pH data are reported. However, quartz did not float with Ca activation in 1-normal sodium hydroxide. These observations correlate with the fact that  $\text{Ca(OH)}_2$  is less soluble than  $\text{Ba(OH)}_2$ . Hence it is possible that nonfloatation above the upper critical pH is due to displacement of  $\text{Ol}$  by  $\text{OH}$  at the Ca-activated surface; that is, the Ca reaction corresponding to Eq 2 is reversible at high pH values. Another explanation is that above the upper critical pH the Ca precipitates as  $\text{Ca(OH)}_2$ , leaving the  $\text{Ca}^{++}$  concentration in solution too low to maintain activation.

Although the Ca-activation data show the same general trends as the Ba-activation data, the Ca-activation data deviate much further from expectations based on the theory given earlier for Ba-activation. These deviations are most marked with oleic acid in stoichiometric excess over calcium. Under these conditions, the critical pH appears to be a function of quantity of calcium added. If all the calcium were precipitated as  $\text{CaOl}_2$  it would be expected that the concentration of  $\text{Ca}^{++}$  and hence its action as an activator would be controlled by the solubility product of the  $\text{CaOl}_2$ , and would be independent of the quantity of calcium in the precipitated form.

The data for barium and calcium activation afford some understanding of the practical problem of soap floating in hard water, without floating quartz. In the pH range from neutrality up to pH 10 or 11 quartz tends to float when either Ba or Ca is in stoichiometric excess over oleic acid. If the quantity of Ba or Ca is not too great, the quartz flotation can be overcome by increasing the collector addition.

### Aluminum Activation

Before making flotation tests with aluminum as activator, a series of potentiometric titrations were made to gain a little acquaintance with the chemical reactions in solutions to which aluminum and

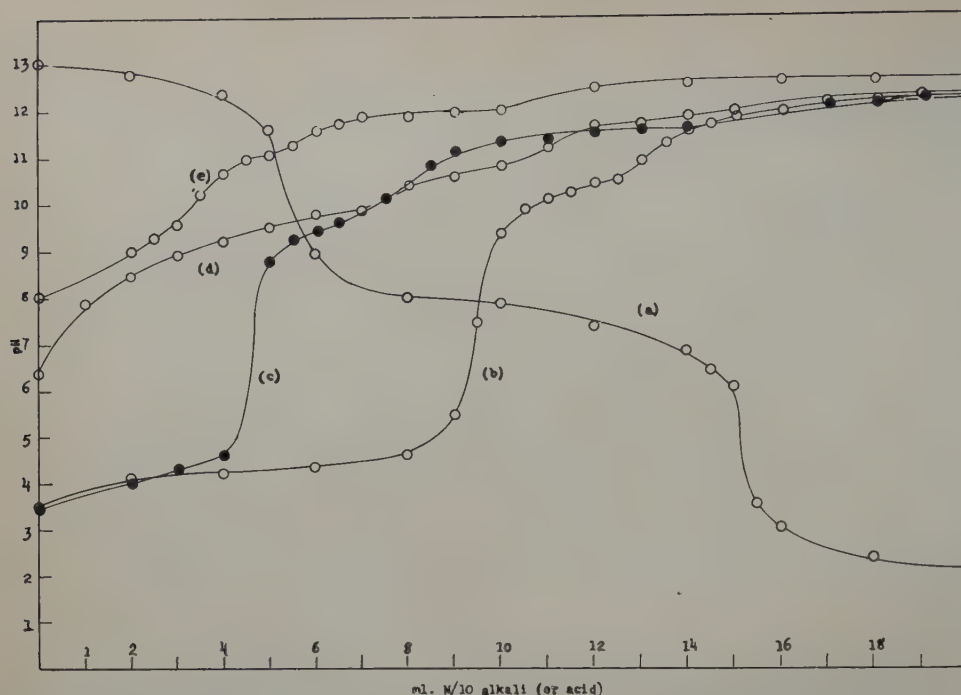


Fig. 5—Titration curves for  $\text{AlCl}_3$ , HOI, and mixtures.

Curve No.	Initial Solution	Titrated with
(a)	10 ml N/10 HOI + 15 ml N/10 NaOH	HCl
(b)	10 ml N/10 $\text{AlCl}_3$	NaOH
(c)	5 ml N/10 HOI + 5.5 ml N/10 NaOH + 10 ml N/10 $\text{AlCl}_3$	NaOH
(d)	7.5 ml N/10 HOI + 7.5 ml N/10 NaOH + 7.5 ml N/10 $\text{AlCl}_3$	NaOH
(e)	10 ml N/10 HOI + 10.5 ml N/10 NaOH + 5 ml N/10 $\text{AlCl}_3$	NaOH

oleic acid are added in varying amounts. The resulting titration curves are given in fig. 5. Our interpretations of these data are as follows:

1. Sodium oleate reacts with a strong acid to produce oleic acid in the pH range 7 to 8. Below pH 7 then, added oleic acid will be present largely as undissociated acid; above pH 8, added oleic acid will be largely in the form of oleate ions (fig. 5a).

2. Aluminum precipitates as  $\text{Al}(\text{OH})_3$  in the vicinity of pH 4. Further additions of alkali raise the pH and then between 9.5 and 11 the precipitate dissolves as  $\text{AlO}_2^-$  (fig. 5b).

3. Aluminum oleate is stable below pH 7, at least down to pH 4 (fig. 5c) and perhaps well below pH 4.

4. As the pH is increased from 8 to 12, aluminum oleate reacts with alkali, probably in several steps. Elliott<sup>8</sup> has indicated the existence of mono, di, and tri-soaps. When the pH reaches 12, all the soap is dissolved with the formation of  $\text{AlO}_2^-$  and  $\text{OI}^-$  (fig. 5d,e). This conclusion fits the stoichiometry of the titration curves and checks also with the observed dissolution of precipitated soap at high pH values.

The vacuum-flotation tests of aluminum activation covered wide ranges of reagent additions in an exploratory fashion:

	Mg per Liter	Millieq. per Liter
$\text{AlCl}_3$	0.3 - 285	0.006 - 6.4
Oleic acid	0.7 - 360	0.0025 - 1.3

As might be expected, flotation behavior with Al-activation is somewhat more complex than with Ba-activation. In particular, Al-activation showed both a lower and an upper critical pH, with flotation between the two, and nonflotation outside the two limits.

To avoid confusion from overlapping curves, the data are plotted in two separate graphs, fig. 6a for smaller amounts of activator (0.3 to 18 mg per liter) and fig. 6b for larger amounts of activator (18 to 285 mg per liter). In these tests it was found more convenient to determine the critical pH for near-complete flotation rather than the critical pH for incomplete flotation (as was done with Ba-activation).

The upper critical pH falls in the range 12 to 13, regardless of the quantities of activator and collector added. This upper pH limit to quartz flotation

with Al-activation may be accounted for by the fact that  $\text{Al}^{+++}$  is converted to  $\text{AlO}_2^-$  at high alkalinities.

For each quantity of activator, there is a wide range of collector additions giving a lower critical pH between 4 and 5. Effective flotation from a lower critical pH of 4 to 5 up to an upper critical pH of 12 or 13 was observed when the oleic acid was below stoichiometric equivalence to the formation of  $\text{Al}(\text{OI})_3$  with the activator. As the oleic acid was raised above this equivalence point, the lower critical pH increased rapidly so that with an excess of oleic acid, flotation was possible only in a narrow pH range near 12 (fig. 6a). Thus, as with Ba and Ca, we believe that excess oleic acid deactivates the quartz by converting the Al into an insoluble soap precipitate by a reaction similar to reaction 4.

Lower critical pH remained from 4 to 5 with more than 10-fold excess of activator, but with very large ratios of activator to collector (bottom portions of fig. 6b) the critical pH increased to give a narrower flotation range. No explanation of these effects should be attempted without more experimental data.

### Iron Activation

Iron, both ferrous and ferric, is a common constituent of flotation pulps. Ferrous salts are unstable in the presence of air, especially in neutral or alkaline pulps, so our experiments were confined to ferric activation.

A number of potentiometric titrations were made to explore the reactions between ferric salts, oleic acid, and alkalis. Results were as follows:

1. Ferric hydroxide forms between pH 2 and 3. The precipitate did not flocculate until the pH reached about 7 (fig. 7a).

2. Ferric oleate is stable over the pH range from below 2 to above 7, neither decomposing to yield oleic acid at low pH nor reacting with alkali to form ferric hydroxide at high pH (fig. 7b,c,d). In fig. 7d the inflection between pH 7 and 8 corresponds to the saponification of the excess oleic acid.

The vacuum-flotation tests of iron activation did not yield reproducible and consistent results, showing that a great deal of study involving other than flotation techniques may be necessary to elucidate the various phenomena involved. In the first place,



the phenomena were not reversible so that the floatability depended upon the procedure used in preparing the pulp. Secondly, precipitates formed from the solution tended to smear the quartz visibly.

Exploratory vacuum flotation tests of ferric activation covered the following ranges of reagent addition:

	Mg per Liter	Millieq. per Liter
FeCl <sub>3</sub>	0.3-175	0.006-3.2
Oleic acid	1.8-450	0.006-1.6

The data are too rough to justify the plotting of contact curves, but did bring out some points of interest:

1. Ferric iron activates quartz at pH values from as low as 3 to as high as 12. That is, it can serve as activator under conditions where Fe<sup>+++</sup> concentration must be extremely low, owing to precipitation of the hydroxide. This is consistent with the titration data, which indicated ferric oleate to be more stable than ferric hydroxide.
2. Both a lower and an upper critical pH were always found. Quartz was not floated under any conditions below a lower critical pH of 3 or above an upper critical pH of 12.
3. For a given quantity of activator, flotation was possible only with collector additions within a less than 10-fold variation from stoichiometric equivalence to the activator. Large excesses of either

activator or collector resulted in nonflotation.

4. Various complexities were observed, some more or less reproducible, such as an additional pH range of nonflotation between the lower and upper critical pH.

**Tin Activation**

Quartz could not be floated at any pH with stannic chloride as activator (35 to 175 mg per liter) and oleic acid (18 to 150 mg per liter). Since stannous salts oxidize easily to stannic under flotation conditions, it appears likely that tin is not a quartz activator in the soap flotation of tin ores.

**Summary and Conclusions**

Chemical conditions for flotation and for non-flotation of quartz with oleic acid as collector and barium, calcium, aluminum, iron, and tin as activators were studied. A vacuum-flotation technique using standard 100-ml glass-stoppered graduates as flotation cells was developed and used for all the experimental work. The experimental procedure was to vary the quantities of collector and activator, and for each reagent combination to determine the critical pH which roughly separates conditions of flotation from conditions of nonflotation.

Except for iron activation, the critical pH could be approached from either side and did not depend

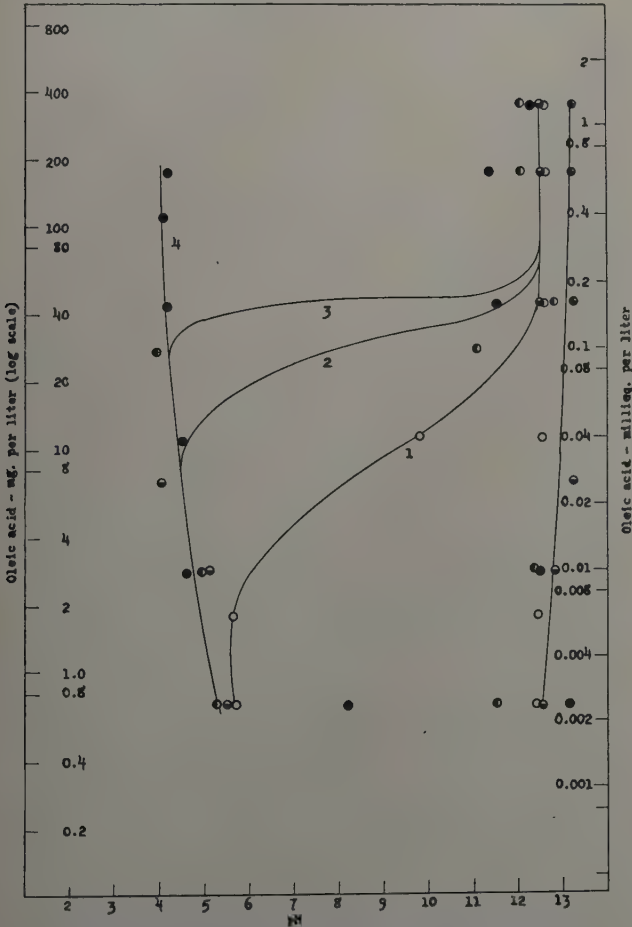


Fig. 6a—Effects of AlCl<sub>3</sub> and oleic acid on pH range for quartz flotation.

For given activator addition, flotation occurs under conditions between corresponding lower critical pH curve at left and upper critical pH curve at right.

Curve No.	Mg per Liter	Millieq. per Liter
1	0.3	0.006
2	1.1	0.025
3	4.5	0.1
4	18.0	0.4

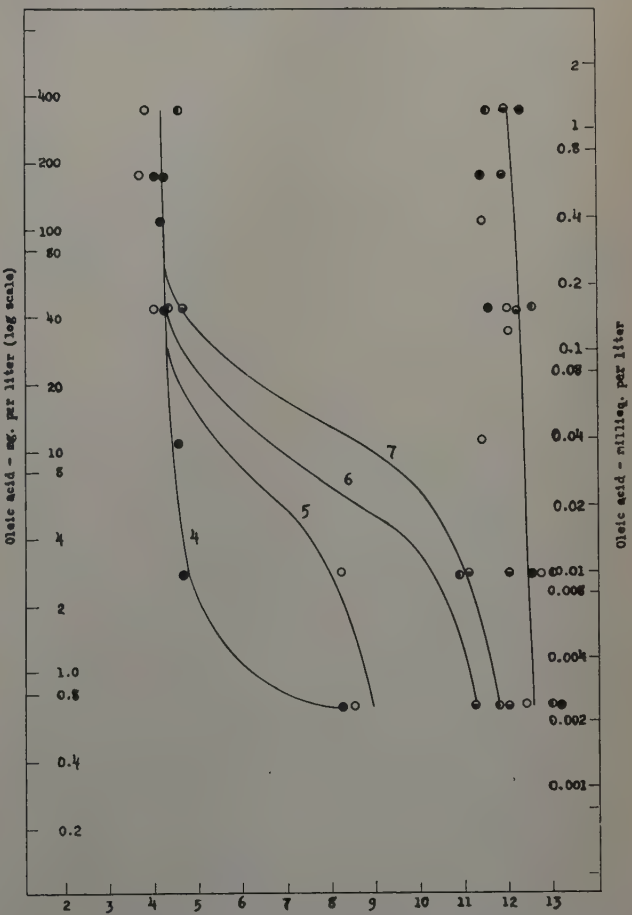


Fig. 6b—Effects of AlCl<sub>3</sub> and oleic acid on pH range for quartz flotation.

For given activator additions, flotation occurs under conditions between corresponding lower critical pH curve at left and upper critical pH curve at right.

Curve No.	Mg per Liter	Millieq. per Liter
4	18	0.4
5	36	0.8
6	142	3.2
7	285	6.4

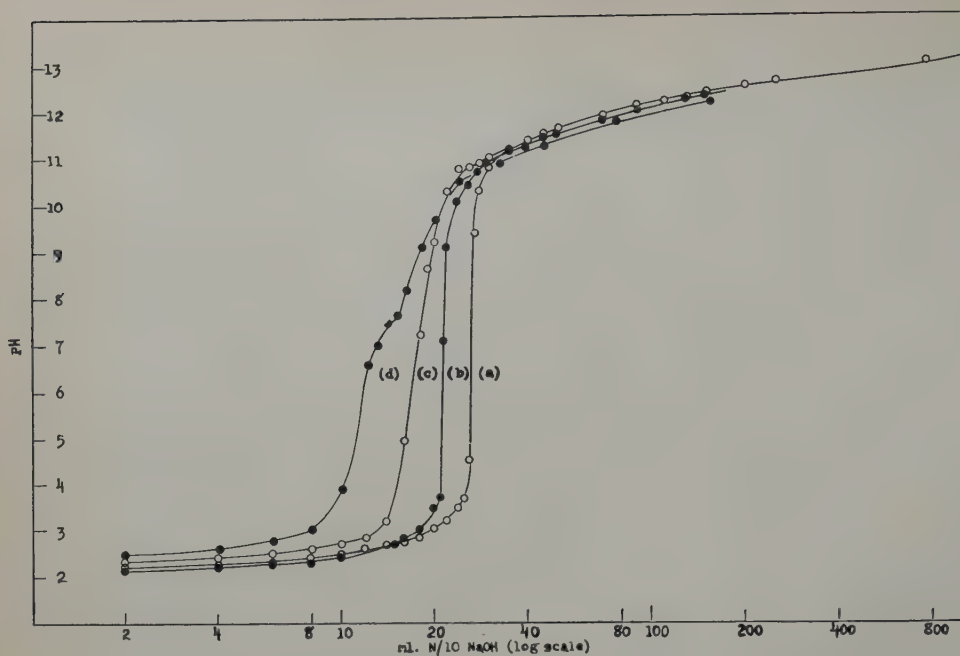


Fig. 7—Titration curves for  $\text{FeCl}_3$ ,  $\text{HCl}$ , and mixtures.

Curve No.

- (a) 10 ml N/10  $\text{FeCl}_3$  + 16 ml N/10  $\text{HCl}$
- (b) 10 ml N/10  $\text{FeCl}_3$  + 16 ml N/10  $\text{HCl}$  + 50 ml N/100  $\text{NaOH}$
- (c) 10 ml N/10  $\text{FeCl}_3$  + 16 ml N/10  $\text{HCl}$  + 100 ml N/100  $\text{NaOH}$
- (d) 10 ml N/10  $\text{FeCl}_3$  + 16 ml N/10  $\text{HCl}$  + 150 ml N/100  $\text{NaOH}$

on order of reagent additions. Thus many of the reactions and adsorptions involved in soap flotation of quartz are reversible, and chemical equilibrium can be closely approached during conditioning. Accordingly, a modified mass-action relation was developed for reactions involving adsorbed substances. This relation is based on Langmuirian chemisorption as an ideal case.

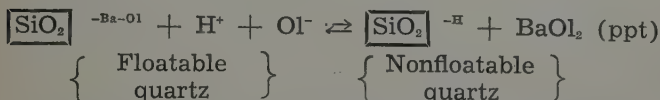
Specific conclusions are summarized below:

1.  $\text{Ba}^{++}$ ,  $\text{Ca}^{++}$ ,  $\text{Al}^{+++}$ , and  $\text{Fe}^{+++}$  all serve as activators for quartz over wide pH ranges, provided they are present in stoichiometric excess over the quantities required to form normal soaps with the oleic acid.  $\text{Ba}^{++}$  and  $\text{Ca}^{++}$ , however, are effective activators only above pH 7 or so, whereas  $\text{Al}^{+++}$  and  $\text{Fe}^{+++}$  are effective over most of the pH range from 4 to 12 or 13.

2. An excess of oleic acid over the quantity required to form a normal soap with the activator tends to depress quartz, in that flotation is possible only at relatively high pH values. This can be accounted for by a deactivation reaction which removes the activating cation from the quartz into a very insoluble metal-soap precipitate.

3. Except possibly at high pH values (namely, 12 and above) flotation or nonflotation of quartz is contingent primarily on activation or nonactivation; collection of activated quartz takes place if oleate is present in the system in small amounts in any form. Bases of this conclusion are:

- a. Upper critical pH values, which might correspond to displacement of  $\text{Ol}^-$  by  $\text{OH}^-$  at the quartz surface, are very high (12 or above for  $\text{Al}$ ,  $\text{Fe}$ ,  $\text{Ca}$ ) or nonexistent ( $\text{Ba}$ ).
- b. Adding large excesses of metal ions over the quantities required to precipitate all added oleic acid as insoluble soaps does not depress the quartz.
4. Flotation or nonflotation of quartz is related to equilibria in reactions of the type:



For the specific case of barium activation, application of mass-action reasoning to this reaction leads

to the relation  $(\text{H}^+)(\text{Ol}^-) = \text{constant}$  for borderline conditions between flotation and nonflotation. The experimental critical-pH curves approximate the theoretical curves calculated from this relation.

5. Aluminum and ferric iron are effective quartz activators in pulps where the concentrations of aluminum or ferric ions must be very small, owing to their precipitation as hydroxides.

6. Stannic tin does not activate quartz.

## References

- <sup>1</sup>I. W. Wark: Principles of Flotation. Australasian Inst. of Min. and Met. (1938).
  - <sup>2</sup>J. C. Smith: Higher Aliphatic Compounds. VIII, Purification of Oleic and Elaidic Acids. *Jnl. of Amer. Chem. Soc.* 974-980 (1949).
  - <sup>3</sup>D. H. Wheeler and R. W. Riemenschneider: Preparation and Properties of Highly Purified Methyl Oleate. *Oil and Soap* (1939) 16, 207.
  - <sup>4</sup>J. B. Brown and A. Y. Shinowara: Preparation of Pure Oleic Acid by a Simplified Method. *Jnl. of Amer. Chem. Soc.* (1937) 59, 6-8.
  - <sup>5</sup>A. M. Gaudin and A. Rizo-Patron: The Mechanism of Activation in Flotation. *Trans. AIME* (1943) 153, 462.
  - <sup>6</sup>I. Langmuir: The Constitution and Fundamental Properties of Solids and Liquids. Part I. Solids. *Jnl. of Amer. Chem. Soc.* (1916) 38, 2221-2295. Also, The Adsorption of Gases on Plane Surfaces of Glass, Mica, and Platinum. *Jnl. of Amer. Chem. Soc.* (1918) 40, 1361-1403.
  - <sup>7</sup>G. E. Boyd, J. Schubert, and A. W. Adamson: The Exchange Adsorption of Ions from Aqueous Solutions by Organic Zeolites. I. Ion-exchange Equilibria. *Jnl. of Amer. Chem. Soc.* (1947) 69, 2818-2829.
  - <sup>8</sup>S. B. Elliott: The Alkaline-Earth and Heavy-Metal Soaps. Page 183. Reinhold Publishing Company (1946).
  - <sup>9</sup>A. M. Gaudin and R. T. Hukki: Principles of Comminution—Size and Surface Distribution. *Trans. AIME* (1946) 169, 67; *Min. Tech.* (Nov. 1944), TP 1779.
- The authors' attention has been called to the following papers which have appeared since their paper was submitted for publication and which present a more detailed study of some aspects of the calcium activation of quartz than that presented herein.
- <sup>10</sup>S. R. B. Cooke and M. Digre: Studies on the Activation of Quartz with Calcium Ion. *Trans. AIME*, 184, 299-305; *Min. Eng.*, Aug. 1949.
  - <sup>11</sup>S. R. B. Cooke: The Flotation of Quartz Using Calcium Ion as Activator. *Trans. AIME*, 184, 306-309; *Min. Eng.*, Aug. 1949.



# Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite

by  
R. Schuhmann, Jr.,  
and  
Brahm Prakash

Chemical conditions for flotation and nonflotation of cassiterite and fluorite with oleic acid as collector and with alizarin dyes as modifying agents were studied by means of small-scale, vacuum-flotation tests. Artificial cassiterite was prepared for these tests by high-temperature methods, especially by crystallization from borax melts.

**F**LOTATION test work on Bolivian tin ores some years ago led to the development of several workable schemes of selective soap flotation, but left unanswered many questions of soap-flotation chemistry.<sup>1</sup> Accordingly, the study of the activation and soap flotation of quartz just reported<sup>2</sup> was undertaken as a start toward the elucidation of some of the unanswered questions. The experimental work on quartz flotation consisted primarily of small-scale, vacuum-flotation tests in which activator and collector additions and pH were varied between wide limits.

In the work to be described in this paper, the soap flotation of cassiterite and fluorite was studied by the vacuum-flotation technique developed in the previous study of quartz. Using oleic acid as collector, conditions for floating cassiterite and fluorite in the absence of modifying agents were first determined. Then the effects of barium, calcium, and alizarin dyes on the flotation of cassiterite, fluorite, and quartz were studied. Alizarin dyes were chosen for study because of the promise they have shown as selective modifying agents in the flotation of non-sulphide minerals.<sup>12, 1</sup> Also, they form a family of reagents in which there should be a systematic correlation of chemical properties with composition. As

---

R. SCHUHMAN, JR., Member AIME, is Associate Professor, Department of Metallurgy, Massachusetts Institute of Technology, Cambridge, Mass., and BRAHM PRAKASH, Student Associate AIME, was a Government of India Scholar, MIT.

This paper is based on a thesis submitted by B. Prakash in partial fulfillment of the requirements for the degree of Doctor of Science at MIT.

AIME New York Meeting, Feb. 1950.

TP 2827 B. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received July 26, 1949.

---

a preliminary to the study of cassiterite flotation, an investigation of the preparation of synthetic cassiterite and X ray studies of natural and synthetic cassiterite were undertaken.

## Experimental Methods

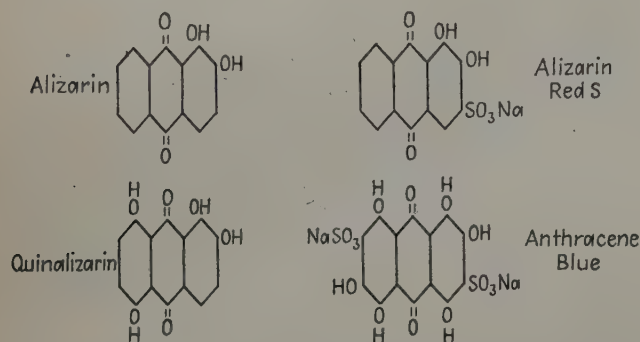
The flotation technique and most of the reagent preparations were described fully in the previous paper.<sup>3</sup> In addition, for the present work it was necessary to prepare satisfactory stocks of clean cassiterite and fluorite and to obtain samples of alizarin dyes.

**Cassiterite:** High-grade cassiterite from the Netherlands East Indies was crushed, sized, and cleaned

carefully by gravity and magnetic separation. The concentrate was wet ground, deslimed, and then leached with hot HCl twice and washed thoroughly with distilled water. The product was not floatable with a frother alone. However, the mineral was still brown and analyzed only 97.5 pct SnO<sub>2</sub>. Other chemical treatments with strong alkali and HF, also ignition at 1100°C, did not improve the color or purity. Hence, it seemed desirable to attempt the artificial preparation of cassiterite, to avoid a possible error from the presence of an activating constituent in the natural mineral. Details of making the artificial cassiterite are given in a subsequent section.

**Fluorite:** Fluorite crystals free from interference colors were hand-picked from a stock of high-grade material (99.6 pct CaF<sub>2</sub>) originally obtained from Ward's Natural Science Establishment. The selected pieces were extracted with ether in a Soxhlet-type apparatus. Subsequent steps in preparation were: (1) wet crushing in mortar and pestle, (2) pebble-mill grinding, (3) alkali leach (4) HCl leach, (5) washing, and (6) desliming at 18 microns. The deslimed product was stored under water. It was all minus 200 mesh, plus 18 microns and had an estimated specific surface of 1500 cm<sup>2</sup> per g. It was not floatable with a frother alone.

**Alizarin Dyes:** The four dyes tested as flotation reagents were the following:



As the formulas show, these dyes are similar structurally but differ in the numbers and locations of —OH and —SO<sub>3</sub>Na groups substituted for —H in the common anthraquinone structure.

Alizarin and alizarin red S were of indicator grade and supplied by Allied Chemical and Dye Corp. Quinalizarin was Eastman, high-purity dye. The Anthracene blue W.R.S. was of commercial grade supplied by E. I. du Pont de Nemours and Co.

#### Synthesis of Cassiterite and X Ray Studies\*

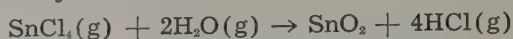
\* The authors wish to acknowledge the generous assistance, both in advice and in making equipment available, given in the X ray studies by Prof. J. T. Norton of the Metallurgy Department and Mr. A. J. Frueh of the Geology Department, MIT.

A variety of syntheses of crystalline stannic oxide are described in the literature, but experimental details and proofs of identity of the products are meager. Two methods were selected and tried:

1. Reaction of SnCl<sub>4</sub> and H<sub>2</sub>O in a hot tube.<sup>3</sup>
2. Crystallization from borax, stannic oxide melts.<sup>4, 5</sup>

The cassiterite used for flotation tests was prepared eventually by crystallization from borax melts, but some interesting results were obtained in attempting to react stannic chloride and water.

**Reaction of SnCl<sub>4</sub> and H<sub>2</sub>O:** First, thermodynamic calculations were made which checked the feasibility of forming SnO<sub>2</sub> at temperatures around 1000°C by the reaction:



The thermodynamic data were taken from Kelley.<sup>6</sup> After considerable preliminary experimentation the apparatus was set up as follows: The reaction tube was either an alundum or a Vycor combustion tube passing through a Burrell furnace which could be held at any desired temperature up to well over 1000°C. Stannic chloride gas and steam, each mixed with nitrogen, were introduced separately into the reaction tube. Quantities of SnCl<sub>4</sub> and H<sub>2</sub>O were each controlled as follows: A metered flow of N<sub>2</sub> was saturated with the vapor at a preset temperature by bubbling the N<sub>2</sub> through the liquid (SnCl<sub>4</sub> or H<sub>2</sub>O) held in a container in a heated oil bath. Vapor-pressure data from Stull's compilation<sup>7</sup> were used in calculating the flow rates of SnCl<sub>4</sub> and H<sub>2</sub>O. Generally a small stoichiometric excess of SnCl<sub>4</sub> was used.

Variations in temperature (range 800 to 1000°C), flow rates, SnCl<sub>4</sub> to H<sub>2</sub>O ratio, positions of inlet tubes, and so on, gave a variety of crystalline deposits of various colors in the refractory tube. However, the yields were consistently very small and the colors were suggestive of contamination. Crystals from one run at 800°C were carefully separated for analysis, and contained 99.1 pct SnO<sub>2</sub>. Debye-Scherrer diffraction patterns indicated this material to be crystallographically identical with natural cassiterite and with artificial cassiterite prepared by crystallization from borax melts.

In one run at 1000°C an alundum boat containing several grams of precipitated SnO<sub>2</sub> was placed in the reaction tube in the hope of getting recrystallization of the added oxide. No change was observed in the contents of the alundum boat, but at the end of the experiment a few milligrams of colorless, transparent needlelike crystals were found deposited on one end of the boat. Iodimetric analysis of a small sample gave 95.8 ± 5 pct SnO<sub>2</sub>.

Since there has been some disagreement in the literature as to the crystal structure of artificial stannic oxides,<sup>8, 9, 10</sup> a brief X ray study was made of the needlelike crystals formed at 1000°C. Photographs were taken on the single-crystal rotational and the Weissenberg goniometric cameras in the Metallurgy Department X ray laboratory. From this work it was found that the crystals were orthorhombic, with the following lattice constants:

$a = 2.66 \text{ \AA}$ ;  $b = 4.81 \text{ \AA}$ ;  $c = 5.84 \text{ \AA}$ . These results confirm Daubree's<sup>3</sup> crystallographic observations of a hundred years ago that stannic oxide was dimorphous. Subsequent workers<sup>8, 9, 10</sup> disputed the existence of the orthorhombic variety and no mention of it was found in recent literature. Accordingly, we believe our measurements of the lattice constants represent the first X ray study of this form of SnO<sub>2</sub>.

Our work confirms the feasibility of preparing crystalline SnO<sub>2</sub> by reaction of SnCl<sub>4</sub> and H<sub>2</sub>O but shows also that considerable further experimentation will be necessary before satisfactory yields of high-purity crystals can be reproduced. By controlling the conditions it appears that either the usual tetragonal cassiterite or an unusual orthorhombic stannic oxide can be made.

**Crystallization from Borax Melts:** Fair yields of high-purity tetragonal cassiterite suitable for float-



tion tests were obtained by the borax-crystallization procedure. Tin metal (c.p. reagent) was converted to dioxide by treatment with dilute nitric acid followed by ignition of the washed stannic acid. Borax (c.p. reagent) was fused in platinum to obtain borax glass. Mixtures of the dioxide and the borax glass containing from 5 to 25 pct dioxide were fused in platinum and then cooled by various procedures.

**Table I. Crystallization of Cassiterite from Borax Melts**

	Test A	Test B
Per cent SnO <sub>2</sub> in melt	15	15
Temperature of fusion	1375°C	1200°C
Time at fusion temperature	14 hr	14 hr
Method of cooling	Cooled in furnace to 1000°C, held for 6 hr, then cooled to room temperature in furnace	
Approximate crystal size	2-20 microns	1-5 microns

A further study of the artificial cassiterite, using the Norelco X ray spectrometer to obtain quantitative measurements of line intensities, showed close agreement of relative line intensities with the calculated values for natural cassiterite.

Details of the various X ray data and calculations discussed above are available in the original thesis from which this paper is abstracted.

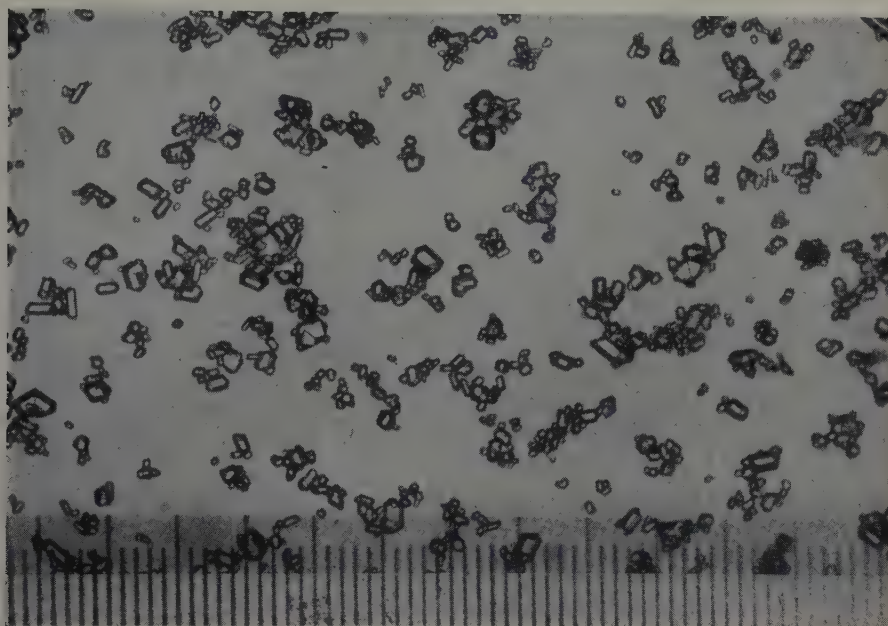
#### Cassiterite Flotation

Vacuum-flotation tests were made with about 0.8 g of 2 to 20 micron artificial cassiterite per 100 ml pulp in each test. Critical pH values were determined for oleic acid additions from 0.28 up to 290 mg per liter. Exploratory tests were also made with BaCl<sub>2</sub> · 2H<sub>2</sub>O (50 mg per liter) and CaCl<sub>2</sub> (41 mg per liter) added to the pulp. The data are plotted in fig. 2.

Cassiterite floats readily without activation over

**Fig. 1—Artificial cassiterite crystallized from borax melts.**

One division equals ten microns.



The borax was separated by water-leaching the solidified melt. The cassiterite crystals were further cleaned with HCl, washed, and stored under water. Data for two typical preparations are summarized in table I. Test A represents the procedure finally used in preparing a stock of cassiterite for flotation tests.

Under the microscope the artificial cassiterite is colorless and appears identical with natural cassiterite, even showing the characteristic twinning of the natural variety (fig. 1). The powder in mass is white in color, in contrast to the brownish color of the cleaned natural cassiterite. Chemical analysis gave 99.8 pct SnO<sub>2</sub>.

Powder diffraction patterns of the artificial and natural cassiterites were compared and found identical using the Debye-Scherrer camera. Also Phragmen diffraction patterns of the artificial material were obtained. From these, the lattice constants were calculated and compared with Vegard's<sup>11</sup> data for natural cassiterite, with the following results:

	Artificial	Natural (Vegard)
<i>a</i>	4.739 Å	4.72 Å
<i>c</i>	3.186 Å	3.17 Å
<i>c</i> / <i>a</i>	0.6714	0.6723

a wide pH range. Both a lower and an upper critical pH were found, with flotation occurring over the entire range in between. The lower critical pH decreases with increase in added oleic acid, from about pH 7 with 1 mg oleic acid per liter down to pH 2 with about 300 mg oleic acid per liter. The data approximate a straight line of slope -0.5 on the log-log plot, the line in fig. 2 following the relation:

$$\frac{(\text{Critical } H^+ \text{ conc.})}{(\text{HOI addition})^2} = 1.4 \times 10^4$$

when the units of both quantities are mols per liter. The deviation of the data from this relation below 1 mg oleic acid per liter is readily accounted for, since in this range of collector addition a large proportion of the collector may be on the mineral so that quantity added is no longer a measure of quantity in solution.

The flotation of cassiterite with oleic acid at pH values as low as 2 and 3 was unexpected in the light of previous experience in the flotation of Bolivian tin ores. Additional vacuum-flotation tests with cleaned natural cassiterite checked the results with the artificial cassiterite. As a further check several flotation tests were carried out on larger quantities of less carefully prepared natural cassiterite and

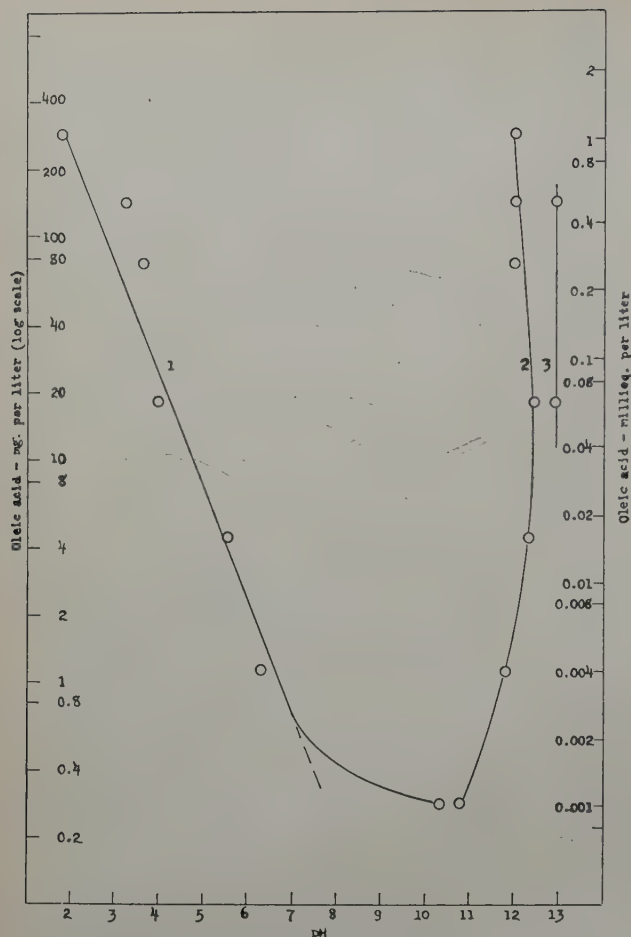


Fig. 2—Soap flotation of cassiterite.

Flotation occurs under conditions between lower and upper critical pH curves.

Curve 1, lower critical pH, in presence and absence of activators.

Curve 2, upper critical pH, no activator.

Curve 3, upper critical pH, with 41 mg  $\text{CaCl}_2$  per liter.

No upper critical pH found with 50 mg  $\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$  per liter.

cassiterite-quartz mixtures, using standard grinding and flotation procedures and the Fagergren 500-g cell. In these larger-scale tests the cassiterite was floated with oleic acid at pH values down to 2. In one test on 500 g of a 1:1 mixture at pH 3, 0.39 lb oleic acid per ton gave 97.5 pct recovery of tin in a rougher float assaying 74.1 pct Sn.

The decrease in lower critical pH with increasing collector addition contrasts with the previously reported behavior of quartz activated with  $\text{Ba}^{++}$ ,  $\text{Ca}^{++}$ ,  $\text{Al}^{+++}$ , or  $\text{Fe}^{+++}$ . For activated quartz, increasing collector increased the lower critical pH, and excess collector tended to depress the quartz, presumably by deactivating it. The sharply contrasting behavior of the cassiterite and the activated quartz on increasing the collector argue against an explanation of cassiterite flotation in terms of accidental activation. However, we have been unable to develop an explanation of the chemistry of cassiterite flotation which will account for the observed relation between collector addition and lower critical pH.

The upper critical pH remained in the pH range from 11 to a little over 12, substantially constant over the 1000-fold range of collector addition tested. The upper critical pH was found by approach from the low pH side, and was irreversible in that non-flotation persisted to near neutrality when the pH

was dropped from a high value. This phenomenon calls for further study.

$\text{Ba}^{++}$  and  $\text{Ca}^{++}$  had no effects on the lower critical pH, indicating absence of activation in acid pulps. However, in alkaline pulps the  $\text{Ba}^{++}$  eliminated the upper critical pH entirely while the  $\text{Ca}^{++}$  raised the upper critical pH by substantially a full unit. All these effects parallel the behavior of  $\text{Ba}^-$  and  $\text{Ca}^-$  activated quartz. Activation of cassiterite by Ca at pH's in the vicinity of 12 was observed previously in the flotation testing of Bolivian tin ores.<sup>1</sup>

**Effect of Alizarin Dyes:** Vacuum-flotation tests were conducted with varying quantities of alizarin, alizarin red, and quinalizarin and the following activator-collector combinations:

Mg per Liter		
Oleic Acid	$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$	$\text{CaCl}_2$
18.05	0	0
18.05	50	0
18.05	0	23.5

Lower and upper critical pH values for these tests are plotted in fig. 3, with pH values as abscissas and dye concentrations (log scale) as ordinates.

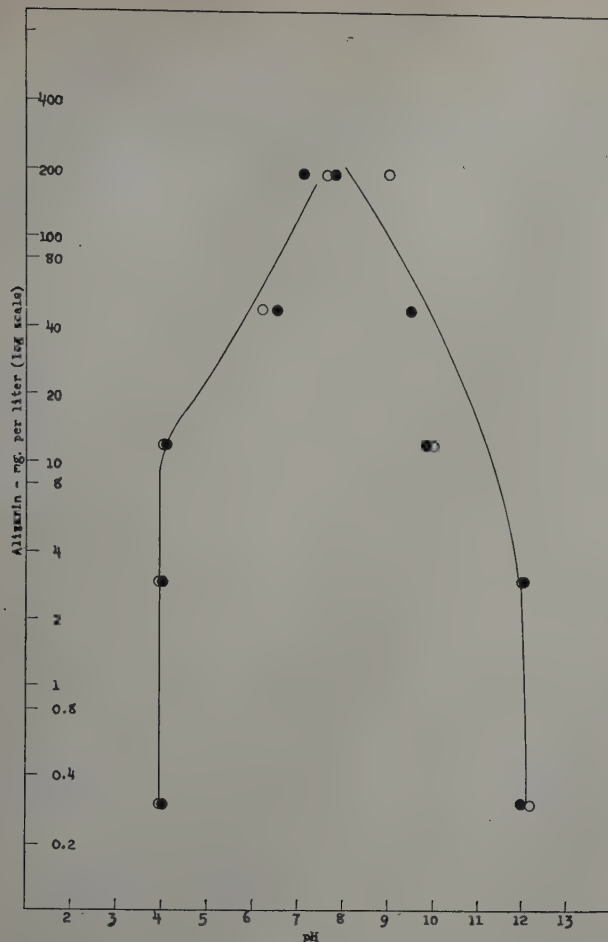
All the curves are of similar shape, showing that all three dyes act as depressing agents for cassiterite both in the absence of an activator and in the presence of  $\text{Ba}^{++}$  and  $\text{Ca}^{++}$ . In general, as dye concentration is raised, the flotation range between lower and upper critical pH narrows until a dye concentration is reached above which flotation is not obtained at any pH.

In the absence of an activator, 200 or 300 mg per liter of any one of the three dyes was required to secure complete depression of cassiterite over the entire pH range. Such concentrations of dye correspond roughly in magnitude to a pound or so of dye per ton under ordinary flotation conditions. Moreover, these quantities of dye are two or three orders of magnitude greater than would be needed to film the cassiterite monomolecularly. Clearly, unactivated cassiterite is not depressed easily with small dye additions, and this may prove advantageous.

Three of the curves involving the activator-dye combinations barium-alizarin red S, barium-quinalizarin, and calcium-quinalizarin showed marked enhancement of the depressing action of the dye through the presence of the alkaline-earth cation. For example, in fig. 3b it can be seen that complete depression at all pH values was obtained with 50 mg  $\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$  per liter and just over 10 mg alizarin red S per liter, whereas without the barium over 200 mg alizarin red S per liter was required for complete depression. The role of the barium in aiding the depression of the cassiterite appears analogous to its activating role in other systems, where it couples the collector to the mineral. However, in this system the barium couples the dye instead and thus assists depression. Pending further study and clarification, we have labeled this phenomenon "depression-sensitization" and the barium a "depression-sensitizer."

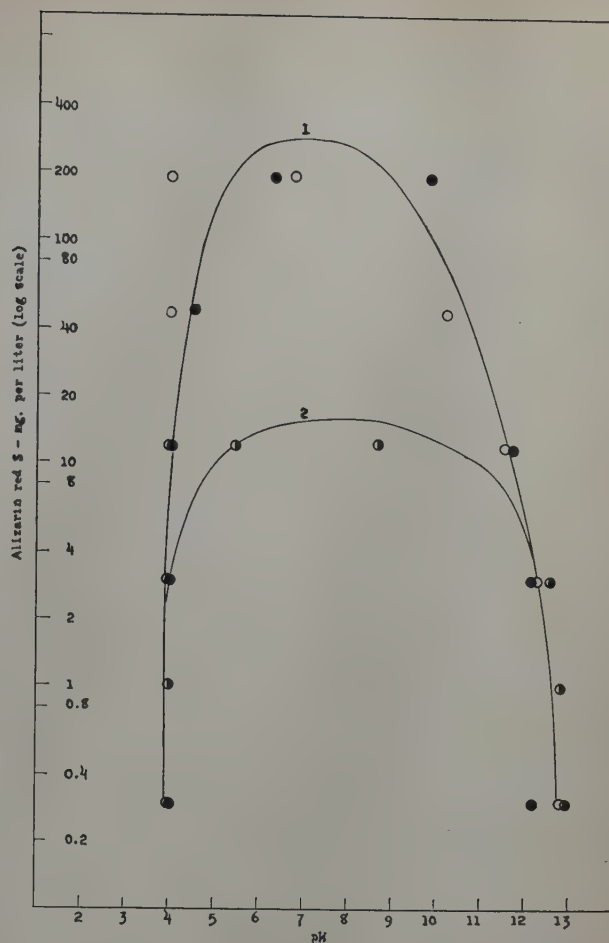
The use of barium, calcium, or lead to improve depression was reported previously, as follows:<sup>1</sup> "Added at first to aid in cassiterite activation, these ions proved more helpful in siderite depression." The observations made in the earlier work parallel





**Fig. 3a (top left)—Effect of alizarin and  $\text{BaCl}_2$  on soap flotation of artificial cassiterite.**

Flotation occurs under conditions between lower and upper critical pH curves, nonflotation outside and above. Blacked-in circles, no metal salts. Circles, 50 mg  $\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$  per liter.



**Fig. 3b (top right)—Effect of alizarin red S,  $\text{CaCl}_2$ , and  $\text{BaCl}_2$  on flotation of artificial cassiterite.**

Flotation under conditions inside curve, nonflotation outside.

Curve 1, no metal salt, and 23.5 mg  $\text{CaCl}_2$  per liter.

Curve 2, 50 mg  $\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$  per liter.

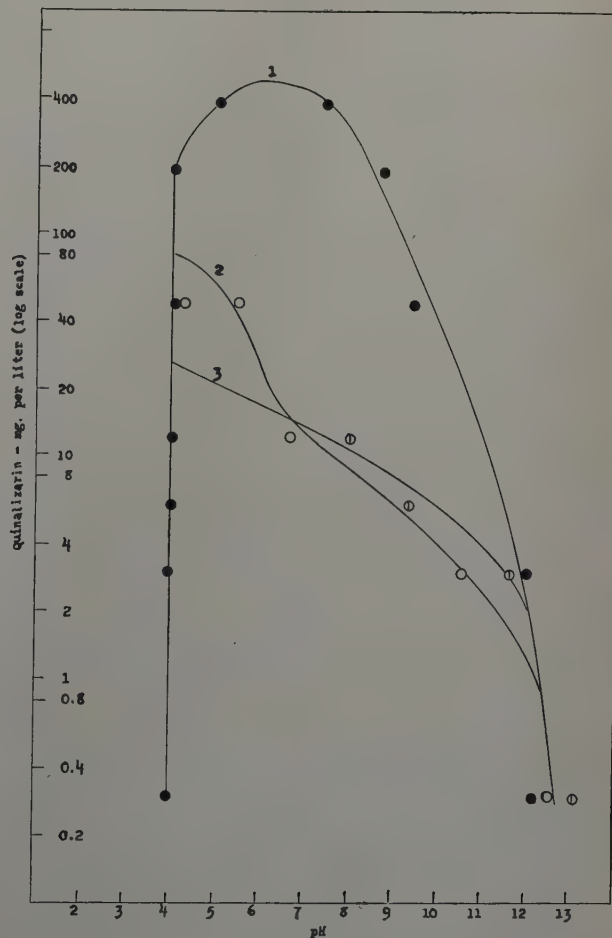
**Fig. 3c (right)—Effect of quinalizarin,  $\text{CaCl}_2$ , and  $\text{BaCl}_2$  on flotation of artificial cassiterite.**

Flotation under conditions inside curve, nonflotation outside.

Curve 1, no metal salts.

Curve 2, 23.5 mg  $\text{CaCl}_2$  per liter.

Curve 3, 50 mg  $\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$  per liter.



in some respects the "depression sensitization" phenomena discussed above, but they were made in a much more complex system. We suspect the depression-sensitization to be a fairly common occurrence.

### Fluorite Flotation

The critical-pH data for fluorite flotation with variable collector addition are plotted in fig. 4. Like unactivated cassiterite, fluorite floats over a wide pH range and wide range of oleic acid concentrations. Also, the trend of decreasing lower critical pH with increasing collector concentration is repeated for fluorite. In the case of fluorite, however, the straight line drawn through the lower, critical-pH data has

a slope of  $-1.0$  and corresponds to the relation:

$$\frac{(\text{critical } H^+ \text{ conc.})}{(\text{HOL addition})} = 13$$

when  $(H^+)$  and  $(\text{HOL})$  quantities are expressed as mols per liter.

Like Ca-activated quartz and cassiterite, fluorite shows an upper critical pH. However, at high-collector concentrations the upper pH limit decreases so that with several hundred milligrams oleic acid per liter the fluorite does not float in alkaline pulps.

The effects of alizarin, alizarin red S, and quinalizarin on the flotation of fluorite with a fixed collector addition (18 mg HOL per liter) are shown in fig. 5. The critical-pH curves are similar to those for Ca-activated cassiterite. With both Ca-activated cassiterite and fluorite, a quinalizarin addition of about 100 mg per liter was required for depression over the whole pH range, whereas several hundred milligrams of alizarin or alizarin red S were required. However, at pH values above 9, the alizarin appears as effective as quinalizarin, in that fluorite is depressed with 10 mg, or less, dye per liter. From these data, the depression of fluorite with moderate addition of alizarin or quinalizarin at pH values above 9 appears to have practical possibilities. However, these possibilities might be illusory because of the relatively high water solubility of fluorite and because Ca-activation will then make other minerals behave like fluorite.

Reviewing data for both cassiterite and fluorite,

## Effect of Alizarin Dyes on Flotation of Ba-activated Quartz

The previous work on flotation of Ba-activated quartz showed the following: (1) with oleic acid added in quantities up to stoichiometric equivalence to barium, lower critical pH values were near and just below neutrality; (2) with a large stoichiometric excess of oleic acid, much higher pH's (11 to 13) were required to obtain flotation of quartz. Thus, the principal need for using reagents to depress Ba-activated quartz will be in systems containing barium in some excess or at best only a little under stoichiometric equivalence with the oleic acid. Accordingly, the four activator-collector combinations given in table IIA were selected for the tests with alizarin dyes. With each of these combinations, the dye addition was varied stepwise from 0.3 to 800 mg per liter, and the critical pH was determined at each step.

In each series of tests the behavior on adding dye was the same: Small additions of dye had no effect on critical pH, but as the dye concentration was increased a critical dye concentration\* was reached

\* Actually a critical range of dye concentrations, as the transition was not abrupt.

above which the quartz ceased to float at all pH values. In view of the large amount of ground to be covered, large steps in dye concentration were taken between flotation tests, so that the critical ranges of dye concentration were determined only very roughly. Results are summarized in table IIB.

Table II. Effect of Dyes on Flotation of Ba-Activated Quartz

A. Activator-collector Combinations								
Series No.	BaCl <sub>2</sub> ·2H <sub>2</sub> O		Oleic Acid		Excess, Miec. per Liter		Lower Crit. pH, without Dye	
	Mg per Liter	Miec. per Liter	Mg per Liter	Miec. per Liter	Ba	HOL		
I	3	0.025	20	0.071		0.046	7.0	
II	50	0.409	120	0.425		0.016	6.9	
III	50	0.409	40	0.142	0.267		6.5	
IV	300	2.456	200	0.708	1.748		6.5	

B. Critical Ranges of Dye Concentration								
Series No.	Dye Concentrations, Mg per Liter							
	Alizarin		Alizarin Red S		Quinalizarin		Anthracene Blue	
	Float <sup>a</sup>	Nonfloat <sup>b</sup>	Float	Nonfloat	Float	Nonfloat	Float	Nonfloat
I	48	192	48	192	12	48	12	48
II	192	576	75	100	48	192	48	192
III	192	576	12	48	48	192	48	192
IV	192	576	48	192	48	192	576	768

<sup>a</sup> Quartz floats with lower critical pH not more than 1 or 2 units above critical value with no dye.

<sup>b</sup> No quartz flotation at any pH.

there are strong indications of specific depressing actions for the combinations calcium-quinalizarin, barium-quinalizarin, and barium-alizarin red S, and indications of specificity in alkaline pulps for the combination calcium-alizarin. On the other hand Ca did not act as a depression-sensitizer with alizarin red S, and Ba did not cooperate with alizarin.

Obviously these data are too exploratory to give any detailed correlation, but the following points may be noted: (1) Large additions of all four dyes depress Ba-activated quartz. (2) The dye quantities required for depression are much in excess of the requirements to form a monomolecular film on the quartz. (3) Alizarin is the least effective depressant.



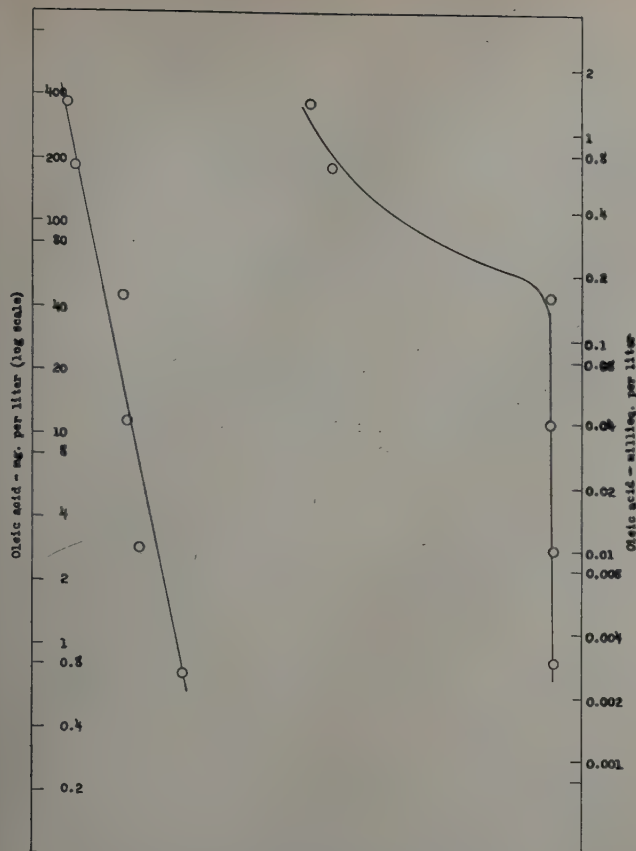


Fig. 4—Soap flotation of fluorite.

Flotation occurs between lower critical pH at left and upper critical pH at right.

This correlates with the finding that Ba did not act as a depression-sensitizer for the depression of cassiterite with alizarin. (4) Alizarin red S and quin-alizarin in concentrations on the order of 100 mg or more per liter depress barium-activated quartz. (5) Anthracene blue was an effective depressant only for the lower barium additions.

#### Selective Flotation of Cassiterite, Fluorite, and Quartz in Artificial Mixtures

One of the reasons for working with cassiterite, fluorite, and quartz was that these three minerals occur together in Colquiri tin ore and a great number of flotation tests had been made in attempting to separate them from each other.<sup>1</sup> Accordingly, as one step in bridging the space between the pure-mineral studies and the earlier ore testing, a few flotation tests were made on artificial mixtures. It was found that the deslimed cassiterite, quartz, and fluorite were readily distinguishable visually in the 100 ml graduates, so qualitative tests were made in the vacuum-flotation apparatus, following the same procedure in conditioning, reagent addition, and flotation as that used in the pure-mineral tests.

The plan followed in this part of the work was to study the pure-mineral data and find conditions of pH and reagent concentration which would give the desired separation, on the assumption that the minerals would behave independently of each other. Then a vacuum flotation was attempted under the chosen conditions. The results are only indicative and should be followed up with quantitative-flotation tests on a larger scale, in thicker pulps, with slimes present, etc.

**Cassiterite-quartz:** In the absence of activators, quartz does not float with oleic acid while cassiterite

floats over a wide pH range. In the presence of  $\text{Ba}^{++}$  and  $\text{Ca}^{++}$ , cassiterite will float in acid pulps while quartz will not. These expectations were confirmed by vacuum-floating cassiterite from quartz. Also, as already described, larger-scale tests in a Fagergren cell gave good metallurgical results in floating cassiterite from quartz in acid pulps. Evidently cassiterite did not supply activating ions to the quartz under the conditions of our tests.

Reference to data in table II and in fig. 3b and 3c indicated that in the presence of Ba and at moderately high pH values cassiterite could be depressed with small additions of alizarin red or quinalizarin, too small to depress the quartz if barium were in stoichiometric excess over oleic acid. After a few trials, quartz was vacuum-floated from cassiterite with each of the following combinations:

A	
$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$	50 mg per liter
Oleic acid	18 mg per liter
Alizarin red S	40 mg per liter
pH 10.5	
B	
$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$	50 mg per liter
Oleic acid	18 mg per liter
Quinalizarin	50 mg per liter
pH 10	

In both these systems, the barium acted as a depression-sensitizer for cassiterite and as an activator for quartz. With somewhat higher dye concentrations both minerals were depressed, and with-

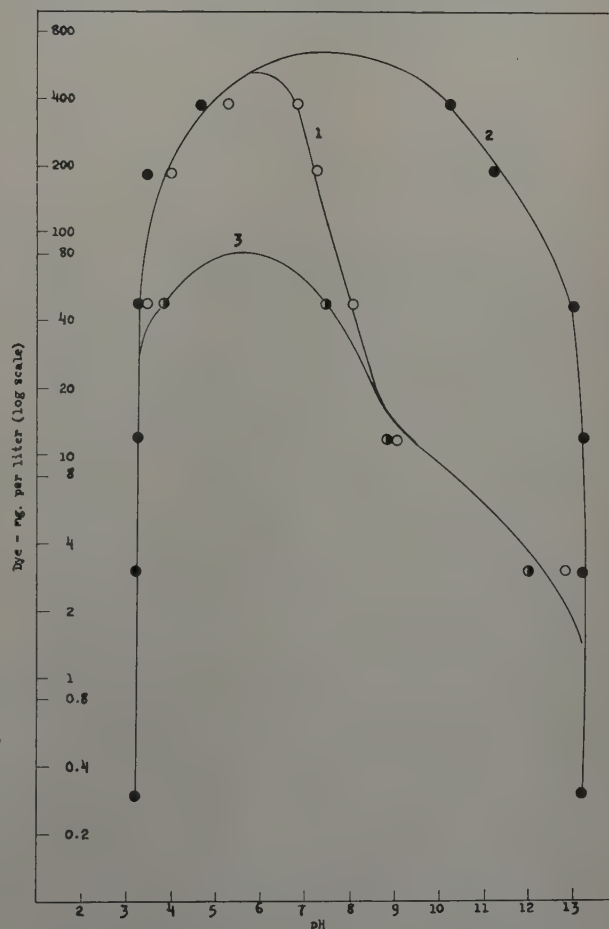


Fig. 5—Effect of alizarin dyes on flotation of fluorite.

Flotation under conditions inside curve, nonflotation outside.  
Curve 1, alizarin  
Curve 2, alizarin red S.  
Curve 3, quinalizarin.

out dye both minerals floated. Presumably, though unfortunately it was not tried, omission of barium would have resulted in flotation of cassiterite from quartz.

**Cassiterite-fluorite:** Examination of the data for cassiterite and fluorite (fig. 3b and 5) indicated that fluorite should float from cassiterite at pH 11 to 12 with 18 mg oleic acid and 100 mg alizarin red S per liter and at pH 6 to 10 with 18 mg oleic acid and 300 mg alizarin red S per liter. These separations were confirmed qualitatively.

Using alizarin red S as depressor,  $\text{BaCl}_2$  was found (fig. 3b) to act as a depression-sensitizer for cassiterite while  $\text{CaCl}_2$  did not so act. Also a high concentration of alizarin red was required to depress fluorite (fig. 5). Thus from the pure-mineral results it appeared possible to float fluorite from cassiterite using barium as a depression-sensitizer for the cassiterite. A selective float of fluorite from cassiterite was made under the following conditions:

$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$	50 mg per liter
Oleic acid	18 mg per liter
Alizarin red S	60 mg per liter
pH 9.2	

The pure-mineral data suggested no procedure for floating cassiterite from fluorite with the reagents under investigation.

**Quartz-fluorite:** Fluorite was floated away from quartz by operating at a pH too low for Ca-activation of the quartz. The test was made at pH 5.6, with 50 mg oleic acid per liter.

**Cassiterite-fluorite-quartz:** Quartz and fluorite were floated from cassiterite with the following combination:

$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$	50 mg per liter
Oleic acid	18 mg per liter
Alizarin red S	40 mg per liter
pH 11	

### Summary and Conclusions

Critical-pH determinations using a small-scale, vacuum-flotation technique were made for the soap flotation of cassiterite and fluorite. Conditions for flotation and nonflotation were determined for varying additions of oleic acid alone and for pulps containing oleic acid with various concentrations of alizarin dyes. The dyes tested were alizarin, alizarin red S, quinalizarin, and anthracene blue W.R.S., which all have the same structure but differ in the number of substituent groups. Exploratory tests were made of the flotation of cassiterite, fluorite, and quartz using oleic acid as collector, alizarin dyes as depressors, and Ba or Ca as auxiliary modifiers.

Pure cassiterite was prepared for flotation by crystallization from borax-tin oxide melts. A detailed X ray investigation showed this synthetic material to be crystallographically identical with natural cassiterite. Attempts to prepare cassiterite by reaction of  $\text{SnCl}_4(\text{g})$  and  $\text{H}_2\text{O}(\text{g})$  at high temperatures were only partly successful, and in one experiment yielded an orthorhombic variety of tin dioxide.

The main conclusions are summarized below:

1. Unactivated cassiterite and fluorite both float over wide ranges of oleic acid concentration and wide pH ranges. For a given collector concentration, flotation occurs at all pH values between a lower critical pH on the acid side of neutrality and an upper critical pH well on the alkaline side.

2. The lower critical pH decreases and the pH range of flotation widens with increase in collector concentration, for both cassiterite and fluorite. This contrasts with the behavior of activated quartz, in which increasing collector raises the lower critical pH and narrows the pH range of flotation. With moderate collector additions, cassiterite floats in pulps as acid as pH 2 or 3.

3. Cassiterite is activated by Ba and Ca in alkaline pulps, giving an increase in upper critical pH over the values obtained in the absence of these ions.

4. In concentrations of a few hundred milligrams per liter, the alizarin dyes depress cassiterite, fluorite, and quartz over the whole pH range, both in the presence and in the absence of Ba and Ca. The dye additions for complete depression are of a higher order of magnitude than the requirements to form monomolecular films on the minerals.

5. The critical dye concentrations for depression are markedly decreased in some systems by addition of a metal salt. Thus, Ba specifically aids in depressing cassiterite with alizarin red and quinalizarin while Ca aids in depressing cassiterite with quinalizarin and alizarin. These effects are chemically parallel to activation, and may be important in other systems.

6. With respect to the effects of dyes and to the magnitude of upper critical pH, Ca-activated cassiterite behaves similarly to fluorite while Ba-activated cassiterite behaves like Ba-activated quartz.

7. Further, more detailed study of the depressing actions of the various substituted alizarin dyes, alone and in combination with various metal salts, should lead both to important fundamental information on flotation chemistry and to useful improvements in the selective flotation of nonsulphide minerals.

### References

- <sup>1</sup> A. M. Gaudin, R. Schuhmann, Jr., E. G. Brown, F. Ergunalp, and R. T. Hukki: Making Tin Flotation Work. *Eng. and Min. Jnl.* **147** (Oct. 1946) 54-59; (Nov. 1946) 72-74; (Dec. 1946) 68-72; **148** (Jan. 1947) 84-87; (March 1947) 70-72.
- <sup>2</sup> R. Schuhmann, Jr. and B. Prakash: Effect of  $\text{BaCl}_2$  and Other Activators on Soap Flotation of Quartz. *Trans. AIME*, **187**; *Min. Eng.* 1950.
- <sup>3</sup> A. Daubree: Researches on the Production of Artificial Crystalline Minerals, Especially the Oxides of Tin and Titanium and of Quartz. *Compt. rend.* (1849) 29, 227.
- <sup>4</sup> G. Wunder: Stannic and Titanic Compounds Crystallized from Fluxes. *Jnl. Prakt. Chem.* (1870) **2** (2) 206; (1871) **4** (2) 340.
- <sup>5</sup> A. Knop: On Some Substances Crystallized from Microcosmic Salt and from Borax. *Ann. Ch. Pharm.* (1872) **158**, 36-58.
- <sup>6</sup> K. K. Kelley: Contributions to the Data on Theoretical Metallurgy. U. S. Bur. of Mines, *Bulls.* 371, 383, 407 and 434.
- <sup>7</sup> D. R. Stull: Vapor Pressure of Pure Substances. *Ind. and Eng. Chem.* (1947) **39**, 517-550.
- <sup>8</sup> H. St. C. Deville: The Production of Stannic Oxide and Rutile. *Compt. rend.* (1861) **53**, 161.
- <sup>9</sup> P. Groth: *Chemische Krystallographic*. Vol. 1, 84, 96 (1906) Leipzig.
- <sup>10</sup> A. Arzruni: Artificial Cassiterite. *Ztsch. Kryst. Min.* (1895) **25**, 467.
- <sup>11</sup> L. Vegard: Results of Crystal Analysis. *Phil. Mag.* (1916) **32** (6) 65; (1926) **1** (7) 1151.
- <sup>12</sup> G. Gutzeit: Chelate-forming Organic Compounds as Flotation Reagents. *Trans. AIME* (1946) **169**, 272; *Min. Tech.* (Nov. 1946) TP 2077.



# Laboratory Control in Coal Washing and Drying Plants

by James J. Merle and Richard A. Mullins

Systematic sampling and analysis in coal-washing plants results in product control and economical operation. A well-organized laboratory system reduces operating costs and increases sales if the operating and sales companies make full use of information it supplies. The operating cost of a control laboratory is less than 1 pct per ton of coal produced.

## History

THE system of laboratory control as practiced by the Ayrshire group of mines was begun with the construction and organization of the first laboratory in conjunction with the completion of its first washery in 1941. The primary purpose of this laboratory was to maintain a check on the quality of the coal produced from the washing plant. The laboratory was equipped to determine proximate analyses, float-and-sink tests, and screen analyses, and was staffed by a chemist, an assistant, and a technician.

JAMES J. MERLE and RICHARD A. MULLINS, Members AIME, are Assistant Preparation Manager and Chief Chemist, respectively, Ayrshire Collieries Corp. and Subsidiaries, Danville, Ill.

Joint AIME-ASME Fuels Conference, French Lick, October 1949.

TP 2855 F. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received Aug. 22, 1949.

Soon it became apparent (1) that product control could be accomplished, (2) that a substantial reduction in operating costs could be effected by close control of coal losses in the rejects, and (3) that information furnished by the laboratory was of continuous value to the operating and sales companies. Consequently the laboratory function was expanded to supply them with additional data as required.

As the parent company expanded and new mines were added, control laboratories were established at each mine equipped with washeries, and a central laboratory was maintained at one of the mines that was located equally convenient to all of them.

## Purpose and Use of Laboratory Control

A well-planned control laboratory assures production of a uniform, high-quality product. Whenever coal of substandard grade is loaded inadvertently, the sales company can be informed and the proper channel of distribution made. Conversely only constant and strict laboratory supervision can insure proper production of the various grades.

Knowing the product, the producer can keep out of many customers' arguments that may be both awkward and expensive. Double checking between customers and mine laboratories have solved many disputes.

Some consumers insist on buying coal on the analysis basis only, with penalties for substandard coal and premiums for coal above normal quality. A substantial percentage of the coal sold by the Ayrshire group is of this nature. The usual procedure is for the customer to sample and analyze the coal as it is unloaded at the plant, and the mine laboratory samples it as it is loaded at the mine for subsequent analysis by the central laboratory. Only the unloaded moisture is considered, rather than the average of mine and plant moisture, and the buyer sends duplicate samples to the central mine laboratory for this determination. The average of the two analyses determines the price of coal at the mine. In this particular case, the laboratory plays an important part in increasing realization for two reasons: (1) every attempt is made to maintain plant efficiency and product quality as high as possible, (2) check on analyses and sampling by the buyer or seller.

WEATHER FAIR  
TEMPERATURE 72° to 93° F

## DAILY LOADING, RECOVERY, OPERATING AND LABORATORY RECORD

RAW COAL INPUT				REMARKS			
Condition	Trucks	Tons					
TRUCKS @ 344	127	4363		RAW COAL GOOD ALL DAY			
TONS PER HOUR 628							
WASHED COAL AND BY-PASSED							
	Cars	Tons	Total per Month				
Railroad Cars Loaded	56	2954					
By-Passed to Trucks		637					
By-Passed to Heat Dryers		5					
TOTAL WASHED		3596					
Refuse		593	593				
Sludge		174	174				
TOTAL		4363					
Time Operating 6 HR. 57 MIN	Weightometer PM	33075					
Time Loss 10 MIN	Weightometer AM	28712					
TOTAL TIME 7 HR. 15 MIN	TOTAL RAW COAL	4363					
OPERATING TIME LOSS							
From	To	Min	CAUSE OF DELAYS				
8:37	9:02	5	HOPPER EMPTY				
10:10	10:08	7	POWER FAILURE				
11:50	11:53	3	CAR LOOSE NO 1 TRACK				
2:12	3:15	3	HOPPER EMPTY				
118							
WASHER ADJUSTMENT LOG							
Treating	#1-6"x0	#2-6"x0	Middlings				
Impulse Adjustments	1 2 3 4 5	1 2 3 4 5	1 2 3 4 5				
A Air Valve Quadrant	3 2 1 1/2 1/2 2	5 2 1 1/2 1 2	2 2 1 1/2 1 1				
W Water Valve Stem Extension	3 1/2 1 1/2 1 1	5 2 1 1/2 1 1	2 2 1 1/2 1 1				
E Ecc Rods from Standard	5 3 3 3 5 5	5 3 3 3 5 5	5 3 3 3 5 5				
C Partition Plate	14 15 19 19 19	15 16 19 19 19	21 20 20 20 20				
Ecc. Shaft	#8 R.R.M.	#8 R.R.M.	#8 R.R.M.				
Ejection Adjustments	Primary	Secondary	Primary	Secondary	Primary	Secondary	Middlings
D Depth Scale Setting	9 3/4	9 3/4	0 3/4	7 1/2	6		
R Float Submergence Scale, Reading WATER	19	19	19	19	19	18 1/2	
R Beam Reading Ash (Decimal)	1.52	1.25	1.53	1.25	1.42	1.32	
O Orifice Rod Extension (Inches)	7 1/2	7	7 1/2	7	3	3	
W Weir Rod Extension (Inches)	2 1/2		3		1 1/2		
Receiver Pressure	2 3/4		2 3/4		2		
Idle Water Head	16 FT		16 FT		14 FT		
Thomas Nightow Plant Foreman							
WASHER DATA		Tons	Total Moisture	% Ash, Dry	Float at 150	Sink at 150	Sink at 150
Raw Coal Input	4363	14.1	17.2	82.9	8.2	17.1	61.0
Washed Coal output	3596	14.2	8.4	98.0	8.2	1.2	27.9
Refuse	593		63.9	18	14.7	98.2	64.6
Sludge	174		39.0	30.6	18.0	62.4	53.0
Yield, Dry Coal %		82.4					
Qualitative Eff. %		98.0					
QUANTITATIVE EFF. %		99.7					
General Eff. %		92.7					
Loss of Coal in Refuse, Tons		10.7					
Dry Sludge, % Raw Coal		4.0					
Dry Solid, % Slurry Effluent		9.7					
% Solids, Recirculating Water		5.7					
PH Recirculating Water		7.6					
RHEO FINE COAL PLANT		HEAT DRYERS		OIL TREATING			
Washed Coal	Refuse	Total Moisture %		Oil Used-Gal	97		
WT % Ash %	WT % Ash %	Feed	19.4	Tons Treated	117		
Float at 1/4	94.2 75 14.1 14.5	Product	13.7	Gal. Per Tons	1.68		
Sink at 1/4	5.8 43.5 85.9 69.1	CENTRIFUGAL UNIT		Oil Stock	7541		
		Feed	53.9				
		Product	25.0				

Fig. 1—Chart of daily loading, recovery,

Relatively simple determination such as screen analyses detect wear that could prevent production of off-size coal objectionable to the customer and result in lowered realization because of higher-priced grades being mixed with lower-priced grades. Similar information on individual units in a washery quite often results in data leading to changes that will result in more economical operation.

## Economics of Laboratory Control

**Value of a Control Laboratory:** The actual assets of a laboratory may be enumerated as follows:

1. Analyses and services rendered by the laboratory to the operating company, sales company, and customers.

2. The laboratory works closely with the engineering department in determining purchase or rejection of new coal lands.

3. Increase in realization and securing maximum recovery by reducing plant losses to a minimum. The washing plants of the Ayrshire group alone reject more than one-half million tons of refuse yearly not including slurry. It has been the experience of the company that the difference in coal lost in the refuse between close control and poor control totals many tons yearly, and at the dollar value per ton of coal the profit or loss can be substantial. It is also significant to note that even a one per cent reduction or increase in coal loss in the refuse would mean much to the income of the mine. Moreover reduction in coal losses usually means that a plant must be operated with increased efficiency. Therefore the quality of the washed product is not reduced by the addition of float coal that was formerly rejected, but is actually improved because of the increased efficiency.

Close scrutiny of the daily slurry samples and their analyses will determine when it is economical to change screens. Many tons of coal are lost in the slurry because the usual practice too often is not to change screens until they are completely worn out. Often the loss of coal from one day's operation will pay for a set of expensive stainless steel screens, since an increase in opening from 1/2 to 1 mm may double the percentage of slurry, which may be an increase of 30 tons per hour in a fairly large plant.

**Operating Costs:** The actual costs of a laboratory system may be summarized as follows:

1. Control laboratories. Labor cost is less than \$0.01 per ton.

2. Central laboratory. Labor cost is \$0.003 per ton of washed coal produced. The costs of a central laboratory are prorated according to the total tonnage of washed coal produced by the Ayrshire group.

3. Costs of material used in relation to cost per ton is negligible in comparison to the labor cost.

It can be seen from the foregoing that the entire laboratory costs are more than repaid, and some laboratory expense would have been necessary under any circumstances for analyzing washed coal.

## Function of the Control Laboratory

Each of the mechanical cleaning plants is provided with a control laboratory staffed with two men: a technician and a technologist.

The duties of the technician are:

1. Sample hourly every size of washed coal produced.

2. Prepare each sample for analysis. This includes primary crushing, riffing, and pulverizing the final sample to minus 100 mesh.

3. Sample the total plant refuse every hour. Determine the percentage of coal in the refuse on composite samples both morning and afternoon by float-and-sink methods. Delatesters are used at all of the operations.

4. Collect samples of washed coal for float-and-sink and screen tests. Float-and-sink tests are determined daily on the composite samples; screen analyses determined weekly on all grades produced.

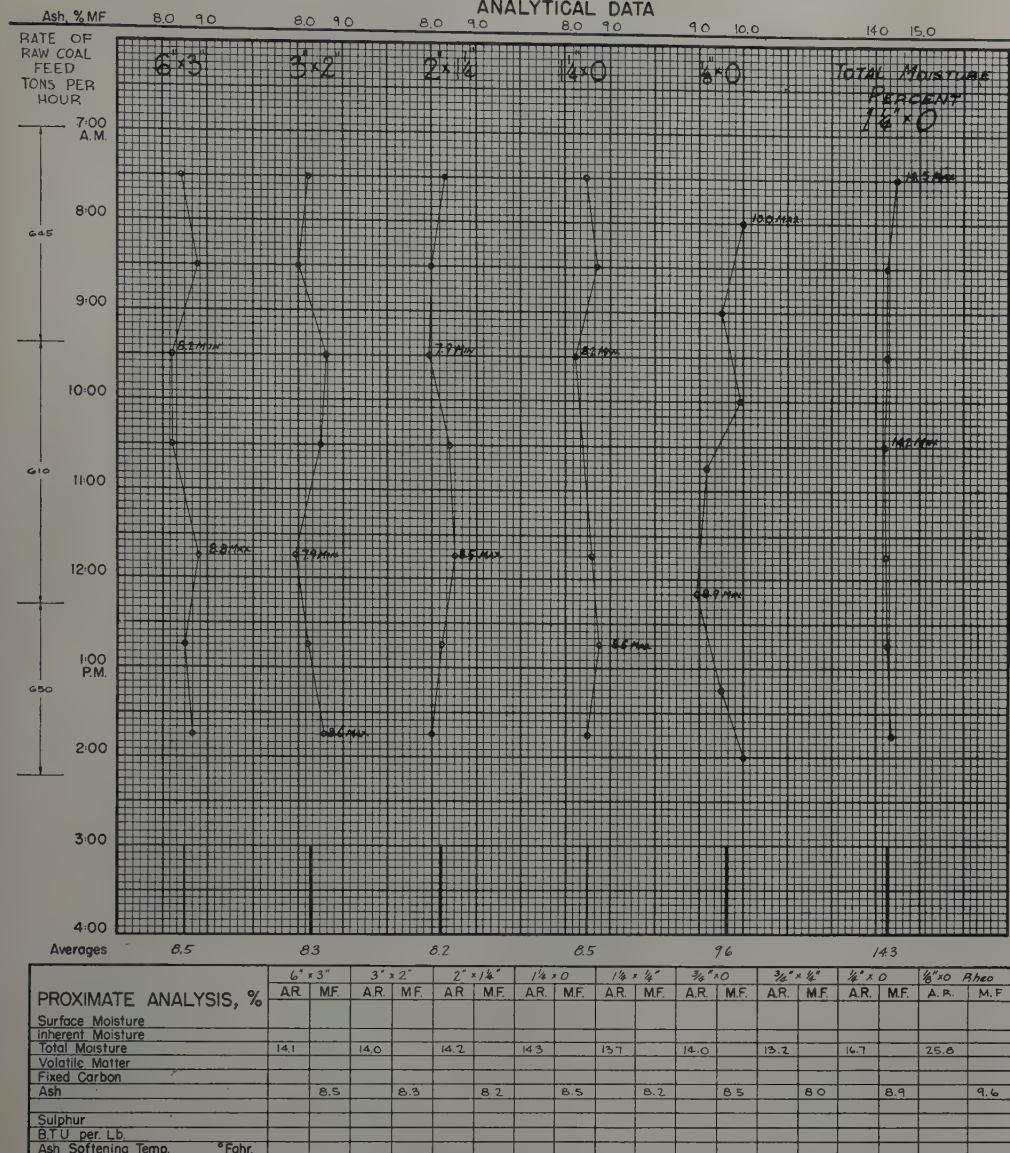
5. Sample feed and product from centrifugal and thermal driers hourly for subsequent moisture determination on each daily composite.

6. Sample slurry, recirculating water and fine coal plant product and refuse for daily determinations. Some of the operators of individual plant units aid in the collection of samples.

7. Sample individual cars when necessary to check immediately for screen consist and ash con-



ANALYTICAL DATA



operating and laboratory record.

*Delmar R. Miller*  
Technologist  
Chief Chemist

tent for coal orders requiring a product of definite specifications.

On idle days the technician aids the technologist in his work and thus in time acquires sufficient training and skill to become a technologist. All of the technologists employed by the company were trained and developed in this manner.

The duties of the technologist are:

1. To supervise and instruct the technician.

2. To determine the ash content of each of the washed coal graded, sampled, and prepared by the technician. By a modified procedure, these results are available about one hour after the sample is taken. The results are transmitted immediately by means of a Selsyn motor-generator set, with identical sending and receiving dials in the laboratory and washer floor in the plant. The laboratory and the washer floor are also connected for communication by telephone. Thus the operator can make any necessary changes in washer settings based upon information supplied by the laboratory.

3. To determine the moisture and ash on composite samples made up of portions of each of the hourly samples. These determinations are by approved ASTM methods and check closely the modified rapid method.

4. To make specific gravity analyses on the samples from the fine coal plant.

5. To prepare all reports, such as fig. 1.

6. To maintain a record of all preparation-plant maintenance, including all labor and material, that will be used to study plant operating costs and to promote changes that will increase plant efficiency.

Description of the "Daily Loading, Recovery, Operating and Laboratory Record" follows and a sample is shown in fig. 1.

1. The plant foreman is responsible for filling in the loading and operating section of the report. The raw coal tonnage is ascertained by weightometer, the washed coal tonnage is supplied by the weighmaster, and includes the number of railroad cars loaded and tonnage of washed coal bypassed to the

2. Table I gives the data from which the curves shown on fig. 1 were plotted:

### Table I. Dry Ash, Per Cent

Time	6x3 In.	3x2 In.	2x1½ In.	1½x0 In.	½x0 In. Rheo	Moisture, Pct 1½x0 In.
7:30	8.4	8.2	8.3	8.4		14.5
8:00					10.0	
8:30	8.8	8.0	8.0	8.7		14.3
9:00					9.5	
9:30	8.2	8.6	7.9	8.2		14.3
10:00					9.9	
10:30	8.2	8.5	8.4			14.2
10:45					9.2	
11:45	8.8	7.9	8.5	8.6		14.2
12:10					8.9	
12:45	8.5	8.2	8.2	8.8		14.3
1:15					9.5	
1:45	8.7	8.6	8.0	8.5		14.4
2:00					10.0	
Average	8.5	8.3	8.2	8.5	9.6	14.3

The technologist plots the results of the hourly ash and moisture determinations, tabulates the daily composite ash and moisture results, specific gravity analysis of the washed coal, refuse and fine coal plant samples, and calculates and tabulates the data shown under "Washer Data" in fig. 1.

Because it is not considered economical to sample the raw coal daily, the raw coal data are ascertained by calculation. The data are accurate since the procedure is to combine, mathematically, the known components that make up the raw coal.

Below is the procedure used in calculating the "Washer Data":

[illegible]

**Fig. 2—Chart of screen analyses**

$$\begin{aligned} \text{Yield, pct} &= \frac{\text{washed coal output, tons} \times (100 - \text{washed coal moisture}) \times 100}{\text{raw coal input, tons} \times (100 - \text{raw coal moisture})} \\ &= \frac{3,596 \times (100.0 - 14.2) \times 100}{4,363 \times (100.0 - 14.1)} = 82.4 \text{ pct} \end{aligned}$$

$$\begin{aligned} \text{Raw coal float, wt pct} &= \frac{\text{pct raw coal float in washed coal} \times \text{wt pct washed coal}}{\text{refuse} + \text{pct raw coal float in sludge}} \\ &= \frac{82.4 \times 98.8}{100} + \frac{13.6 \times 1.8}{100} + \frac{4.0 \times 30.6}{100} \\ &= 81.41 + 0.24 + 1.22 = 82.9 \text{ pct} \end{aligned}$$

$$\begin{aligned} \text{Raw coal float ash, pct} &= \frac{\text{pct raw coal float in washed coal} \times \text{washed coal float ash}}{\text{pct raw coal float}} \\ &+ \frac{\text{pct raw coal float in refuse} \times \text{refuse float ash}}{\text{pct raw coal float}} \\ &+ \frac{\text{pct raw coal float in sludge} \times \text{sludge float ash}}{\text{pct raw coal float}} \\ &= \frac{81.4 \times 8.2}{82.9} + \frac{0.24 \times 14.7}{82.9} + \frac{1.22 \times 8.0}{82.9} \\ &= 8.05 + 0.04 + 0.12 = 8.2 \text{ pct} \end{aligned}$$



ANALYTICAL DATA

Ast, % MF

		SCREEN ANALYSES							
		6" x 3"		3" x 2"		2" x 1 1/4"		1 1/4" x 0"	
Loc A.M.		Size	WT%	Size	WT%	Size	WT%	Size	WT%
		Plus 6"		Plus 3"		Plus 2"		Plus 1 1/4"	
8:00	6" x 4"	8.11		3" x 2"	96.9	2" x 1 1/4"	94.2	1 1/4" x 3/4"	84.9
	4" x 3"	66.0		2" x 1 1/4"	2.5	1 1/4" x 3/4"	3.5	3/4" x 1/4"	36.7
	3" x 2"	2.2		1 1/4" x 0"	0.6	3/4" x 0"	1.8	1/4" x 1/8"	13.2
	2" x 0"	0.1			00.0		100.0	1/8" x 0"	12.0
		100.0							60.0
9:00									
10:00									
11:00									
12:00									
1:00 P.M.									
2:00									
3:00									
4:00									

Averages

		6" x 3"		3" x 2"		2" x 1 1/4"		1 1/4" x 0"		1 1/4" x 3/4"		3/4" x 1/4"		1/4" x 1/8"		1/8" x 0"		Rho	
		AR	MF	AR	MF	AR	MF	AR	MF	AR	MF	AR	MF	AR	MF	AR	MF	AR	MF
PROXIMATE ANALYSIS, %																			
Surface Moisture																			
Inherent Moisture																			
Total Moisture		13.9		13.8		14.2		14.2		13.3		17.1		26.3					
Volatile Matter		40.1	46.5	40.4	46.9	40.6	47.3	40.5	47.2	40.7	47.2	40.4	47.1	41.1	47.4	38.8	46.8	33.4	45.2
Fixed Carbon		38.7	45.0	38.7	44.9	38.3	44.6	38.2	44.5	38.6	44.8	38.3	44.6	38.7	44.6	36.8	44.4	32.3	43.9
Ash		7.3	8.5	7.1	8.2	6.9	8.1	7.1	8.3	6.9	8.0	7.1	8.3	6.9	8.0	7.3	8.8	8.0	10.9
Sulphur		2.5	2.9	2.6	3.0	2.5	2.9	2.5	2.9	2.4	2.8	2.6	3.0	2.4	2.8	2.6	3.1	2.4	3.3
B.T.U. per Lb		11425	13270	11473	13310	11437	13350	11411	13300	11499	13340	11411	13300	11566	13340	10981	13210	9485	12870
Ash Softening Temp. ° Fahr		2220		2175		2190		2150		2170		2140		2140		2135		2160	

and dustiness index tests.

*Richard A. Mullins*  
Technologist  
Chief Chemist

Raw coal sink, wt pct = 100 — raw coal float, wt pct

or, = pct raw coal sink in washed coal + pct raw coal sink  
in refuse + pct raw coal sink in sludge

$$= \frac{82.4 \times 1.2}{100} + \frac{13.6 \times 98.2}{100} + \frac{4.0 \times 69.4}{100} = 0.99 + 13.36 + 2.77 = 17.1 \text{ pct}$$

Raw coal sink ash, pct =  $\frac{\text{pct raw coal sink in washed coal} \times \text{washed coal sink ash}}{\text{pct raw coal sink}}$

+  $\frac{\text{pct raw coal sink in refuse} \times \text{refuse sink ash}}{\text{pct raw coal sink}}$

+  $\frac{\text{pct raw coal sink in sludge} \times \text{sludge sink ash}}{\text{pct raw coal sink}}$

$$= \frac{0.99 \times 27.9}{17.1} + \frac{13.36 \times 64.8}{17.1} + \frac{2.77 \times 53.8}{17.1} = 1.62 + 50.63 + 8.75 = 61.0 \text{ pct}$$

Qualitative efficiency, pct =  $\frac{\text{pct washed coal float} - \text{pct raw coal float} \times 100}{100 - \text{pct raw coal float}}$

$$= \frac{98.8 - 82.9 \times 100}{100 - 82.9} = 93.0 \text{ pct}$$

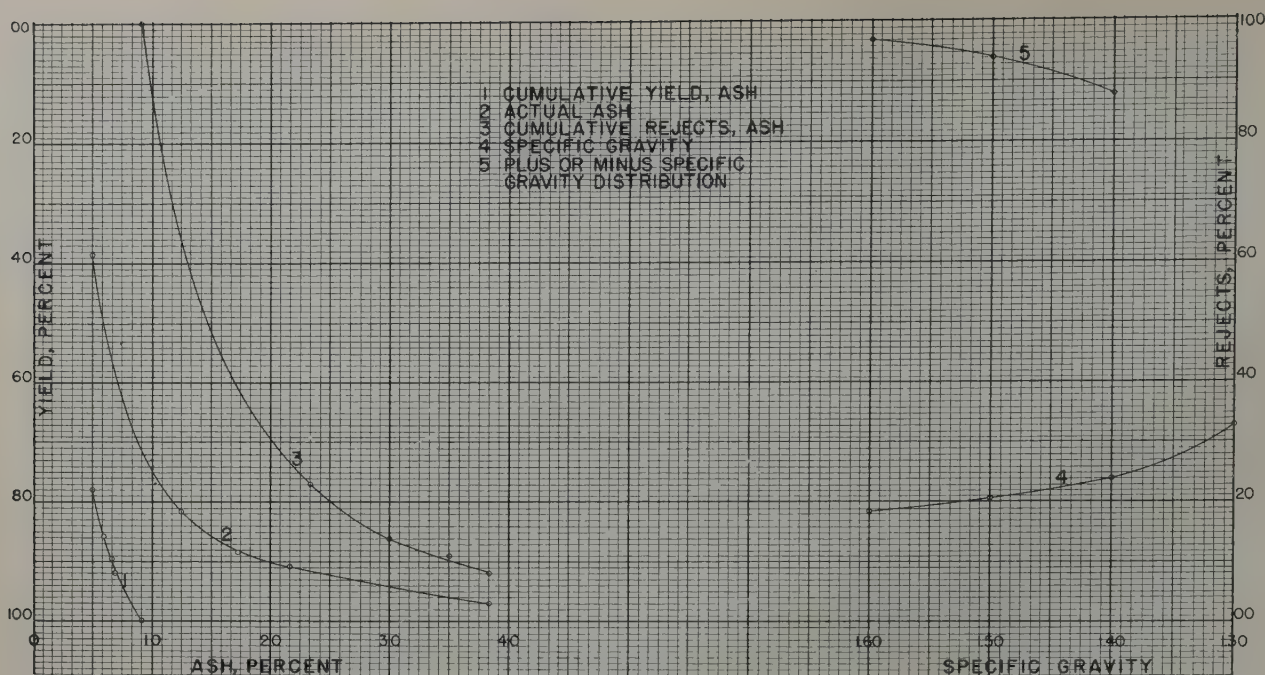


Fig. 3—Washability curves.

$$\begin{aligned} \text{Quantitative efficiency, pct} &= \frac{\text{pct raw coal float} - \frac{\text{pct refuse float} \times \text{pct refuse}}{100}}{\text{pct raw coal float}} \times 100 \\ &= \frac{82.9 - \frac{1.8 \times 13.6}{100}}{82.9} \times 100 = 99.7 \text{ pct} \\ \text{General efficiency} &= \frac{\text{qualitative efficiency} \times \text{quantitative efficiency}}{100} \\ &= \frac{93.0 \times 99.7}{100} = 92.7 \text{ pct} \end{aligned}$$

Drakeley's equations are used for calculating efficiencies. Even though other methods may be as accurate, it is felt that float-and-sink tests are a true criterion of washer efficiency since the separation is by specific gravity. Drakeley's formulas are based entirely upon specific gravity tests on refuse, washed coal, and raw coal. Some efficiency formulas depend upon both the ash content of the raw coal float and the refuse. Usually the refuse has a high ash content, from 50 to 70 pct, and contains large amounts of carbonates and iron compounds. It is difficult to duplicate determinations of ash accurately under these conditions, a fact recognized by the ASTM and consequently considerably more tolerance is allowed by the Society in the determination for ash in coals containing pyrite and carbonates. Quite often efficiencies calculated under these conditions are variable and sometimes are over 100 pct.

No efficiency formula has ever been derived that actually describes the efficiency of a plant completely, but any one of them can be of value if used continuously to study the operation of a given plant, in the routine adjustment of the plant from day to day, in the development and improvement of the plant, and in determining changes in the quality of the raw coal mined.

Data representing more than one day's results are reported weekly. Screen analyses and dustiness

index tests are reported by the control laboratory, fig. 2.

#### Function of the Central Laboratory

The central laboratory is staffed with a chief chemist, chemist, technologist, and a technician. This laboratory performs the dual function of a central laboratory for all of the mines and a control laboratory for the mine at which it is located. On idle days, the technologist and technician aid in the general work of the central laboratory.

The central laboratory coordinates the efforts of all the control laboratories and performs other work such as:

1. Proximate analyses weekly of composite samples collected at the different mines by the control laboratories, see fig. 2. In addition, the raw coal mines collect samples of the coal produced for analysis periodically by the central laboratory.
2. Ultimate analyses for all of the mines periodically.
3. Analysis of ash periodically. This consists of a mineral break-down of the ash and includes, percentages of iron oxide, alumina, lime, silica, magnesia, alkalies, sulphate, and phosphorus pentoxide.
4. Investigation of coal quality in connection with acquisition of new coal lands. An investigation of a new coal property not only consists of the usual



washability study (fig. 3), but includes proximate analyses of the raw and theoretically washed coal, a description of the physical characteristics of the coal seam, see fig. 4, analysis of the seam by sections and described by drawings, ultimate analysis of the washed coal, grindability and swelling index of the washed coal, percentage of pyritic, organic, and sulphate sulphur in the washed coal, and a burning test of the washed coal. The latter includes apparent and true density of the clinker, ash-fusion temperatures of the clinker, and ignition temperature of the coal.

5. Supervision of drinking water and acid-water treatment and control at all of the mines.

6. Water analyses for control of boiler-feed water. The central laboratory (fig. 5) is equipped and staffed to do other work such as: byproduct analyses, plasticity range, swelling and grindability indexes, some steel analyses, oil viscosity, research data of coal beneficiation processes, and investiga-

Table II. List of Equipment Used in the Central and Control Laboratories

Control Laboratory	
Coal moisture ovens, ASTM	Precision Scientific Co.
Electric muffle furnaces	Hoskins Mfg. Co.
Electric hot plate	Hoskins Mfg. Co.
Drying ovens	Fisher Scientific Co.
Analytical balance	Wm. Ainsworth & Sons
Air drying oven	Ayrshire Collieries Corp.
Raymond pulverizer	Robbins & Myers, Inc.
Dustability index apparatus	Ayrshire Collieries Corp.
Sturtevant sampling crusher	Sturtevant Mill Co.
Delatester	Reich Bros. Mfg. Co.
Hendrick test screen	Hendrick Mfg. Co.
Central Laboratory	
Water still	Barnstead Still & Sterilizing Co.
Fieldner electric furnace for volatile matter determination	Hoskins Mfg. Co.
Organic combustion furnace	Hoskins Mfg. Co.
Geiseler plastometer	Ayrshire Collieries Corp.
Fisher unitized gas analyzer	Fisher Scientific Co.
Laboratory chemicals	Elmer & Amend
Coal-ash melting point apparatus	Fisher Scientific Co.
Froth flotation apparatus	Denver Equipment Co.
Saybolt Universal Viscosimeter, ASTM	C. J. Tagliabue Mfg. Co.
Ro-Tap sieve shaker	W. S. Tyler Co.
Coal grindability apparatus	Janett Mfg. Co.
Friability test apparatus	Ayrshire Collieries Corp.
Laboratory furniture	Leonard Peterson & Co., Inc.

tion of individual units and circuits in any one plant to improve performance and reduce costs.

Conclusions

A laboratory system can be of value only if it is well organized and if the operating and sales com-

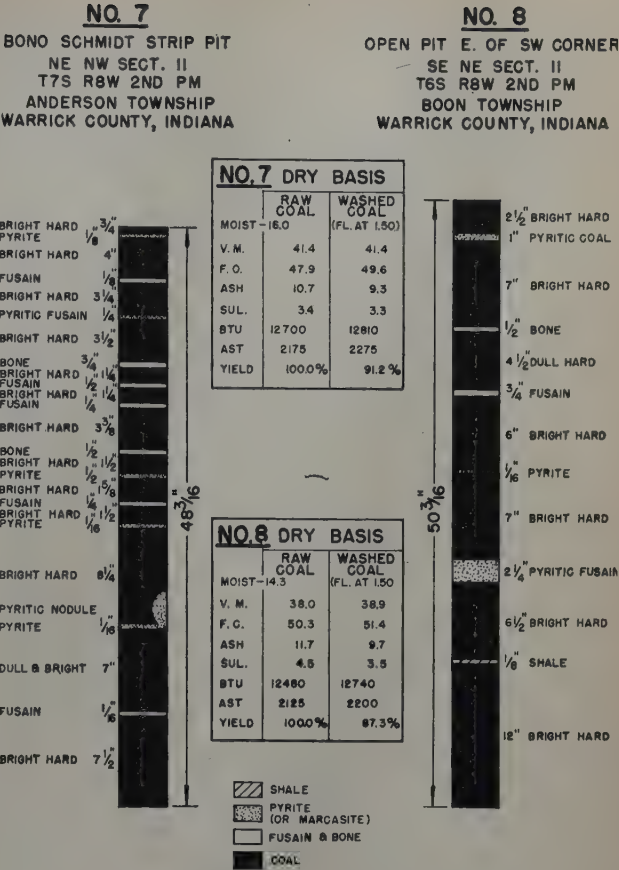


Fig. 4—Columnar sections of coal seam.

panies make full use of the information it supplies. A well-organized laboratory system more than pays for itself in many ways because it reduces operating costs and provides the seller with all the information he needs about his product in placing the coal for efficient, economic, and satisfactory service. Advertising to the consumer alone has a value that cannot be measured in dollars and cents but nevertheless is valuable because it increases the will of the customer to buy from an organization that can maintain and assure him of a uniform product.

No doubt much more useful information could be developed in a laboratory system than is done by the Ayrshire organization, but it is felt that economic considerations should limit the laboratory activities to a point at which they are not only useful, but practical as well.

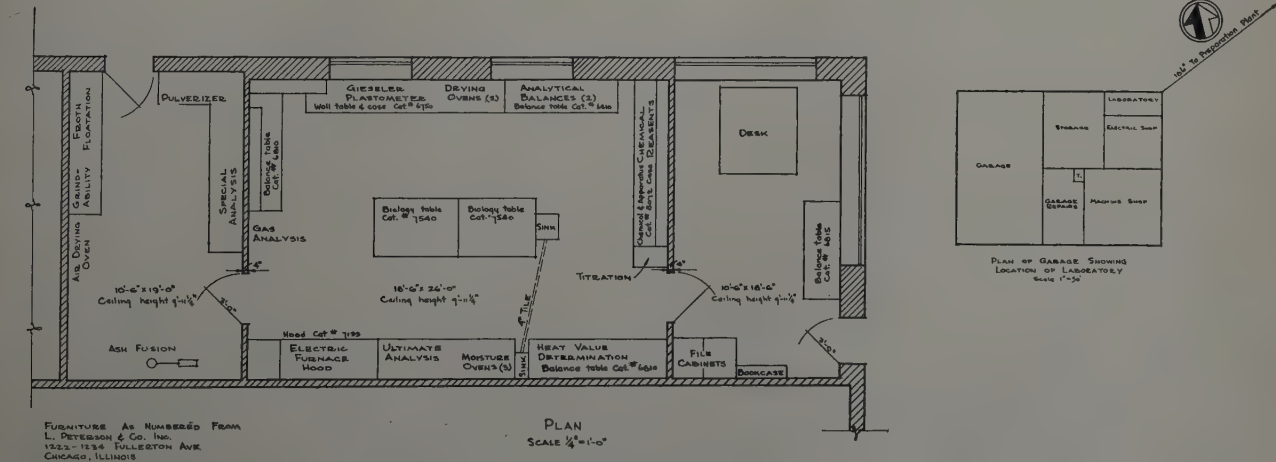


Fig. 5—Laboratory at Harmattan mine.

# The Mechanism of Coarse Coal and Mineral Froth Flotations

by

Shiou-Chuan Sun

and

R. E. Zimmerman

An evaluation of the mechanism surrounding the froth flotation of coarse coal and mineral particles as a result of experiments conducted at The Pennsylvania State College. Maximum size ranges for both bituminous and anthracite coals were established as well as the factors affecting the floating of coarse particles.

CONSIDERABLE interest is developing in the froth flotation of coal and the maximum particle sizes which can be treated by such method. The economic necessity of recovering extreme coal fines and slurries plus the restrictions being placed upon stream pollution have brought about many new and prospective installations of flotation plants.

Most coal slurries or silts have a wide size range, some sizes of which may be unsuitable for effective froth flotation. This fact also enters into the froth flotation of other minerals and the principles that

---

SHIOU-CHUAN SUN and R. E. ZIMMERMAN, Members AIME, are Assistant Professor and Division Chief, respectively, Division of Mineral Preparation, The Pennsylvania State College, State College, Pa.

Joint AIME-ASME Fuels Conference, French Lick, October 1949.

TP 2843 F. Discussion (2 copies) may be sent to Transactions AIME before June 30, 1950. Manuscript received Sept. 12, 1949.

apply to coal would also apply to minerals in general.

A series of experiments were made by the authors for the purpose of investigating factors influencing

the froth flotation of coarse mineral particles and an attempt has been made to evaluate those factors.

This paper presents the results of a study of the froth flotation of coal particles in the size range greater than 35 mesh. During the study a "multi-bubble" hypothesis has been developed to explain the various phenomena observed in the behavior of coarse particles in froth flotation of coal.

The terms coarse, intermediate, and fine particles as used in this paper are arbitrary and differ for various minerals. For coal alone the term intermediate particles means that the particle size is within the range of 35 and 200 mesh of Tyler sieve sizes. Sizes of coal particles coarser and finer than this size range are termed respectively as coarse and fine particles.

**Methods of Conducting Flotation Experiments:** All of the flotation tests, except those shown in table I, were carried out in a laboratory Fagergren flotation machine, using 300-g samples of solids for each test. All mineral samples were prepared from selected lumps of high purity by crushing, grinding, and screening to the desired sizes. Except coals, all the other samples were cleaned through magnetic



separation, tabling, washing with dilute hydrochloric acid and distilled water, and finally dried and stored in large glass jars. Before each test the dry sample was soaked 30 min in water. Tap water was used for all flotation tests. The pH values were regulated with sodium hydroxide and hydrochloric acid and measured by means of a Beckman pH meter. The rate of aeration was regulated and measured by connecting the petcock of flotation cell

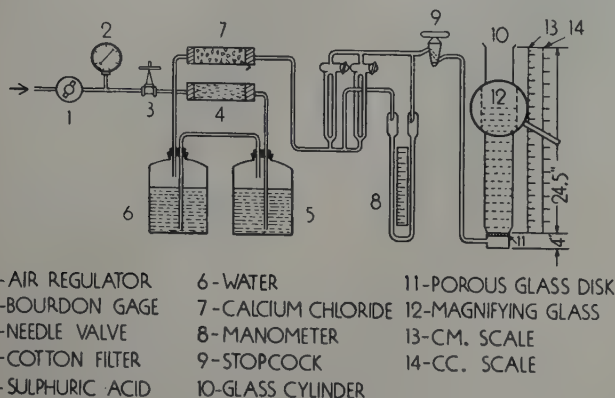


Fig. 1—Schematic diagram of a froth meter.

to one of the three different scaled rotameters which was in turn connected to the regulating system of compressed air. The rotameters were standardized by means of a water displacement method<sup>1</sup> and connected to a manometer. The magnitude of agitation in the flotation cell was indicated by the speed of impeller, which in turn was adjusted with a motor of varying speed and measured by a revolution counter.

The tests of table I were carried out in a small

Denver agitation flotation machine with a capacity of 35 g of solid feed. The froth was removed at four time intervals of 0 to 0.5, 0.5 to 1, 1 to 2, and 2 to 3 min and collected respectively into four beakers which had been standardized. The volume of the collected material was measured immediately after removal and again after the froth had disappeared. The difference in these two measurements was the volume of froth (col. 2). The collected water and coal were separated by filtration and the filtrate was measured in a graduate. Of the collected water, 90 pct was assumed as being removed mechanically from the pulp and the other 10 pct as froth water. The coal particles were dried and weighed. The number of coal particles (col. 4) were counted with the aid of a microscope. The average weight of a single coal particle (col. 5) in the samples of 3x6 mesh was determined by weighing, and that finer than this size was calculated. In the calculation of col. 8, the surface area of a single coal particle was averaged from that of its corresponding sphere and cube.

**Methods of Conducting Visual Observations with a Frothmeter:** The behavior of particles and bubbles in a flotation cell is difficult to observe because of the complexity of the system. For this reason, a frothmeter, as shown in fig. 1, was constructed and used to observe the conditions existing in an operating cell. The glass cylinder of the frothmeter, illuminated longitudinally from the back and two sides with three fluorescent light tubes, has an inside diameter of 2.2 in. and can be aerated at any rate between zero and 500 cc air flow per second.

The data of table II were established by filling the glass cylinder with 1000 cc of distilled water

Table I. A Proximate Estimation of the Number of Air Bubbles Required for the Flotation of Various Sizes of Bituminous Coal Particles Taken from the Midland Mine of Houston, Pa., in an Agitation-type Flotation Cell with 0.45 Lb per Ton Pine Oil at Constant pH 7.0 ± 0.2

Test No.	Particle Size (Mesh)	Time Period of Froth Removal (Min)	Measured Vol. of Removed Froth (Cc.)	No. of 2 Mm Diam Air Bubbles in the Removed Froth (Calculated)	No. of Particles in the Removed Froth (Counted)	Averaged Wt of One Particle (Mg)	No. of Particles Floated by One Air Bubble (Calc.)	No. of Air Bubbles Used for Floating One Particle (Calc.)	Max. No. of Air Bubbles Theoretically Packed on One Particle (Calc.)	Recovery Pct
		1	2	3	4	5	6	7	8	9
1	3x6	0-0.5	7.75	1,930	20	35.6	0.0104	96.0	47.7	2.04
2	3x6	0.5-1	12.411	3,090	5	57.0	0.0016	618.0		0.18
3	3x6	1-2	12.0	2,990	3	166.0	0.0010	995.0		1.42
4	3x6	2-3	20.0	4,980	3	166.5	0.0006	1,600.0		1.43
5	6x8	0-0.5	20.96	5,220	934	7.85	0.1785	5.6	12.6	20.90
6	6x8	0.5-1	15.94	3,980	151	9.25	0.0380	26.3		3.99
7	6x8	1-2	21.33	5,300	102	9.95	0.0192	52.0		2.89
8	6x8	2-3	35.78	8,900	29	10.1	0.0033	307.0		0.84
9	8x10	0-0.5	26.65	6,630	1,955	5.82	0.2940	3.4	6.0	31.91
10	8x10	0.5-1	29.96	7,450	1,550	5.98	0.2085	4.8		26.50
11	8x10	1-2	13.38	3,330	502	6.12	0.1515	6.6		8.75
12	8x10	2-3	23.39	5,820	298	6.35	0.0513	19.5		5.41
13	10x14	0-0.5	31.4	7,820	6,640	2.2	0.834	1.2	2.98	41.90
14	10x14	0.5-1	33.58	8,350	3,545	2.4	0.435	2.3		24.72
15	10x14	1-2	28.39	7,060	960	2.8	0.137	7.3		7.94
16	10x14	2-3	28.01	6,980	369	2.5	0.028	18.92		2.60
17	14x28	0-0.5	51.2	12,790	52,600	0.5	4.1	0.24	1.99	75.10
18	14x28	0.5-1	36.74	9,150	4,953	0.5	0.54	1.85		8.55
19	14x28	1-2	66.01	16,420	1,269	0.7	0.077	12.95		3.37
20	14x28	2-3	60.57	15,090	214	0.7	0.014	70.40		0.43
21	28x35	0-0.5	47.31	11,800	338,000	0.11	28.6	0.035	0.38	87.90
22	28x35	0.5-1	58.42	14,520	21,100	0.13	1.5	0.667		7.84
23	28x35	1-2	71.85	17,900	3,550	0.13	0.2	5.01		1.32
24	28x35	2-3	50.95	12,690	1,790	0.14	0.14	7.08		0.93
25	35x48	0-0.5	55.90	13,900	967,000	0.033	69.5	0.0144	0.19	91.20
26	35x48	0.5-1	34.59	8,600	44,700	0.036	5.2	0.1922		4.57
27	35x48	1-2	60.55	15,100	15,600	0.038	1.0	1.0		1.69
28	35x48	2-3	46.74	11,620	1,975	0.039	0.17	5.89		0.22
29	48x65	0-0.5	69.00	17,200	2,630,000	0.012	147.0	0.0068	0.095	87.5
30	48x65	0.5-1	47.77	12,380	214,000	0.013	17.6	0.0568		7.74
31	48x65	1-2	68.24	17,000	71,400	0.014	4.19	0.2390		2.87
32	48x65	2-3	59.15	14,720	47,400	0.014	3.22	0.3106		1.84
33	150x200	0-0.5	209.85	52,200	65,500,000	0.0005	1,260.0	0.0009	0.012	93.50
34	150x200	0.5-1	102.2	25,500	2,321,000	0.0006	91.2	0.0110		3.97
35	150x200	1-2	94.64	23,600	572,000	0.0006	23.21	0.0431		0.98
36	150x200	2-3	71.40	17,800	151,300	0.0007	8.51	0.1177		0.30

**Table II. Visual Observation of the Number of Air Bubbles Attached to Different Sizes of Coarse Mineral Particles in a Frothmeter Containing 1000 cc of Chemical Solution and Having a Constant Rate of Aeration of 300 cc Per Square Inch Per Min**

Minerals	Size of Particle (Tyler Mesh)	Wt of Particle (Mg)	Behavior of Particle (Observed)	Time of Particle Falling in Solution Before Starting to Float, Sec	No. of Attached Air Bubbles (Observed)	Calculated Load Carried by Each Attached Air Bubble (Mg)
(1)	(2)	(3)	(4)	(5)	(6)	(7)
Bitu. Coal <sup>a</sup>	+3	578.1	Sink		70-80 <sup>b</sup>	7.2-8.3 <sup>b</sup>
Bitu. Coal	+3	288.3	Float <sup>c</sup>	97	40-50	5.7-7.2
Bitu. Coal	3x6	98.2	Float <sup>c</sup>	32	15-20	4.9-6.5
Bitu. Coal	3x6	47.0	Float	8	6-11	4.7-5.9
Bitu. Coal	6x8	9.1	Float	3	2-3	3.0-4.5
Bitu. Coal	8x10	6.3	Float	2	1-2	3.1-6.3
Bitu. Coal	10x14	2.5	Float	1	1	2.5
Bitu. Coal	14x20	0.7	Float	0.5	1	0.7
Quartz <sup>d</sup>	6x8	31.6	Sink		4-5 <sup>b</sup>	6.3-7.9 <sup>b</sup>
Quartz	8x10	17.2	Float <sup>c</sup>	21	3-4	4.3-5.7
Quartz	10x14	7.5	Float <sup>c</sup>	7	1-2	3.6-7.3
Quartz	14x20	1.8	Float	1	1	1.8
Galena <sup>e</sup>	8x10	27.5	Sink		3-4 <sup>b</sup>	6.9-9.2 <sup>b</sup>
Galena	14x20	5.7	Float <sup>c</sup>	3	1	5.7

<sup>a</sup> Bituminous coal taken from East diamond mine, West Kentucky, tested in a solution of 105 mg per liter light oil and 125 mg per liter pine oil at constant pH 7.5  $\pm$  0.2.

<sup>b</sup> The number of bubbles attached to sink particles were observed two minutes after their settling on the bottom of the glass cylinder.

<sup>c</sup> Only a few of the many added particles were floated with difficulty.

<sup>d</sup> Pure quartz particles tested in a solution of 22 mg per liter L-Am-Hcl and 40 mg per liter pine oil at constant pH 10.6  $\pm$  0.2.

<sup>e</sup> Pure galena particles tested in a solution 165 mg per liter iso-amyl-X and 125 mg per liter pine oil at constant pH 7.5  $\pm$  0.2.

and flotation reagents. This solution was aerated with a constant rate of air flow of 300 cc per sq in. per min. The screen-sized mineral particles were soaked 30 min in distilled water, then conditioned 5 min in the same solution as that used in the glass cylinder, and finally introduced slowly a few at a time into the solution of the glass cylinder by means of a glass funnel. The behavior of the mineral particles (cols. 4, 5, 6) was determined with the aid of a magnifying glass and a stop watch. The floated and sunk particles were collected, dried, and weighed (col. 3).

A similar procedure was used for obtaining the data of table III, with the following exceptions: (1) the rate of aeration was varied for different tests; and (2) the sample used for each test was 7 g in weight and consisted of all sizes of coal particles.

### Rising of Coarse Particles Requires Attached Multi-Bubbles:

The results of the experiments described are tabulated in tables I and II. Although some of the data had to be approximated, they are sufficiently close to show certain important factors which affect the floating of coarse particles.

They show that for successful flotation operation, the number of particles capable of being carried by a single air bubble is decreased with the increase of particle size. When the size of a particle becomes too coarse, then the downward force exerted by the particle will be larger than the buoyant force of a single bubble. The result is that this coarse particle has to be floated with more than one attached bubble. The upper floatable size limit is reached when the particle becomes so coarse that it can be floated only by having its entire surface filled with a loosely

**Table III. The Behavior of Anthracite Coal Particles and Bubbles in a Frothmeter Containing 1000 cc of Distilled Water, 165 mg of Petroleum Light Oil and 125 mg of Pine Oil, at Constant pH 7.5  $\pm$  0.2**

Test No.	Rate of Aeration, Cc per in <sup>2</sup> per Min	Calculated Layers of Closely Packed 2.0 Mm Air Bubbles Layer per Sec	Wt of Largest Anthracite Coal Particle Floated, Mg.	Results of Visual Observation	
				Pulp Turbulence	Behavior of Particles and Bubbles
(1)	(2)	(3)	(4)	(5)	
1	50	0.97	3	None	Bubbles were few and widely spaced from one another. Pulp volume increased very little. Few intermediate but not coarse particles attached to bubbles. No collision.
2	100	1.94	4	Weak	
3	150	2.91	5	Weak	
4	200	3.89	13	Weak	
5	250	4.86	37	Moderate	Bubbles were moderately abundant and less widely spaced. Pulp volume slightly increased. Many intermediate but only few coarse particles attached to bubbles. Few collisions.
6	300	5.82	44	Moderate	
7	350	6.79	79	Moderate	
8	400	7.76	157	Strong	Bubbles were plentiful and closely spaced. Pulp volume moderately increased. Intermediate and coarse particles attached to bubbles. Floating particles collided gently with the wall of glass cylinder.
9	450	8.73	242	Strong	
10	500	9.70	231	Strong	
11	550	10.67	214	Strong	
12	600	11.60	115	Strong	
13	650	12.63	47	Violent <sup>a</sup>	Particles were drawn into many local vortexes to collide with one another and with the wall of glass cylinder by eddying current. Pulp volume greatly increased.
14	700	13.59	22	Violent <sup>a</sup>	
15	800	15.52	11	Violent <sup>a</sup>	
16	900	17.46	8	Violent <sup>a</sup>	

<sup>a</sup> Violent turbulence is characterized by the presence of many local vortexes and eddying currents in the pulp.



Fig. 2 — Micrograph shows the attachment of multi-bubbles on the surface of a large but very thin bituminous coal particle floated with 0.8 lb per ton petroleum light oil and 0.5 lb per ton pine oil at pH 7.3. Magnification 10X.



packed monolayer of individual bubbles as shown in fig. 2.

Mathematically, the motion of a mineral particle attached to bubble or bubbles in a homogeneous and quiescent pulp can be expressed on the basis of Newton's law of motion as:

$$F_b - F_g - F_r = ma \quad [1]$$

in which  $F_b$ ,  $F_g$ , and  $F_r$  are, respectively, the force of buoyancy, gravity, and resistance;  $m$  is the total mass of the particles and its attached air bubbles;  $a$  is the upward acceleration; and all are in CGS units. Eq 1 can be written in detail as:

$$(V_m + \Sigma V_b) D_p g - (V_m d_m + \Sigma V_b d_b) g - F_r = (V_m d_m + \Sigma V_b d_b) a \quad [1a]$$

in which  $V_m$ ,  $V_b$ , and  $\Sigma V_b$  are, respectively, the volume of one mineral particle, one attached bubble, and all bubbles attached to one mineral particle. Symbols  $d_m$ ,  $d_p$ , and  $d_b$  are, respectively, the mass density of mineral, pulp, and bubble.

It can be seen from Eq 1 that the particle is ascending when  $F_b - (F_g + F_r) > 0$ , or  $F_b > (F_g + F_r)$ . When the particle is not moving, or under equilib-

rium condition, then  $a$  and  $F_r$  are both equal to zero, and Eq 1a reduces to Eq 1b.

$$\Sigma V_b (d_p - d_b) = V_m (d_m - d_p) \quad [1b]$$

Practically,  $d_b$  is very small and may be neglected, then

$$\Sigma V_b d_p = V_m (d_m - d_p) \quad [2]$$

$$\Sigma V_b = V_m \left[ \frac{d_m}{d_p} - 1 \right] \quad [2a]$$

$$\frac{\Sigma V_b}{V_m} = \frac{d_m}{d_p} - 1 \quad [2b]$$

From the above equations it can be concluded that:

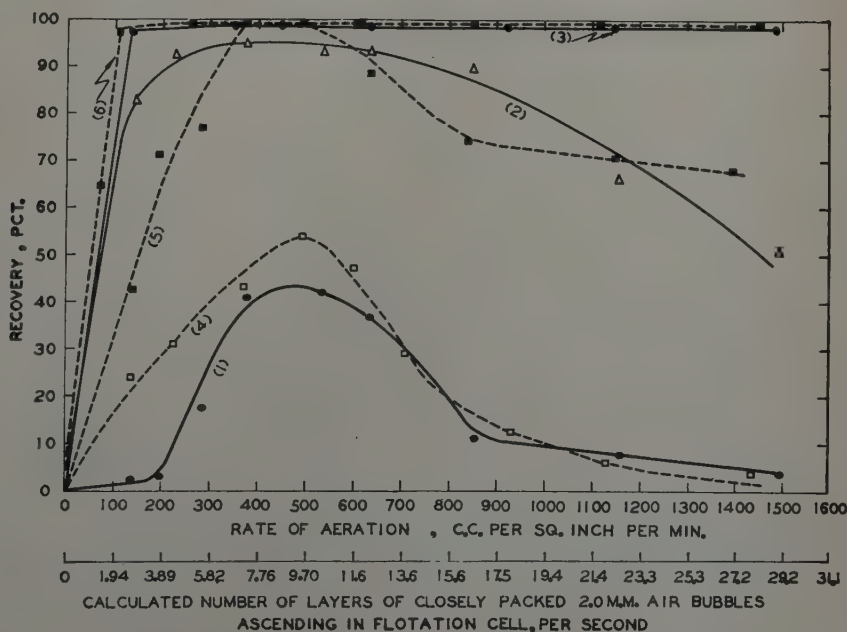
1. With a constant ratio of mineral density to pulp density, the volume or number of attached bubbles required for floating a single particle is directly proportional to the size of the particle.

2. With particle size and pulp density being kept constant, the volume or number of attached bubbles required for floating a single mineral particle is directly proportional to the density of the particle.

Fig. 3—Effect of aeration on the flotation of minerals.

Flotation time, 3 min; pulp dilution, 7.7; impeller speed 1623 rpm; pH,  $7.5 \pm 0.2$ .  
Curves 1, 2, and 3 indicate respectively 8x10, 14x20 and 48x65 mesh Pittsburgh bituminous coal floated with 0.66 lb per ton petroleum light oil and 0.4 lb per ton pine oil.

Curves 4, 5, and 6 indicate respectively 35x48, 65x100, and —200-mesh galena floated with 0.2 lb per ton iso-amyl-X and 0.36 lb per ton pine oil.



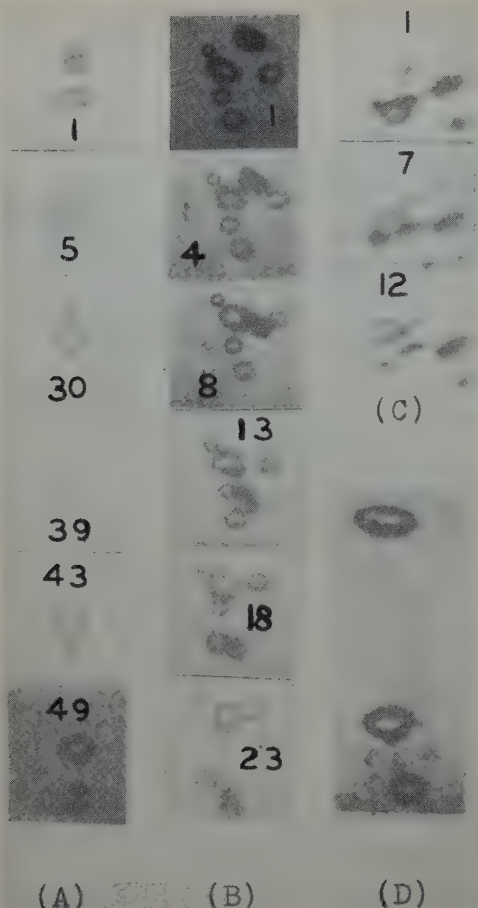


Fig. 4—High speed motion photographic pictures show the difficulty of establishing permanent attachment between coarse particle and air bubble.

3. With other factors being kept constant, the buoyant force of the particle attached to bubbles is directly proportional to pulp density.

**Multi-Bubble Hypothesis:** The satisfactory correlation of the experimental data, as presented in this paper, has been obtained by assuming that the phenomena of the flotation of coarse mineral particles can be explained by means of a proposed "multi-bubble" hypothesis. The essential ideas of this hypothesis can be grouped into three parts: (1) the rising of an extremely coarse particle at a sufficient speed<sup>3</sup> for successful flotation requires the attachment of more than one air bubble to the particle surface; (2) the attachment of many air bubbles to a coarse particle of sufficient contact angle requires a suitable rate of aeration to provide a thick net of several layers of closely spaced air bubbles in the traveling path of the particle, and an amount of agitation sufficient to keep the particle in suspension; and (3) violent pulp turbulence, usually resulting from either excessive rate of aeration or extremely high amount of agitation or both, is detrimental to the existence of multi-bubbles on a coarse particle.

The hypothesis is not entirely new. Gaudin<sup>8</sup> has calculated the necessary ratio of bubble size to particle size for successful flotation, and reported the upper size limit of floatable particles. Plante and Sutherland<sup>4</sup> have observed the attachment of more than one air bubble to a single rye grain, and de-

rived equations to express the buoyant force of the many attached bubbles. Furthermore, many flotation investigators in the past may have noted the attachment of many bubbles to a single coarse mineral particle. On the other hand, it is also a fact that this previous work covered more or less only the idea of part 1 but not that of parts 2 and 3 of the present multi-bubble hypothesis. The last two parts of the hypothesis, dealing chiefly with the influence of aeration, agitation, and pulp turbulence on the flotation of coarse particles, the authors believe to be new.

#### Effect of Aeration on Particle-bubble attachment:

By means of a frothmeter, the events of particle-bubble attachment and pulp turbulence under various rates of aeration were observed, as shown in table III. The method of conducting these observations has been previously described. Table III shows that the attachment of intermediate particles to bubbles can take place in the presence of almost any number of air bubbles and can survive the crisis of violent pulp turbulence. On the other hand, a permanent attachment of an extremely coarse particle to multi-bubbles requires the presence of not a few bubbles but a thick net of several layers of closely spaced bubbles in the traveling path of the particle. The optimum rate of aeration for the attachment of coarse particle to multi-bubbles is limited to a range of 400 to 600 cc air per sq in. per min, which is equivalent to respectively 7.76 and 11.6 ascending layers of closely packed 2.0 mm bubbles per sec. These are in agreement with the results of actual flotation tests, as shown in fig. 3.

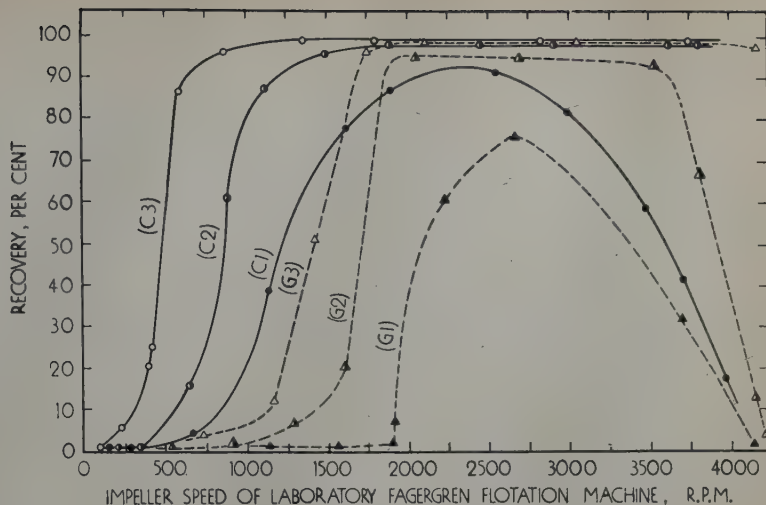
The reasons for the lack of permanent attachment of coarse particle to multi-bubbles in the presence of insufficient number of air bubbles were disclosed by means of high speed motion photographs, as shown in fig. 4. These motion photographs were obtained by means of a technique similar to that described by Spedden and Hannan, Jr.,<sup>5</sup> with the exceptions of: (1) having no microscope, (2) using an Eastman D-II high speed camera operating at approximately 1080 frames per second. Referring to fig. 4, the sequence of action photographed proceeds from top to bottom with number of frames shown. Parts A and B on fig. 4 show that the momentum of motion of an 8x10 mesh galena particle in a solution of 20 mg potassium ethyl xanthate per liter of water is probably larger than the bound force of one or two attached bubbles, so the attachment was torn apart in a split second. Part C indicates that the attachment of two 10x14 mesh coal particles to one air bubble in a solution of 25 mg petroleum light oil per liter of water is incompatible and easily broken by collision with a third particle in the pulp. Part D shows that particle-bubble attachment is increased by reducing the size of galena particles to 100x150 mesh, as indicated by the increase of thickness and size of the air bubble passing through a stream of falling particles in a solution same as that of parts A and B.

**Pulp Turbulence:** The low floatability of coarse particles but not of intermediate particles at excessive aerations and/or agitation, as shown in figs. 3 and 5, is caused chiefly by the presence of violent turbulence and to a much less extent by the lowering of pulp density. It was observed by means of the frothmeter that coarse particles attached to many bubbles are clumsy and more subject to collision



Fig. 5—Effect of agitation on the flotation of minerals in a laboratory Fagergren machine with constant aeration of 440 cc per sq in. per min and a constant flotation period of 3 min.

Curves G1, G2, and G3 indicate respectively 28x35, 48x65, and 100x200-mesh galena floated with 0.2 lb per ton iso-amyl-X, 0.36 lb per ton pine oil, at constant pH  $7.5 \pm 0.2$ . Curves C1, C2, and C3 indicate respectively 10x14, 35x48, 100x200-mesh Pittsburgh bituminous coal floated with 0.66 lb per ton petroleum light oil, 0.4 lb per ton pine oil, at constant pH  $7.5 \pm 0.2$ .



with other materials in the pulp. The frequency of collision is increased with the increase of pulp turbulence, as shown in col. 5 of table III. In a violently turbulent pulp, the surface of a coarse particle is subjected to constant rubbing by the tearing force of eddy currents and the disruptive force of other materials. The result is that the particle-bubble attachment is difficult to maintain under this circumstance.

On the other hand, bubbles mineralized with intermediate and small particles were observed to have smooth surface (see part D of fig. 4), higher mobility, firm particle-bubble attachment and elastic bubble wall. The high speed movie, similar to fig. 4 but not shown here, indicates that the collision of this kind of bubbles in solution is similar to the striking of two tennis balls in air. The collided bubbles are usually bounced away and occasionally coalesced into one larger bubble, but seldom broken. Consequently, the floatability of intermediate and small particles is less affected by pulp turbulence.

In regard to the small influence exerted by the lowering of pulp density, col. 5 of table III indicates that at excessive rate of aeration the volume of air bubbles entrapped in the pulp is greatly increased, as indicated by the increase of pulp volume, and the pulp density is lowered. To a lesser extent, pulp

density should also be lowered at excessive agitation, as caused by the sucking of more air into the flotation pulp. According to the equations given above, the lowering of pulp density reduces the buoyant force. This may have an ill effect on the floatability of coarse particles but not of small particles.

**Agitation and Mineral Density:** With respect to the effect of agitation on the flotation of coarse particles, fig. 5 shows that the minimum magnitude of agitation required for a successful flotation operation is increased with the increase of particle size and mineral density. This can be explained by the observed fact that the suspension of mineral particles in liquid is a prerequisite of particle-bubble attachment. The amount of agitation used for keeping the coarse and heavier particles in suspension has to be larger than that of the smaller and lighter particles. The declining of floatability of coarse particles at an extraordinarily large amount of agitation is caused by the increase of pulp turbulence, as discussed previously.

With respect to mineral density, fig. 6 shows that the upper floatable size limit of mineral particles under the most optimum flotation conditions is decreased with the increase of mineral density. This

Fig. 6—Maximum particle size of different minerals floated under the most optimum conditions of flotation in a laboratory Fagergren machine with constant aeration of 426 cc per sq in. per min and constant impeller speed of 2193 rpm.

Curve 1: Bituminous coal (1.35) taken from Midland mine, Houston, Pa., and floated with 1.2 lb per ton petroleum light oil, 0.8 lb per ton pine oil, at constant pH  $7.0 \pm 0.2$ .

Curve 2: Philadelphia and Reading anthracite coal (1.54) with 2.04 lb per ton Squibb mineral oil, 0.864 lb per ton pine oil at constant pH 7.2.

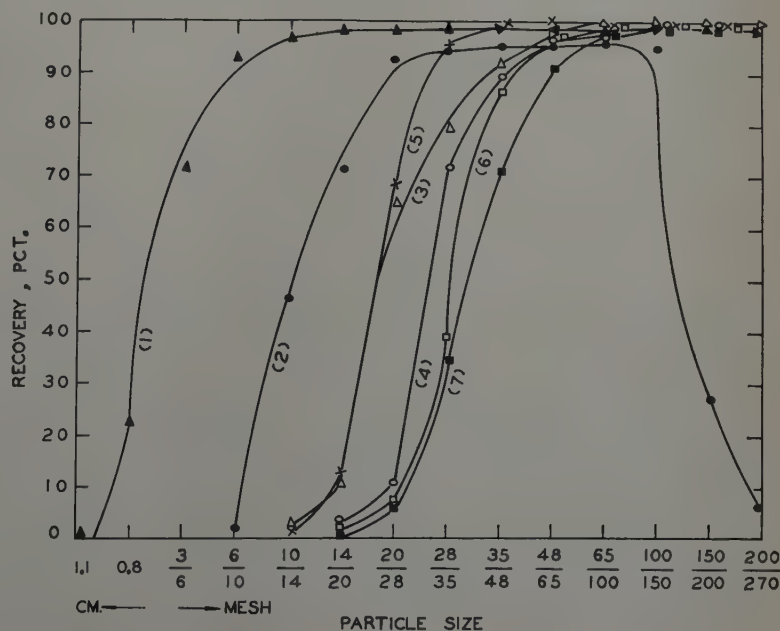
Curve 3: Quartz (2.65) with 0.33 lb per ton L-am-Hcl, 0.6 lb per ton pine oil at constant pH  $10.6 \pm 0.2$ .

Curve 4: Fluorite (3.1) with 0.27 lb per ton  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$ , 0.53 lb per ton Na-oleate, 0.6 lb per ton pine oil, at constant pH 8.8.

Curve 5: Sphalerite (4.0) with 0.11 lb per ton  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$ , 0.33 lb per ton iso-am-X, 0.6 lb per ton pine oil, at constant pH 6.5.

Curve 6: Pyrite (5.0) with same reagents as curve 5 at constant pH 4.5.

Curve 7: Galena (7.5) with same reagents as curve 5 at constant pH 7.5.



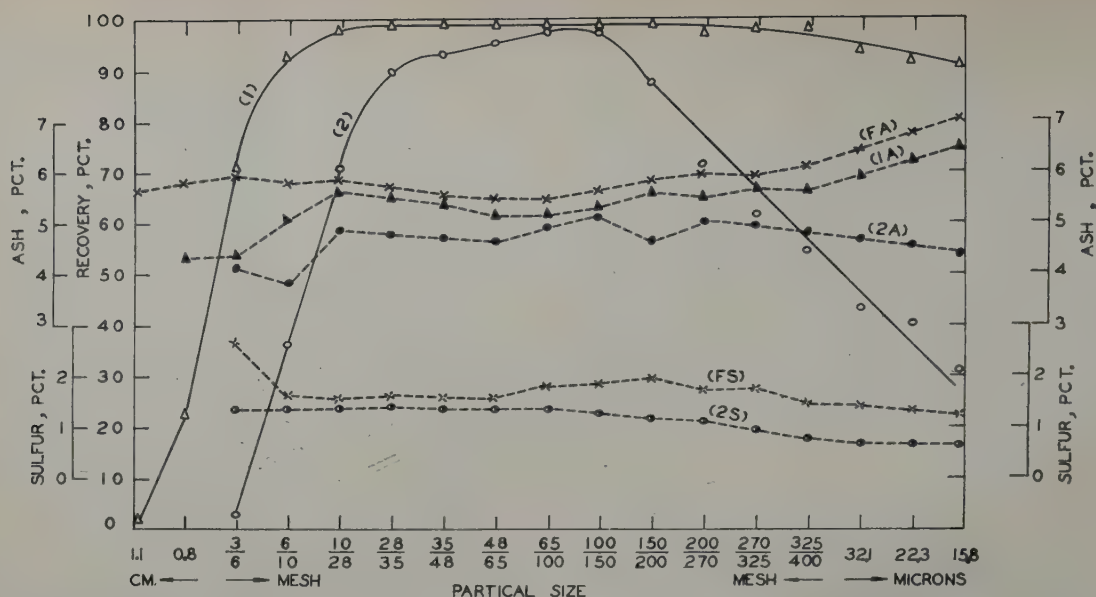


Fig. 7—Effect of flotation reagents on upper floatable particle size of bituminous coal taken from Midland mine, Houston, Pa., and floated at constant pulp dilution 7.7 and pH  $7.2 \pm 0.2$ .

Curve 1: Floated with 1.2 lb per ton petroleum light oil and 0.8 lb per ton pine oil. Curve 2: Floated with 0.4 lb per ton pine oil. Curves FA, 1A, and 2A: Ash per cent of feed, concentrate 1, and concentrate 2, respectively. Curves FS and 2S: Sulphur per cent of feed and concentrate 2, respectively.

apparently is caused by the fact that when two particles have the same weight the one with higher density must have a smaller specific surface area<sup>2</sup> than the other with a lower density. The number of air bubbles that can be attached to coarse particles is directly proportional to the specific surface area of the particle. From fig. 6, it can be seen that the maximum size of bituminous and anthracite coal particles which can be floated practically is respectively 3x6 and 14x20 mesh.

**Flotation Reagents and Deep Scraping:** In regard to flotation reagents, fig. 7 shows that within a certain limit the coarseness of the floatable particles is increased with the increase of amount or grade of the flotation reagents used. This is caused by the improvement of both the contact angle of particles and the number and quality of air bubbles. The result is that the attachment of multi-bubbles to coarse particles increases.

The floated coarse particles were observed to be held in most cases in the interface between the pulp surface and the froth instead of in the body of the froth. This is probably due to the coarseness of the particles and the low density of the froth body. Consequently deep scraping of the froth is needed to remove the floated coarse particles from a flotation cell.

### Summary

1. The maximum size of bituminous and anthracite coal particles which can be efficiently floated in a laboratory Fagergren flotation machine is found to be respectively 3x6 and 14x20 mesh.

2. The factors influencing the froth flotation of coarse mineral particles are aeration, agitation, pulp turbulence, flotation reagents, and mineral density. The phenomena of flotation of coarse particles can be interpreted by means of a proposed multi-bubble hypothesis.

3. The optimum rate of aeration for the flotation of coarse particles in a laboratory Fagergren flotation machine is in the vicinity 450 cc air per sq in. per min.

4. The minimum amount of agitation required for the flotation of coarse particles must be large enough to keep the particles in suspension, and is increased with the increase of mineral density and particle size.

5. Violent pulp turbulence, usually resulting from excessive aeration or agitation or both, is detrimental to the successful flotation of coarse particles.

6. Other factors being constant, the coarseness of floatable particles, to a certain limit, can be increased with the improvement of flotation reagents.

7. The upper floatable size limit of mineral particles under the most optimum conditions of flotation is reduced with the increase of mineral density.

8. Deep scraping is needed to remove the floated coarse particles from a flotation cell.

### Acknowledgment

The writers wish to express their appreciation to Professor D. R. Mitchell for his heartening encouragement and helpful criticism. Thanks are due Mr. Charles W. Young and Mrs. Barbara Weigand for their cooperation in the work of high speed motion photography. Also appreciation is expressed to Dr. T. C. Tsu for his kind assistance in deriving the equations and to Messrs. H. T. Darby, C. D. Neubling, and H. Lovell for their work in chemical analysis.

### References

- <sup>1</sup>A. W. Fahrenwald: The Aerating Capacity of Flotation Machines. Pamphlet 64. Univ. of Idaho (July, 1943).
- <sup>2</sup>A. W. Fahrenwald: Size of Mineral Particles in Relation to Flotation Concentration. Pamphlet 2. Univ. of Idaho (1931).
- <sup>3</sup>A. M. Gaudin: Flotation. Pp. 106, 152. (1932) New York. McGraw-Hill Book Co.
- <sup>4</sup>E. C. Plante and K. L. Sutherland: The Physical Chemistry of Flotation X, The Separation of Ergot from Rye. *Jnl. Phys. Chem.* (1944) **48**, 203.
- <sup>5</sup>H. P. Spedden and W. S. Hannan, Jr.: Attachment of Mineral Particles to Air Bubbles in Flotation. *Trans. AIME* (1949) **183**, 208; *Min. Tech.* (March 1948) TP 2354.



# Development Work with Trackless Equipment

by Elmer A. Jones

Development work in mines of St. Joseph Lead Co., Southeast Missouri, using trackless loading equipment shows definite advantages: Speed of cleaning, ability to work on steep grades and sharp crosscuts, good later tracking conditions, and possibility of wasting rock in old abandoned stopes.

**T**HE Desloge mine is one of a group of mines owned by the St. Joseph Lead Co., in the so-called Lead Belt of Southeast Missouri. It has been operating approximately 40 years producing a daily tonnage ranging from 1600 to 3800, depending on various factors. On an average, the present mine is 300 ft deep with an extreme extent of 2.7 miles by 1.7 miles.

Most of the present mining is in the lower horizons of the Bonne Terre dolomite, a fairly soft rock requiring no support except occasional channel iron and roof pins. Due to the contour of the underlying sandstone and porphyry, some development work has been through short stretches of these different formations. In some areas the lower part of the dolomite is quite sandy and generally is considered abrasive.

**Sizes and Methods of Drifting Have Changed:** Development work has, like other phases of mining, changed considerably in the last 20 years, from cleaning by hand to the use of a mechanical method. The conception of a standard drift has also changed. Before the advent of mechanical loading in stopes there were many pieces of main line drift averaging 7 ft high x 9 ft wide. With the coming of the St. Joe shovel the drift was increased to 8 ft high x 10 ft

wide. Now with the Joy loader and shuttle car, a 9x11 ft drift is coming to be standard size.

Methods of cutting drifts have changed considerably over the last few years. Some changes in drilling equipment have been made such as reducing the weight of column, clamp, air feed, and fittings to go with the HC10 Cleveland jackhammer. Other changes have been made in drill steel such as the use of detachable bits; throw-away variety, screw type, and the tungsten-carbide insert bits. Later developments have introduced spiral-welded pipe with victaulic couplings, millisecond exploders, bulky powder, and the use of electric cap lamps. Experimental work is going on continually in various phases of development work, some to be tried and discarded, others gradually to become standard practice.

To sum up, the standard method of development now in use at Desloge is as follows: The work is done on a two-shift cycle 7:00 p.m. to 3:00 a.m. and 7:00 a.m. to 3:00 p.m. Two men known as development drillers comprise a crew on each shift. These two men arrive at their work about 7:30 to find a round blasted by the previous shift. After scaling down back and walls they proceed to the cleaning operation, using either a Joy loader and shuttle car, a drag hoist mounted on a slide ramp, or an Eimco Finlay to load into a 2½ ton mine car. Cleaning may take from 2 to 4 hr depending on conditions such as length of round broken, length of haul, and other variables. After fitting up pipe and setting up columns and drills they are ready to drill a round which should measure 9 ft high x 11 ft wide and break from 5 to 7 ft advance. This may also take from 2 to 3 hr. Then they are ready to charge and

ELMER A. JONES, Member AIME, is Assistant General Mine Superintendent, St. Joseph Lead Co., Bonne Terre, Mo.

AIME Columbus Meeting, September 1949.

TP 2859 A. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Sept. 16, 1949.

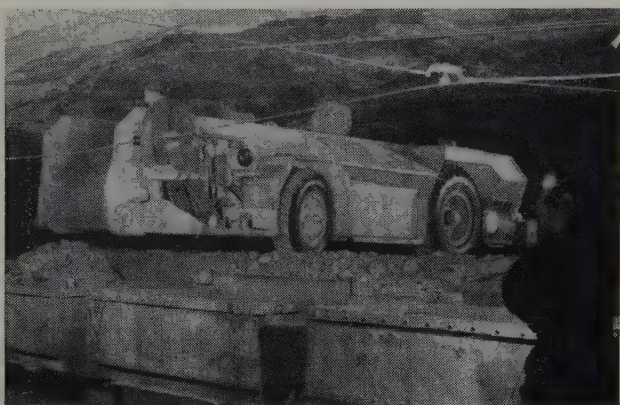
blast using 1x8 in. Gelamite Nos. 1 or 2 dynamite depending on the character of the rock. Failures to complete a round are not uncommon because of mechanical trouble, water, and various other reasons, but generally these crews will finish their swing.

**Trackless Equipment Comes into the Picture:** On January 1, 1948, the management made quite a momentous decision as far as Desloge was concerned. This was to shut down the mine as an ore producer and to go all out on development and remodeling. This program was designed to put development work a year ahead of mining and also to enable the ore to be hauled to Leadwood or Federal mill instead of Desloge.

This decision, increasing development work by about 100 pct over our wartime rate, called for more development men, more equipment, and more constant use of available machines. It was therefore necessary to split up old time development crews,

it was necessary to plan future development to accommodate these units. In other words we should enter an ore body from the side rather than the center and have our discharge point for shuttle cars about 4 ft above haulage way. One also should bear in mind that concentrating effort in a certain area would result in less time spent in long moves, closer supervision, maintenance, and so on.

By January 1, 1948, it was planned to cut 13,000 ft of drift during the current year. Thirteen development crews were organized, five of them on a three-shift basis, eventually to have Joy loaders and shuttle cars as regular equipment. The first loader and shuttle car, having been in operation, was put into a drift immediately using our men, who were experienced to some extent. The next four loaders and shuttle cars, called 18 HR and 60 E respectively, were delivered, a unit at a time, until by May 15, we were cutting five drifts using loaders and shuttle cars, on a three-shift basis.



**Fig. 1—Shuttle car dumping into mine car.**

taking on former operating drillers, and to plan on working as many crews as possible on a three-shift basis. Fortunately St. Joe had twelve Joy loaders and shuttle cars on order and Desloge mine was to get a total of five units. It was decided that they would be tried in drift work.

#### **General Description Joy Loader and Shuttle Car:**

The Joy 18 HR is a continuous type loader with gathering arms in front head to throw the rock onto a universal chain conveyor (fig. 1). The material is discharged by moving the conveyor up or down and by swinging it 45° each way from center line to facilitate loading of shuttle car or mine car. The loader has a rated capacity of 5 tons per min with a maximum capacity of 10 tons per min. It is 66 in. high, 6 ft 4 in. wide, 25 ft long, and gathering arms will clean 5 ft 9 in. wide. The machine weighs about 15 tons, and is caterpillar mounted. The main driving motor is 75 hp. Two 7½ hp motors are mounted on conveyor frame to pull the conveyor. In addition there is a 5 hp hydraulic pump motor mounted in the rear to actuate swing of conveyor, raising and lowering head, and other motions.

The 60 E shuttle car weighs 12 tons and will carry about 13 tons of lead ore. It has a conveyor bottom for rapid discharge of load. The two 10 hp dc traction motors of the car can be powered by means of cable reel or trolley. The conveyor is operated by a separate motor of 10 hp. The car is 60 in. high and 25 ft long with a wheel base of 9 ft. The traveling speed is 6 mph empty and 4½ mph when loaded.

When it was known that Desloge would have Joy loaders and shuttle cars for future operating units,

**Problems Presented Give Good Experience:** The problem of training our drillers to use this equipment was readily apparent. Service representatives of the Joy Manufacturing Co. were available and were helpful in starting the training. The men on evening and night shift were, by force of circumstance, their own teachers, but found it easy to break in on the machines. In a few days each crew was doing a fairly good job of cleaning and hauling rock. Maintenance was a real problem as our repair men also had to learn the hard way. Some of the drifts being cut were 1 pct down-grade and as a result there was some water with which to contend. As long as the rock was hard and dolomitic the loading was fairly easy. In one drift a sugary sand was encountered. This sand disintegrated badly, and, mixed with water, was practically impossible to load as it never got back to the end of the loader. Here we had to substitute an Eimco Finlay loader and move the Joy to another drift.

Each drift had slightly different conditions. In the first one where a shuttle car was used, the waste rock was disposed of in an old worked out stope. Our first plan was to discharge the shuttle car on an up-grade making a higher and higher fill. We did not get far with this idea as the car would sink into the loose muck on the upper end of the fill and the flights going under the car would stick in this loose material. The remedy for this was to make a rock ramp on about a 10 pct grade ending at a log wall so that the rock could be piled up and later moved by a drag hoist and dipper to the desired spot. By the time this difficulty was overcome the develop-



ment crews had become so proficient in the use of the equipment that they were saving about an hour on their cleaning time and thus were able to do the additional work of spreading the waste rock on their shift. One immediate advantage of using the equipment was soon seen in this drift. In working three shifts the problem of ventilation was important. By saving time on the cleaning it was possible for each shift to have their round complete in about 6½ hr, giving 1½ hr for the smoke to clear before the next shift.

Another drift was started at an angle of about 60° from an old main line. The first 100 ft of this was cut using a hoist mounted on a slide ramp loading into mine cars. When the second loader and shuttle car were delivered the old loading equipment was taken out and replaced with the new machines. First we cut back up hill to the main line from a point about 30 ft in the drift and made a loading ramp about 4 ft above the level of the track. Due to a lack of sufficient small locomotives we had ordered Joy car pullers. One of these was installed on a bluff near the ramp and by using sheaves and moving the starting equipment to the ramp we were able to spot the cars under the end of the shuttle car without using more men. This crew then disposed of their waste rock directly into two ton mine cars and were dependent on main line car service.

A third different condition arose when it was necessary to cut a drift out of a stope on an upper level. A raise connected the two levels and it was possible to dump waste rock or ore into this raise from the shuttle car. This meant using a chute man to pull the rock out of the raise into mine cars and gave the development crews an opportunity to get rid of their rock.

After the first crew completed the drift we had another job that presented several new difficulties. A new lower level main line was planned to open up new ore bodies and provide haulage to Baker mine. It was to be 2400 ft long with several raises and crosscut drifts from the main line and about 40 ft lower than the old haulage level. It was possible to attack this job from three different locations. It also involved moving the loader, shuttle car, and drag hoist through about 3000 ft of old abandoned drifts, open stopes, and down an incline. This was accomplished by building a 440 ac power line as the loader moved ahead and installing 275 v dc feeder and return line as the shuttle car progressed. In some parts of the road there were small rock piles resulting from roof falls that had to be loaded and disposed of to clear the passage for the loader. The last difficult part of the trip was down an old bluff about 25 ft high. This had to be shot to decrease the slope from about 45° to about 30°. When the move was complete a drift from an old stope was started at right angles to the new main line location. This was only about 30 ft and then right angle turns were made to the left and right to start actual work on the new main line. This rock was disposed of in the old stope by using an Ingersoll-Rand 235 hoist pinned to solid, which piled the rock high in all directions. Very shortly after the drift to the left was started, porphyry was encountered. This was much harder than our ordinary dolomite and tungsten-carbide insert bits were used with good results. The condition continued for about 400 ft with no noticeable change in the efficiency of the drifting operation. By the time the drift broke through into part two of this haulage project the haul to dispose of waste rock was 950 ft. Each round

was producing about 50 tons of rock which meant that the shuttle car had to make four trips to complete the cleaning job. The longest haul in the three sections of this total drift was 1250 ft.

**Length of Haul:** Since the use of the Joy loader and shuttle car in our development operations, about 15,000 ft of drift has been cut at the Desloge mine. Experience has been varied in determining economic hauling distances with one car. Several times the haul has exceeded 1000 ft. On two occasions, two trolley poles have been mounted on the shuttle cars, trolleys hung in the drift, and very satisfactory hauling results obtained. As the development crews become skilled it appears that a two-man drift crew can be expected to do their regular work in an 8-hr shift up to a distance of 1400 ft. In this maximum distance trolley wire should be installed to speed up the last 500 ft of the job. If the drift were to be longer than 1400 ft, we would stop and build track to a new discharge ramp located a safe distance back from the face. We have successfully done this job and are now well on the way toward completion of a drift 2000 ft long.

**Track Work:** Previously at Desloge and at present in other mines of the St. Joseph Lead Co., several different methods of tracking behind development crews were and are employed. Each method used required that trackmen be used at regular intervals. At Desloge there were four long main lines of development cut completely before any track was laid. Tracking here with 60 and 70 lb rails was done with the greatest efficiency we have experienced. Practically no digging was required for ties, perfect alignment was easily obtained, and exceptional speed was made in laying a finished track.

Ballast was obtained from digging out the necessary water ditches and cleaning up available loose rock. In the first drift we found that it was cleaned too well by the development crew and it was necessary to haul back into the drift sufficient fine rock to ballast the track completely. It was our experience in building main line through these clean drifts that the complete job of tracking was accomplished by one half the man shifts that would have been used tracking behind our old style, drift-cleaning equipment. The net result was also a much straighter and cleaner main line than could have been expected before.

**General Experience:** During the year and one half these machines have been in development including an interruption by a strike lasting 11 weeks, 15,000 ft of drift have been driven. Valuable experience has been gained both in discovering what the machines can do and what should be done to keep the machines operating.

Most of the main line work was on grades of 1 pct, both up and down. Occasional crosscuts were driven from the main lines, generally at sharp angles and on grades up to 10 pct. These grades and sharp curves were accomplished with no trouble either to the loader or car. In trying to load on one short crosscut that was 18 pct up, it was necessary to tie the car to the loader, a practice which is not recommended as feasible. On another occasion, a ramp to discharge rock into a pile was built on a 15 pct grade for a length of about 80 ft with a flattened out section at the end. This job was successfully carried out with trolley wires running up this grade. Turns in crosscuts as sharp as 25 ft radius were also made without difficulty. The only serious trouble encountered due to physical conditions was in trying to



load disintegrated sand in water which is virtually impossible to accomplish.

Another advantage soon evident was in speed of cleaning. In our method of drifting, a center cut is used which throws rock back down the drift as far as 300 ft. Other machines work very slowly in thin fly rock but these loaders and shuttle cars are very fast in this kind of loading. It is possible to bulldoze long distances in thin rock and, when ready to load, the shuttle car can be made available. Superficial time studies and comparisons indicate a saving of about an hour on an average swing of drift. As soon as this was evident most of the crews immediately started drilling a longer swing of drift. A 7-ft average for each round over a week's work was not un-

larly with a long train. It also requires an extra man on the crew to handle the locomotive. By using a car puller and moving the train up-hill under tension, it was easy to have the car always in the right spot. We also rigged up starter buttons within reach of the shuttle car operator so that he could control the movements of the car puller from his station. By using the car puller, we were able to save an extra man and do a quicker job of spotting cars.

**Maintenance:** Maintenance of machinery in any development operation is a problem. With trackless equipment it should be realized that this is a complicated machine, heavy in construction and expensive. It is not news to mine operators that good development men are usually poor maintenance men.



Fig. 2—Typical stope loading operation.

common and 6-ft rounds were the rule rather than the exception. These average rounds compare with drift rounds in the rest of the district of  $4\frac{1}{2}$  to 5 ft. The greater speed of loading is attributed to: (1) ability to load fly rock faster, (2) ability to load in good rock at a rate of 90 tons per hr, and (3) a shorter period over all the cycle to wait for disposal of rock after it is loaded.

**Power:** Our loaders were ordered, due to local conditions, to run on 220 v ac. Each unit was accompanied by one 50 kva dry transformer with 440-220 taps. We found that with our fluctuations in voltage, it was best to keep the transformer run up within 800 ft of the shovel. Current on the 440 v side was supplied through 2-0 or 300,000 c.m. rubber-covered wire.

The shuttle car, operating on 275 v dc is limited to certain conditions. Our motor generator sets are placed around main line haulages about 4000 ft apart. Much of this development work was far off present main line track so that the problem of sufficient power and return lines was almost a constant one. However, by using 4-0 trolley or 300,000 c.m. feeder line with 300,000 c.m. return to a railroad track, it was possible to use the shuttle car in any job. With normal demands on motor-generator sets it is probable that an economic limit of about 3000 ft is practical. The wisest plan is to check voltage on both machines and not allow too great a voltage drop on either of them.

**Car Spotters Aid in Unloading:** Car puller proved to be a necessary adjunct to successful operation of the shuttle car when it is discharging into mine cars (fig. 2). Spotting cars under the end of a shuttle car with a locomotive is a slow, uncertain job, particu-

However, equipment of this nature requires care on the part of the operator and also good maintenance to prevent the machine from wearing out and losing production by having breakdowns.

Our experience has been with men, untrained mostly, in operating and maintaining this equipment. Although now they are much better than they were, they still lack certain fundamental knowledge of the machine and also the value of preventative maintenance. For full value and lowest possible repair costs, considerations should be given to picking the right man as an operator and to giving him proper training in the operation of the machine, the working parts, and the correct lubrication of the equipment. Then, if this kind of man is available, provide enough supervision to be sure he does what he is supposed to do.

**Conclusion:** Management of St. Joseph Lead Co. bought this equipment with the idea that it would replace the old Thew shovel which had outlived its usefulness due to the fact that ore bodies were becoming smaller and lower in height. It was hoped that it would supplement the St. Joe shovel by being more flexible. This proved to be the case when it was successfully used in trackless development work. Whether or not it could be purchased and used only in development work and prove more economical than the Eimco, Conway, or drag-hoist is still more or less problematical. Having had this experience with trackless equipment, we find that we have a unit that is very flexible. It can be used successfully in stope operations, low ground, and narrow, twisting and turning ore bodies, and also in drift work in all its phases peculiar to St. Joseph Lead Co. operations.



# Determination of the Coefficient of Linear Thermal Expansion of Rock Specimens by Means of Resistance Wire (SR-4) Strain Gauges

by Louis Moyd

The Concrete Research Division, U.S. Corps of Engineers, has developed a simple procedure for determining the coefficients of linear thermal expansion of rocks by means of resistance wire (SR-4) strain gauges. Gauges are cemented to the surfaces of specimens, which are then brought alternately to temperatures of 35°F and 135°F. Results are accurate to the seventh decimal place.

IN the course of investigations of the thermal properties of various rocks proposed for use as aggregates in concrete structures, a relatively rapid method of determining coefficients of linear thermal expansion was required. A method which proved to be satisfactory for this purpose was developed in the petrographic laboratory of the Concrete Research Division, U.S. Corps of Engineers, at Clinton, Miss. The method is based on the fact that strain effects in a specimen can be determined by measuring variations in the resistance of an electrical conductor firmly cemented to the surface of the specimen.

LOUIS MOYD, Member AIME, formerly Geologist, Concrete Research Division, U. S. Corps of Engineers, now Consulting Mineralogist, Bancroft, Ont., Canada. AIME San Francisco Meeting, February 1949.

TP 2861 H. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received March 23, 1949; revision received Sept. 21, 1949.

Equipment for the determination of strain, electrically, has been developed by the Baldwin-Southwark Division of the Baldwin Locomotive Works in Philadelphia. This equipment includes small strain gauges made of coils of resistance wire bonded between thin sheets of paper (SR-4 gauges), and a portable "Strain Indicator" which records variations in the length of the gauges in microinches per inch. Gauges of this type are now in general use for determinations of various mechanically-induced strains, but their use for the determination of coefficients of thermal expansion appears to be a new application. The advantages of the method are that the test specimens may be relatively small and require only a minor amount of preparation, that a large number of tests can be run at the same time, and that the installation and operation of the equipment is relatively simple. The method and equipment described are now in regular use for determining the coefficients of linear thermal expansion of rock samples submitted to this laboratory. Rocks which have been tested include granites, syenites, gabbros, and other igneous rocks, limestones and dolomitic limestones, sandstones, and quartzites.

**Specimen Preparation:** To standardize the conditions of the tests, the rock specimens are sliced by diamond saw into slabs about  $1\frac{1}{2} \times 1\frac{1}{4}$  in. In the case of bedded or foliated rocks, three oriented slabs are cut from the same specimen, and represent, respectively, a direction in a plane of the bedding or foliation of the rock; a direction along the bedding or foliation and at a right angle to the first; and a direction across the planes of bedding or foliation. One of the large surfaces of each slab is lapped for a few minutes with No. 100 abrasive powder to remove saw scars and irregularities. The slabs are soaked in petroleum ether and then dried in an oven to remove all traces of oil picked up in the sawing

and lapping operations. When the specimens are dry, two SR-4 gauges (with active gauge-lengths of  $\frac{3}{4}$  in.) are cemented to the lapped surface of each slab with Duco cement. All bubbles are pressed out, and weights are placed on the gauges (with an intervening layer of blotting paper) and left on overnight while the cement hardens and cures.

**Instrumentation:** When the cement has cured, the slabs are placed on a panel which can hold up to 36 specimens, and the strain gauges are connected into the test circuits by means of binding posts. Panels made of Transite and of Lucite are now in use. The circuits, as presently arranged, go through two eleven-point, three-bank, silver-contact selector switches. Each gauge connection is coded so that its reference points on the switches are known, and the gauges are read in the same sequence each time. The portable strain indicator is a completely self-contained unit including batteries, and is primarily a Wheatstone bridge, with additional components permitting dial calibrations in microinches per inch, and an adjustment for the use of gauges of different lengths and initial resistances. To minimize temperature effects on the gauges themselves, the circuit requires that a mounted dummy or compensating gauge be balanced against the active gauges. After an extensive series of tests, it was found that the best results were obtained when the compensating gauge was mounted on a slab of material with a known (and relatively large) coefficient of linear thermal expansion, and kept in the same environment as the specimens being tested. In practice, the compensating gauge is mounted on a slab cut from a quartz crystal, with the gauge direction parallel to the C-axis of the crystal. A test gauge is mounted parallel to the compensating gauge on the same slab. As a check on the quality of the test results, slabs having known coefficients of linear thermal expansion are interspersed among the test specimens. In practice, an oriented slab of quartz and one of fluorite are used. The panel containing the test slabs is placed in a cabinet, the humidity and temperature of which can be controlled. Three thermocouples are attached to a slab similar in size and shape to the test slabs, one of the thermocouples to the upper surface, one to the lower surface, and one cemented into a hole drilled longitudinally through the slab. The thermocouples are connected to a recording thermometer outside the cabinet. It was determined that the 100° temperature range between 35° and 135°F was most significant for concrete aggregates, since effects of freezing are introduced into concrete below that range, and temperatures above that range are rarely, if ever, encountered even on exposed surfaces in the warmest climates. The portable strain indicator is placed outside the cabinet and connected to the test panel by a multistrand telephone-type cable. A simplified wiring diagram of the test circuit is shown in fig. 1; the equipment used is shown in figs. 2 and 3.



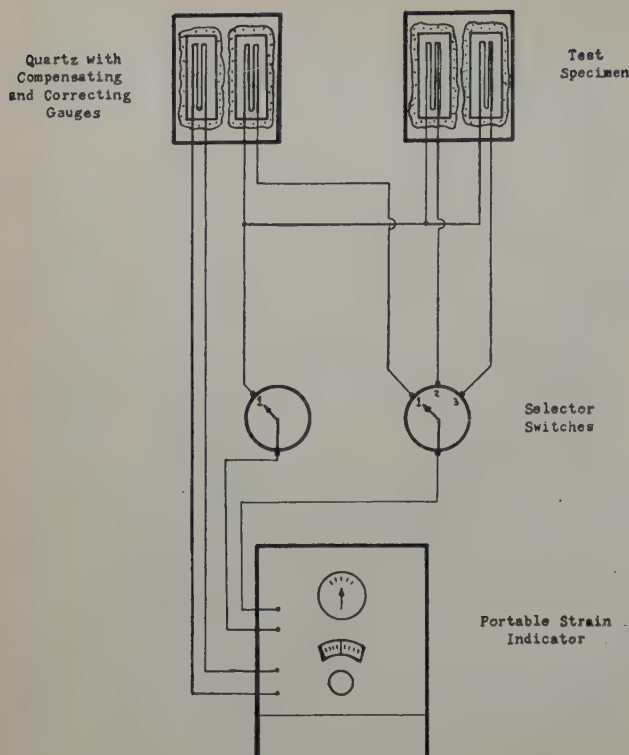


Fig. 1—Wiring diagram for determination of coefficients of linear thermal expansion by use of resistance wire (SR-4) strain gauges.

**Procedure:** After the cement holding the gauges to the test slabs has been sufficiently cured, and all the gauges are connected into the test circuit, the panel is placed within the controlled-humidity and temperature cabinet. The humidity is maintained at a minimum, and the temperature is raised to 135°F, then reduced to 35°F. No readings are taken during this first cycle, since it has been found that very erratic results are obtained until the slabs and mounted gauges have been flexed at least once. The temperature of the cabinet is then varied from 35° to 135°F, alternately, for a series of ten complete cycles, and the gauges are read at both extremes of temperature. The temperature of the test slabs is considered to be uniform throughout when all three thermocouples reach the same temperature. It was found, however, that gauge readings were more consistent if they were taken after the slabs were allowed to remain at the peak temperature for about half an hour. This condition was not found at the lower temperature. With the equipment used, it is possible to obtain readings covering two full cycles per 8-hr day, thus each complete test takes six days including one day for preparation of test specimens.

**Computations:** All of the gauge readings are expressed in microinches ( $1 \text{ in.} \times 10^{-6} = 0.000001 \text{ in.}$ ) per inch. A correction factor for each run is based on the difference in the readings at the maximum and minimum temperature of the gauge attached to the same quartz slab as the compensating gauge. Theoretically, there should be no difference in these readings, since the resistance of the compensating gauge should completely balance the resistance of the test gauge. Use of this correction factor in the computation of the coefficients of linear thermal expansion of the test specimens results in good determinations for those specimens with known coefficients, and therefore this factor is a necessary part of the computations. It is possible that the cor-

rection eliminates errors introduced into the system through extraneous forces, such as variations in room temperature, which might affect resistances in lead-in cable, switches, and portable indicator. In computing the coefficient for each test specimen, results of both gauges are averaged, unless one has given erratic readings, in which case it is presumed that the gauge or the wiring is faulty and the readings of this gauge are disregarded. A typical computation of the coefficient of linear thermal expansion for a test specimen, based on SR-4 gauge readings of a run between 35° and 135°F, is shown below:

Coefficient of quartz, upon which compensating gauge is mounted, is taken at $4.30 \times 10^{-6}$ . This multiplied by the 100° temperature difference equals $430 \times 10^{-6}$ , which is the compensation factor.	
Reading of active gauge on quartz, at 35°F (attached next to compensating gauge)	$1.435 \times 10^{-6}$
Reading of active gauge on quartz, at 135°F	1.395
Difference = Correction Factor	—40
Reading of gauge on specimen, at 135°F	1.060
Reading of gauge on specimen, at 35°F	—965
Apparent expansion of specimen	95
Compensation factor	430
Correction factor	+40
Total factor	470
Total factor	470
Apparent expansion of specimen	+95
True expansion of specimen	565
True expansion of specimen, $565 \times 10^{-6}$ per in. divided by 100° temperature range equals $5.65 \times 10^{-6}$ in. per in., which is the coefficient of linear thermal expansion for the specimen.	

Fig. 2—Resistance wire (SR-4) strain gauges cemented to rock slices and connected into circuit through binding posts on Lucite test panel.

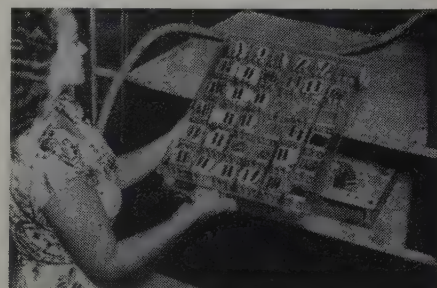
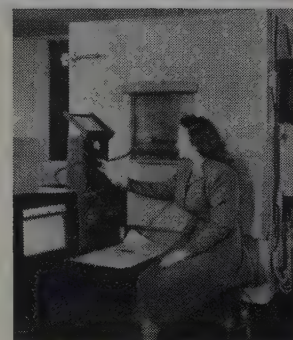


Fig. 3—Equipment used in determination of coefficients of linear thermal expansion of rock slices, including controlled-temperature and humidity cabinet, multiple-strand cable, switch panel, portable strain indicator, and recording thermometer.



This is the result of only one run ( $\frac{1}{2}$  cycle). The averaged results for the complete 10 cycle test (20 runs up and down the temperature range), always give good determinations (accurate in the first decimal place) for the interspersed, known specimens, and therefore it is considered that acceptable results have been obtained for all of the rocks being tested.

**Acknowledgments:** The application of resistance wire strain gauges for the determination of the coefficients of linear thermal expansion was conceived by the late Elliott P. Rexford, Geologist, Los Angeles District, U.S. Corps of Engineers, who used the gauges to investigate the thermal expansion compatibility of mortar and coarse aggregate in several large samples of concrete. Albert Weiner, Engineer, North Atlantic Division, U.S. Corps of Engineers, assisted in the development of the procedure in present use.



# Deposits of Heavy Minerals on the Brazilian Coast

by Joseph L. Gillson

**B**RASIL has had an industry based on ocean beach deposits of heavy minerals containing monazite, zircon, rutile, and ilmenite for well over 40 years, but except at the very earliest period, prior to 1906, and again during World War II, has this industry been at all comparable in size with similar operations in other countries. Limiting factors have been neither a lack of reserves nor lack of markets (except for zircon). Although poor local transportation and shipping facilities have handicapped the development in Brazil, these conditions were perhaps equally bad (when mining operations started) in the Indian State of Travancore, and in New South Wales and Queensland, Australia, in which states the principal production of beach deposits of heavy minerals has occurred. The outstanding restraints on the Brazilian industry have been the lower quality of the Brazilian minerals, the relative small size of individual deposits that were known until recently,

JOSEPH L. GILLSON, Member AIME, is Geologist, E. I. du Pont de Nemours and Co., Wilmington, Del.

Paper published with the permission of the Director of the Development Department.

AIME San Francisco Meeting, February 1949.

TP 2856 H. Discussion (2 copies) may be sent to Transactions AIME before July 30, 1950. Manuscript received Sept. 20, 1949.

the lack of interest of Brazilian investors in mining ventures, and the unfavorable climate for investment of foreign capital under Brazilian mining law. The lower quality of the minerals is indicated by the fact that the Brazilian monazite carries only 5 to 6 pct  $\text{ThO}_2^*$ , and the ilmenite 56½ pct  $\text{TiO}_2$ ,

\* This was important in the days when the manufacture of gas mantles was the only outlet for the chemicals made from monazite. Today, since the other rare earths in monazite are used in industry more than thorium, the Brazilian monazite is of equal value to the Indian.

whereas the corresponding Travancore minerals carry 9 pct  $\text{ThO}_2$  and 60 pct  $\text{TiO}_2$  respectively. The Brazilian zircon is a dirty brown color (which can be removed readily) as compared with the gleaming

whiteness of the Australian zircon. An inadequate market for the abundant zircon that will be available in any operation in Brazil is certainly a heavy restraint, since otherwise it could be an important byproduct. The oxide of zirconia, baddeleyite, that has come from Brazil is from other sources.<sup>1</sup> Since the Brazilian beach sands carry only a little rutile, that mineral has not been available as a valuable byproduct to help finance the operations.

Substantial tonnages of rutile have been exported from Brazil. The sources of this mineral are stream placers and residual deposits over gneisses in the States of Goyaz, Minas Geraes, Ceara, etc. (Leonardos,<sup>2</sup> and Chambers<sup>3</sup>).

With these handicaps, Brazilian production from beach sands has been marginal, and there has been only one consistent producer. This is a company, originally French, which was called *Companie de Franco-Brazillienne*, but, with the downfall of France in 1940, became Brazilian under the name *Monazita e Ilmenita do Brasil* ("Mibra"). This company has operated several deposits in the vicinity of Guarapary, in the State of Espirito Santo. Its office and plant has been at Guarapary.

Recently a company called "Fomil" (Fomento Monazita-Ilmenita) started operations. This company was organized by the Foote Mineral Co. of Philadelphia, and such share as a foreign company can hold under Brazilian law has now been acquired by the Lindsay Light and Chemical Co. of Chicago.

When the writer went to Brazil in the fall of 1940, he found that no general survey of the beaches had been made except for a thorough study of part of the area by Abreu.<sup>4</sup> The paper by Miranda<sup>5</sup> had not then been written. The federal Geological Survey and Department of Mines knew only of the existence of the deposits in Espirito Santo south of Vitoria, and had a hazy idea that others might be found north of Vitoria. One deposit in Baía was known in Rio, but the larger one at Guaratiba was not known, although a Frenchman had prospected

it rather thoroughly 20 years before. No knowledge of the deposits in Natal existed.

The discovery of extensive and numerous deposits resulted from visits to areas favorable geologically and there making local inquiries for knowledge of black sands, and also "wild catting" potential beaches. Reserves in deposits newly found exceed the sum of all those known previously.

Throughout this article the black titanium mineral is called by its commercial appellation, ilmenite, although mineralogically it is arizonite except in two deposits. Arizonite is the ferric iron analogue of ilmenite, and is higher in  $\text{TiO}_2$  (ref. 6, p. 1042). In a recent paper by Overholt, Vaux, and Rodda,<sup>16</sup> an opinion is expressed that arizonite is weathered ilmenite. The writer questions this generalization as not being in accord with world occurrences.

### Location of the Deposits

The Brazilian deposits of heavy minerals on ocean beaches occur in a zone about 175 km long, extending north from the northeast corner of the State of Rio de Janeiro, up into the contiguous State of Espirito Santo as far as the mouth of the Rio Dôce. Further north there is a short zone in the southern part of the State of Baía, about 40 km long. No other deposits are known except in the States of Paraíba do Norte and Natal on the "hump" of Brazil (figs. 1, 2, and 3).

This limitation of the deposits to specific zones has a direct and positive relation to the local geology. The deposits occur where the bottom beds of the coastal plain are being eroded, or were recently eroded, by the sea.

The sections of the coast in the States of Rio de Janeiro and Espirito Santo are moderately well populated and developed, but they lack roads parallel to and near the coast, and there are only a few ports, only one of which can be entered by large ships and at which there are docks and facilities for handling cargo. This is the port of Vitoria, the capital of the State of Espirito Santo. Only some of the smaller deposits are near enough to Vitoria to permit consideration of trucking the products from the mine to the dock. The small port of Guarapary, about 50 km south of Vitoria, can be used by vessels up to 3000 tons, but port facilities are inadequate. Other ports along this section of coast are so small that only lighters and small fishing boats can enter.

An anchorage north of Itapemirim (fig. 2) can be considered as a potential loading point for the big deposits in the north end of the State of Rio de Janeiro, but roads south from the anchorage are nonexistent below Marataíses.

The deposits in the southern part of the State of Baía can be reached from the small port of Caravellas but roads serving the coastal area are very poor. Offshore loading from these beaches is possible, however, because of the protection afforded by reefs, and before 1905 there was extensive shipping off these beaches.

The deposits in the northeast coast have been inadequately explored. Those near the mouth of the Paraíba River below São João de Passôa are reasonably accessible, but probably too low grade, while others in Natal, near the fishing village of Cunhaú, are large but poorly located for shipping.

### Types of Deposits and General Geological Features

A very large area in east central Brazil is underlain by pre-Cambrian gneisses.<sup>7</sup> The coastal belt of gneisses extends north from the State of Santa Catharina, with but one interruption, for 2000 miles into the north coastal section in the State of Ceara. Drs. de Oliveira and Leonardos<sup>7</sup> have shown that the belt is nearly 300 miles wide in the vicinity of Rio de Janeiro, 150 miles wide in Baía, and nearly 400 miles wide in Pernambuco and Paraíba do Norte.

The gneisses reach the coast from the southern limit of the pre-Cambrian area as far north as Cabo Frio, east of Rio, and the familiar Sugarloaf Mountains of Rio harbor are spectacular physical features of the gneissic terrain, and owe their form to exfoliation and cleavage along the planes of structure.

From Cabo Frio north a short distance to the town of Macaé, the gneissic cliffs continue to the sea, but from there north almost without interruption, there is a fringe of coastal plain sediments along the actual coast (figs. 3 and 4). For long distances, however, the coastal sediments are so thin that the underlying gneisses are seen at the water's edge or in islands offshore. The delta of the big Rio Dôce (fig. 2) buried the older coastal plain sediments in the vicinity of Regencia, and from there north to Caravellas the shore is cut in a higher section of the Tertiary beds. For a short stretch near Caravellas the lower section is exposed, and again deposits are found. Higher and higher beds are found at the coast to the north, and the stratigraphic section is so thick that it has been explored for oil in the vicinity of São Salvador (fig. 4).

In Paraíba do Norte and in southern Natal, the coast is cut in Cretaceous beds, which also carry a little ilmenite.

The geographic association of the beach deposits of heavy minerals with the bottom beds of the

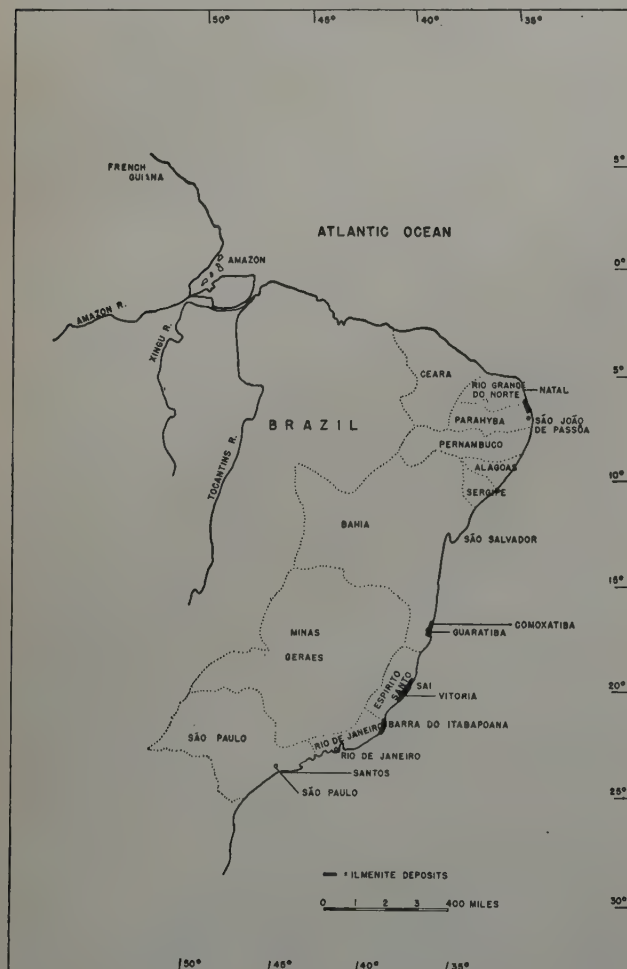


Fig. 1—Index map of Brazil.



coastal plain is too consistent to be coincidental. The formation of the deposits of heavy minerals is related so intimately with the shoreline developments that a review of the recent physiographic history is necessary.

The gneissic area has had at least two periods of peneplanation. They may correspond in general age with the Cretaceous and Tertiary peneplanes of the Appalachian region.

Only in the soil overlying the Cretaceous land surface was there a concentration, by extraction of soluble minerals, of the grains resistant to chemical decay. Such minerals are ilmenite, rutile, zircon, staurolite, sillimanite, kyanite, spinel, tourmaline, and others. Magnetite, which occurs in the unaltered gneisses with the ilmenite, had been almost completely eliminated by weathering. This important elimination of an objectionable associate of ilmenite before transport of the products of rock disintegration to the coast is an essential step in the later formation of beach deposits of high-grade ilmenite. Similar conditions had obtained in India, Ceylon, and other places where high-grade deposits of ilmenite occur. On the other hand, where a black sand is found in deposits formed only by mechanical disintegration of the parent rock, the ilmenite is contaminated with magnetite from which it cannot be separated completely. Such a deposit of titaniferous magnetite is found in Brazil in the delta of the Rio Dôce which is made up of material brought down from the deeply dissected valley recently carved in the hard rock.

The source of the abundant monazite can be inferred to be from pegmatites which are widely distributed in the gneisses. This inference is made simply because monazite is known to be an accessory in many pegmatites in North America.

Following the Cretaceous peneplanation, uplift caused rejuvenation of the drainage, and the deep surface soil containing these resistant minerals was removed by rapid erosion and dumped into a shallow sea lying over the continental shelf, where the sand was deposited at once and the accompanying clay washed away. Thus, the basal beds of the coastal plain sediments carry everywhere a small percentage of these resistant minerals. In no place has the content of these minerals been found to be sufficient to justify consideration of mining the Tertiary rock as such. Had not subsequent marine erosion of these beds occurred with further concentration of the minerals by waves and currents, there would be no deposits of workable grade in Brazil.

The more recent history of the Brazilian coast has been marked by repeated changes in level of the land with respect to the ocean. There has been tremendous faulting parallel to the coast, which has dropped the east side and left inland, elevated and undisturbed, the old-age erosion surfaces that now have no connection whatsoever with the precipitous cliffs and land surface in early maturity, characteristic of the down-dropped coastal section. In no place is this contrast seen more clearly than in the automobile drive from São Paulo to Rio de Janeiro. The city of São Paulo is spread out on an old land surface along a big river, Paraíba do Sul, which flows tranquilly north and parallel to the coast on a gentle grade, oblivious of the tremendous falls and canyons ahead. Upon reaching the "Serra" near Rio, one looks back on this land surface of old age, but forward down toward the sea over a tremendous fault scarp, up which the roads twist and wind in breath-taking curves, and down which the youthful



Fig. 2—Map of a part of Estado do Espírito Santo showing ilmenite deposits.

streams rush headlong to the sea, 1000 m below. At other points, a puffing railroad engine pushes only one light car at a time up the cogwheel roads from Santos to São Paulo, and from Rio to Petropolis and Rio to Terezopolis, climbing up this eroded fault scarp.

After the present topographic form of the coastal area by deep dissection of the fault scarp was well advanced, drowning occurred, and the deep harbors of Rio and Vitoria resulted from the submergence of the mouths of old rivers, and of low-lying areas. By submergence, the harbor at Rio, the "most beautiful harbor in the world," came into being, and also every stream along the coast became marked by a wide estuary at its mouth. This drowning, comparable in degree and probably in age with that of the North American continent which produced the

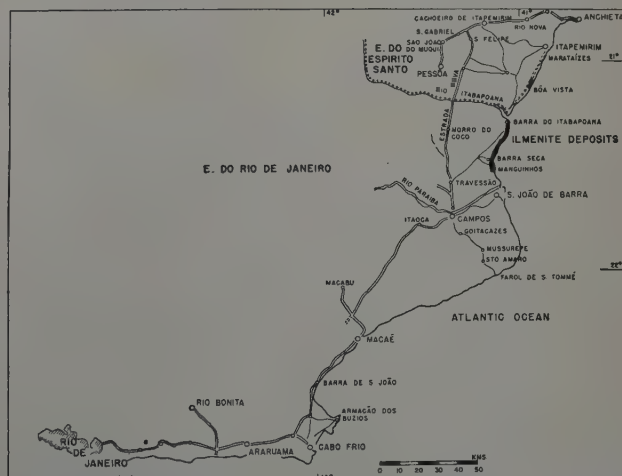


Fig. 3—Brazilian coast from Anchieta to Rio de Janeiro showing ilmenite deposits at Barra do Itabapoana and Bôa Vista.

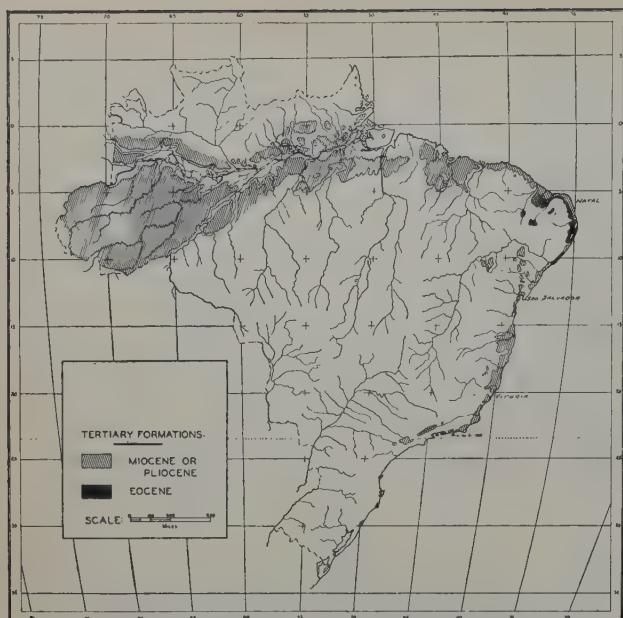


Fig. 4—Geological map of Brazil.

estuaries of the Chesapeake, Delaware, and Hudson, was sufficiently remote in time (probably early Pleistocene) for subsequent advanced development of the coast line. Many of the more pronounced headlands have been lopped off, and deeply indented bays were closed off by sandbars, behind which salt marshes developed which gradually filled.

Very recently geologically, there has been a modest uplift of about 5 m, and textbook examples of all of the features characteristic of recent emergence are visible along the Brazilian coast. Old sea-cut cliffs stand back from the sea, with old shore lines buried under a mass of thick vegetation. Elevated offshore bars stand now "high and dry" as ridges between the present shore and the old sea cliffs. The estuaries at the mouths of smaller streams are closed by elevated bars behind which are lagoons.

Where the pre-Cambrian gneisses come down to the sea, the shore line prior to this emergence had been very rugged (as at Rio). The slight emergence has put a strand line around the base of the old cliffs. Except for this emergence and consequent formation of a strand line, there would have been little flat land on which to build a city such as Rio. The popular suburb of Rio, called Ipanema, is built on the elevated bar built across an old bay. This is now closed off from the sea to form the lake called Lagoa Freitas.

Where the more easily eroded coastal-plain sediments lie at the coast north of Macaé, the modification of the shore line, after submergence, had been further advanced than in the harder and more resistant crystalline rocks. A particularly well-defined sea cliff had formed by the rapid erosion of the soft sediments by the waves. An earlier terrace had been established on these soft sediments, probably by stream benching, and the height of this terrace at a particular place depends on the distance the old valley lay from the seacoast prior to submergence. The sea cliff formed a scarp below this terrace which, in many places in the belt where the black sand deposits occur, is 20 to 50 m high. The old elevated sea cliff is such a conspicuous part of the Brazilian coastal landscape, that it is known generally by the residents as the "barreira" (barrier or cliff).

Since the emergence, in places marine erosion has been sufficient to cause the complete destruction of the elevated strand at the base of the scarp, and attack on the old sea cliff has been renewed.

Prior to the emergence, waves were breaking against the barreira and offshore bars were developing from headland to headland in the familiar process of shore-line straightening. A sketch attributed to Douglas Johnson<sup>8</sup> showing "submaturity" in shore-line development of submergence, represents most accurately (if too simplified) the coast of Brazil in the State of Espirito Santo. If the effect of slight emergence were added to that picture, and a strand line were drawn in front of the old cliffs and bars extending across the bays, then the present shore line would be indicated.

Prior to the emergence, the waves dashing against the sea cliff undercut it, causing large chunks of the poorly consolidated material, of which the Tertiary sediments are made, to fall into the waves from time to time. The waves digested these chunks, carrying the minerals of light weight away, and leaving the heavy minerals on the beach about where they fell. Thus, a black beach formed at the base of the barreira; black, since the most abundant of these heavy and resistant minerals is ilmenite. Such concentrations of heavy minerals are forming today as marine placers at Maíba in Espirito Santo, and at Comoxatiba in Baía. Elevation of these beaches left at the foot of the barreira a black sand deposit now thickly covered with brush, or used for the cultivation of mandioca, as at Saí and Ponta da Fruta.

Prior to the recent emergence, enrichment of the sand on the offshore sand bars occurred in a manner very similar to the action of concentration on a Wilfley table. The sand derived from the destruction of a headland was moved along the coast under the influence of along-shore currents. A little concentration took place during this migration since the grains of lowest weight were lifted by the restless motion of the water and remained in suspension. As the bar grew in size and height, it began to give resistance to the incoming swells from the ocean, and breakers formed. As these "rollers" dragged bottom on the bars, they caused a movement cross-wise with the length of the bar, and this movement had sufficient force to carry the lightweight minerals off the bar into the deeper water on the landward side. These rollers thus acted like the wash water on a Wilfley table. The heavy minerals remained behind on the bar, thus the sand was enriched by subtraction of those minerals of light gravity.

For a while the bars were beaches themselves, but, with the progressive elevation of the coast, emergence lifted them above the waves and permitted the wind to do some work. In most places the wind dumped some barren sand on the elevated bars so that the top layer may be quite barren. Continued emergence exposed a strand line on the sea side of the elevated bars. This strand is now the actual shore and in most places contains no "heavy mineral" of importance.

There are thus the following types of deposits of heavy minerals on the Brazilian coast: (1) Elevated beaches. (2) Elevated bars. (3) Modern beaches. (4) Dune deposits formed by the action of the wind on (1), (2), and (3). Since dunes are not a conspicuous feature of the Brazilian coast, presumably because of the rapid growth of protective vegetation, deposits of this kind are rare. The only one important in size and grade is at Cunhaú in Natal.



(5) A delta deposit at the mouth of a big river which transported material derived from recent deep-river erosion into fresh rock.

### Comparison with Similar Deposits Elsewhere

The relative small size of the individual Brazilian deposits as compared with those in Travancore, India, results presumably from the fact that the "protore,"\* that is, the sand in the coastal plain

\* This adopts Ransome's term used for submarginal ore converted to commercial ore by secondary enrichment.<sup>9</sup>

sediments, was itself so low grade that the quantity of mineral available in one place was only of modest proportions, whereas locally in Travancore, the "terra rosa" which formed during tropical weathering as a deep soil zone over the gneisses, was considerably enriched in heavy minerals. In Travancore, black sand lies in every roadside ditch cut in the red earth at Manavalakurichi, at Quilon, and along the highway to Alleppey.<sup>6</sup> In Brazil, only traces of black minerals are visible in drainage channels around Vitoria or Caravellas. Hence, in Brazil, deposits formed only where the physical conditions of wave, wind, and ocean current were most favorable to concentrate the moderate amount of black sand available. In Brazil, however, the protore was available for scores of kilometers, and numerous deposits resulted. In Travancore, the low-lying shore did not offer general conditions for rapid marine erosion. The heavy mineral was brought down to the sea by big rivers, and the two deposits on the Travancore coast lie only where the mouths of these rivers were at one time. In Brazil, except for case 5, rivers had little or nothing to do with transporting the material, except originally in Tertiary time when they brought it down to make the coastal plain sediments.

In turn, reasons for the insignificant deposits of heavy minerals of commercial importance on the North American coast, in spite of intense concentrating conditions in the ocean itself, reflect this exceedingly low tenor of heavy minerals in the sands of the coastal plain sediments, which have been exposed to marine erosion, and in material brought down by the rivers.

The two deposits in Florida now being worked are not on or near modern beaches. One of these lies between Jacksonville and the coast, and the other 50 miles inland, at Trail Ridge on the former site of Camp Blanding. Both are large, but they consist of very low-grade sands. In age they are probably older than the Brazilian. The writer believes that they resulted from some special conditions of sedimentation which, because of the antiquity of the deposits, are difficult to interpret in detail. The Jacksonville deposit is related to an abandoned channel of the St. John's River and may have formed at a time when the mouth of that river was in that vicinity. The height-of-land called Trail Ridge was, in early Pleistocene time, the entire existing peninsula of Florida. It was a sand spit directed toward that hill now called Iron Hill near Lake Wales, on which the Bok Tower stands. Iron Hill was at that time an island in the sea (ref. 10, fig. 12, p. 35). The deposits of heavy minerals formed on the west or the gulf side of this spit.

The small beach deposits formerly worked at Ponte Vedra, near Jacksonville, and recently at Vero Beach, Fla., are storm-line deposits (ref. 11, p. 1592).

Black beaches on many volcanic islands, like those in the West Indies, formed exactly as case 3, by the

undermining of the cliffs by waves. The source material (lavas), however, contained no ilmenite of acceptable commercial quality but did yield an excess of other black minerals of no value, such as titaniferous magnetite, augite, etc. The black beaches and dunes of New Zealand resulted from similar conditions of fluvial erosion and marine sorting as in Brazil, with the streams and waves working on a source material of volcanic ash very rich in titaniferous magnetite. In New Zealand where ancient and active dunes are of very large size, wind played and continues to play a very important role. Vegetation on the Taranaki coast of New Zealand where the deposits occur must have been almost non-existent after a violent volcanic eruption of Mt. Egmont. The wind is still too strong along that coast to permit vegetation to get a start on the dunes in many places, as at Nukumarū, for example.

The layers of rutile-zircon sands on the eastern Australian coast described by Fisher<sup>12</sup> are elevated beach deposits, derived from the erosion of old sediments. There was little ilmenite in the source material.

### Description of Individual Deposits in Brazil

The deposits in Brazil can be grouped geographically, as follows:

1. Those south of Vitoria in the States of Rio de Janeiro and Espirito Santo, which are not only the most numerous but include the largest individual ones.

2. Those in Espirito Santo, north of Vitoria.

3. Those in Baía, principally at Guaratiba and Comoxatiba. These deposits are of intermediate size and contain ilmenite of a quality comparable to the Indian.

4. Those along the hump of Brazil in the States of Paraíba do Norte and Natal.

Because of the number of individual deposits, it is not possible in any paper of moderate length to describe each individually. Such descriptions must remain for a government monograph. Since detailed maps and drilling records have been filed with the Brazilian government, the information is available to interested persons.

**Deposits South of Vitoria:** The deposits south of Vitoria can be divided into those south of Barra do Itapemirim and those north, mainly in the stretch of coast between Anchieta and Guarapary.

Those south of Barra do Itapemirim are: the single deposit of moderate size at Boa Vista de Siri, and the very large deposits, of which twenty-two were mapped, lying between Barra do Itabapoana\* and

\* This name, obviously Indian in origin, is really very easy to pronounce—Ita-ba-po-ana.

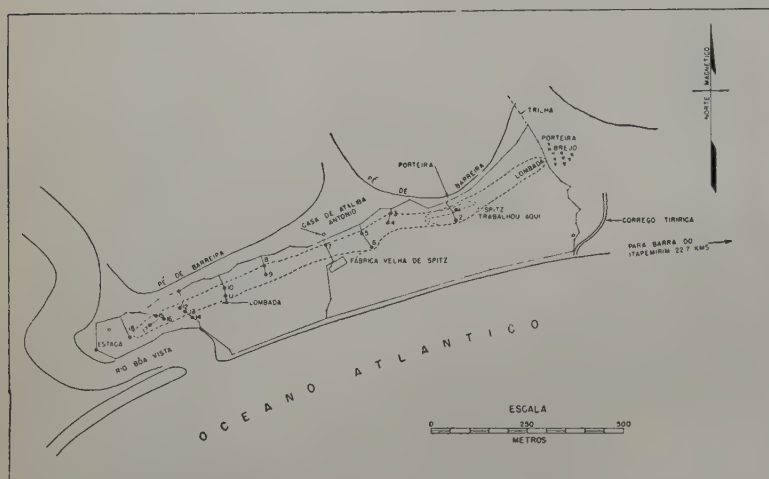
the mouth of the Paraíba do Sul at São João de Barra. With one or two exceptions, these numerous deposits are all of the elevated bar type.

The Itabapoana River is of very moderate size and forms the boundary between the States of Rio de Janeiro and Espirito Santo. The town of Barra do Itabapoana is a small port from which sugar and mandioca are shipped in very small coastal vessels. Offshore anchorage is unsafe. The town is reached by a road which is passable except in very wet weather from a station on the Leopoldina Railroad called Travessão, which is about 16 km north of Campos.

The deposits begin at the very edge of the town of Barra do Itabapoana, and extend south along the coast for over 20 km. They were designated by



letters, which run from A to V. Most of them are narrow, long ridges with a moderate surface expression. Although a little black sand can be seen in the surface sand, the rich layers in most cases are buried under 50 cm or so of barren or low-grade sand. Some of the deposits are small and others only a few hundred meters long. Two others, called P and U, are each nearly  $2\frac{1}{2}$  km long. The width varies, but the high-grade zones are not over 20 to 50 m wide. The depth to barren sand is less than 4 m in most places. The grade of the sand varies through wide limits, but for all deposits in the area the calculated average is 32 pct heavy mineral, which includes all minerals over 2.9 in gravity. Of this heavy mineral assembly about 55 pct is ilmenite;



5 pct rutile; 25 pct zircon; and 5 pct or more, monazite.\* The balance is composed of other heavy

\* Exact quantities of monazite are impossible for the writer to give because of sampling difficulties. In splitting down a large composite to obtain a small sample for analysis, the monazite is not easily controlled since it is heavier and most of the grains are so much finer than those of the other minerals. The writer was sure of his results only in cases where a large sample was concentrated by ore-dressing methods, and all of the products weighed and analyzed, preferably with a Geiger counter.

minerals, principally tourmaline, staurolite, sillimanite, kyanite, and corundum. Spinel is abundant in the Itabapoana sand, but rare or absent in that of all of the other deposits.

Because of the attenuated nature of the deposits, with a low tonnage per meter of length, with no usable roads in existence parallel to the beach, with no safe anchorage nearer than 50 km, and two wide rivers to cross within this distance, and no developed power, the Itabapoana deposits present problems for mining and shipping in spite of the extent of the reserves.

The deposit at Bôa Vista de Siri is an interesting example of a single, isolated bar-type deposit which lies close to the barra. Fig. 5 is shown to illustrate this only as a typical occurrence. Bôa Vista can be reached via the beach from Marataíses, a resort town a short distance south of Barra do Itapemirim. The beach can be traveled in a jeep very readily at low tide. It is one of the few sections of beach along the Brazilian coast visited by the writer that can be negotiated in a wheeled vehicle.

This deposit was worked for monazite by an Alsatian named Spitz, around 1910 to 1912. How the monazite was shipped in those days is not known. Some parts of the deposit are quite rich in monazite, but the total tonnage is not large. The deposit has not been worked in recent years. It belongs to "Mibra."

The deposits between Barra do Itapemirim and Vitoria are more noteworthy for their number than for their size, but all of the recent production has come from this group. From south to north, the deposits south of Anchieta are known as Caju and Piuma, and south of Guarapary they are Paraty, Ubu, Maiba, Ubahy, Meaípe, and Lima. North of Guarapary is a deposit, presumably exhausted, which supplied the bulk of the tonnage produced to date by "Mibra." Several kilometers further north is one at Ponta da Fruta that is quite different mineralogically from all the others in southern



Fig. 6 (above)—Dalas, or rockers, used in concentration of monazite.

Although primitive, they effectively produce a concentrate acceptable to the trade, with a moderate recovery. This plant is at Paraty.

Fig. 5 (left)—Outline map of Bôa Vista de Siri, State of Espírito Santo.

Brazil. In this deposit the "ilmenite" is similar to a true ilmenite, since it analyzes about 54 pct  $\text{TiO}_2$ , and contains principally ferrous rather than ferric iron, thus differing from arizonite which is the titanium mineral of most of the deposits. Here "Fomil" began production in 1946, principally for zircon.

Another peculiarity of the deposits in the vicinity of Guarapary is a rather high garnet content. Garnet is a rare accessory in the other deposits. The deposit at Ponta da Fruta is unusual also in that it contains considerable andalusite.

All of the deposits in this section are of the elevated bar type except the one at Maiba, which is a modern beach placer. Some of these deposits, though small, are rather rich in monazite, and were worked for that mineral alone in a period around 1910 to 1914 (by the Germans), and more recently by a local company, and later by Lindsay Light and Chemical Co. of Chicago. Some thin layers can be found that are so rich in monazite that they are very yellow in color. Since monazite is so much heavier than the other minerals, a concentrate of it can be made (with low recovery) by gravity methods alone. The earlier operators used a hand-operated device called in Portuguese a "dala" (fig. 6). A few shovels of sand are placed in the dala, in which a strong current of water is flowing. The dala is rocked like a baby's cradle, which helps to shake the heavy minerals to the bottom, and the light ones to the top, and the current of water causes the light grains to migrate more rapidly to the lower end.

"Mibra" has a plant at Guarapary consisting of tables and magnetic separators. "Fomil" installed tables and electrostatic machines as well as magnetic separators.

"Mibra" and "Fomil" have shipped the bulk of



their production from Vitoria, having transported it there partly by launch, and partly by truck.

**Deposits North of Vitoria:** Several deposits occur north of Vitoria, but south of Ara Cruz, and can be reached by road. These are known as Carapebús, Capuba, Jacaréipe, and Bôa Vista de Nova Almeida. Carapebús was worked at one time during the 1910 to 1912 era, and is attractively located because shipments could be made from the point called Ponta Piraem, at the extreme south end of the deposit, by barge across the relatively protected waters of the outer harbor of Vitoria. Capuba and Jacaréipe are contiguous deposits of the long attenuated bar type. Bôa Vista is of insignificant size.

The deposits north of the river at Ara Cruz (Rio Piraqué) are inaccessible except by trail, but one called Saí is of interest because it is fairly large and is the best example of the elevated beach type. The black sand lies at the base of the barreira, in a thin layer, sloping out toward the sea. It represents an old marine placer exactly like the modern ones at Maiba and Comoxatiba, but now, due to the 5-m uplift, the "beach" is no longer washed by the waves.

The deposits in the area are uninteresting since the ilmenite is mixed with magnetite altered almost completely to hematite, but still weakly magnetic, and the best grade of concentrate that could be made analyzes only 53 pct  $\text{TiO}_2$ .

The last deposit of any importance in this section is one at Barra do Riacho about 8 km north of Saí.

Further north 25 km is the delta of the Rio Dôce, one of the big rivers of central Brazil. Its delta is spread out widely over the coastal plain, and in the delta are numerous layers of concentration of black minerals, dominantly titaniferous magnetite.

The mineral grains are much coarser than those in all of the other deposits, and garnet is the only other heavy mineral of importance. The best ilmenite concentrate made analyzed only 37 pct  $\text{TiO}_2$ .

The deposit is intriguing because of its size and geologically interesting because the mineral association shows how essential the period of deep weathering of the old land surface was to the later formation of a high-grade deposit. The weathering, by chemical decay, eliminated the magnetite and other objectionable minerals, thus enriching the soil in ilmenite, zircon, rutile, and so on.

About 20 km up the Rio Dôce is a group of lakes (fig. 2) that have a great deal of geological and historical interest, and some deposits of insignificant size which, however, contain ilmenite of the highest quality in the southern field. These lakes are lagoons or land-locked estuaries, left far inland by the outward rapid construction of the delta of the Rio Dôce. Their eastern shores are actually elevated sand bars built across the mouths of small streams, when that was the ocean shore. The area has historical interest in that, following the American Civil War, a colony of Southern white families sought refuge from Northern "carpetbaggers" by emigrating to Brazil. A number settled around these beautiful lakes, but the pioneer economy of the day was too harsh and the settlement failed. Today there is no apparent record of this colony, neither in the buildings nor in the memory nor cultural inheritance of the people. A little earlier, the great emperor, Dom Pedro II, had come to Lagôa Juparana several times for holiday, and an island in the lake is known as "Ilha do Imperador." The logistics of transporting the imperial

entourage to the region in the 1850's must have been a problem, and royalty must have endured physical hardships to which few vacationists today would submit. The writer spent a night in a bayou of the delta trapped in a dug-out "canoe" by "Água Pé" (water hyacinths), and his bout with the mosquitoes, in spite of nets and repellants, was unpleasant. Nights must have been equally difficult a century ago, and in those days there were no outboard motors to propel dug-out canoes at dizzy speeds, as today! The black sands of Lagôa Juparana lie on beaches at the foot of the barreira which lines the shore of the lake, and in a few sand bars across bays. The tonnage, unfortunately, is too low for economic consideration.

**Deposits in Baía:** The short zone of favorable areas in southern Baía is so far from the other areas, north and south of Vitoria, that an operation to be established in Baía would be so isolated from the others that it would be completely separate.

Furthermore, the two known deposits are too far apart (about 40 km) to be operated together without some complications. Since the reserves of each are large enough to sustain only a moderate size operation, production costs will be increased by small volume. The northern one of the two deposits, called Comoxatiba, is a modern marine placer, and at high tide the waves dash against the base of an unscalable cliff, thus restricting the daily hours of potential working and limiting the kind of equipment to be used to the most mobile of units.

Other factors that will delay the development of these Baian deposits are the shallow depth of the channel to Caravellas, which limits access to boats of 12-ft draft, the lack of rail connection to Rio, and the high incidence of malaria in the district. All equipment used in construction of the huge military airport at Caravellas was brought from Rio by barge and unloaded at a beach head established below Caravellas.

The coast north from Caravellas is marked by almost continuous reefs composed of layers of the Tertiary sediments which are cemented by iron oxide. Although these reefs present hazards to navigation and have caused many disastrous wrecks, nevertheless they provide protected waters along-shore for barge loading from the beaches. Protected anchorages exist offshore for ships to anchor while loading from barge to steamer. Thus shipping of ilmenite and the associated minerals can be accomplished off the beaches, just as it is done in India.

The southern deposit is near a little point called Guaratiba, and a few kilometers north of the attractive little fishing village of Alcobaga. The deposit is an elevated bar, 5600 m long, and 25 to 200 m wide. It is the longest bar with continual sand of workable grade that was explored on the Brazilian coast. The deposit is covered with a forest of large trees. This is unique since all of the others lie in fairly open grassy land, or in cultivated fields. Another unusual feature of the Guaratiba sand is the presence in it of andalusite as the most abundant accessory mineral. The quantity of monazite present is only moderate. The ilmenite analyzes better than 61 pct  $\text{TiO}_2$ , and hence is of better quality than the Indian. An analysis of a big sample shows the ilmenite actually to be nearly pure arizonite with only 2.5 pct FeO. Had the tonnage of reserve in the Guaratiba deposit been two or three times larger, development of this deposit could be most attractive.



The Comoxatiba deposit lies 40 km north, near a village of that name. Until recently the road north from Alcobaça was practically impassible, although the writer made it once in a Model A Ford, with the help of several men with axes. Near Comoxatiba the sea had eroded all of the elevated strand below the old sea cliff, and is gnawing away anew at the Tertiary beds. Being marked by freshly exposed surfaces, the cliffs colored by the reds and buffs of the argillaceous sands, are conspicuous far out to sea.

Like all deposits of the beach type, the black sand belt is confined to a cove, the concavity of which has prevented lateral migration of the heavy minerals. The deposit is over 3 km long, and consists of the top layer of the beach, 1 to 2 m thick, which in many places is so rich that it is very black, with thin yellow streaks of monazite. At low tide the beach is 15 to 30 m wide. The sand is richer in monazite than any other of the larger deposits. The ilmenite is probably not as high grade as that of the Guaratiba deposits, although the writer has only one analysis available, made on the most magnetic fraction. This analyzed only 57.14 pct  $\text{TiO}_2$ .

This deposit was worked about 1903 by John Gordon, who shipped the monazite in ballast to Germany. He concentrated the mineral in sluices. It is believed that he shipped the considerable quantity of about 25,000 tons.

An interesting fact about the Comoxatiba deposit is that it continually renews itself by the progressive destruction of the cliff of Tertiary sands. How rapid is the rate of renewal is not established, but it is possible that much of the monazite now on the beaches is a new accumulation since Mr. Gordon's day.

**Deposits in Northeast Brazil:** Search for deposits along the "hump" of Brazil followed a hunch developed after discussing possible potential areas with Dr. Glycon de Paiva of the Brazilian Geological Survey. A trip was made to Natal by car from Recife. This section of the Brazilian coast has a better developed transportation system than further south, since the terrain is not so difficult.

The first indication of occurrence of ilmenite in the red sands of the Cretaceous beds was in the river shore near Cabadello, the port of São João de Passôa in the State of Paraíba do Norte. These beds are probably too low grade to work. At Natal the same red beds occur and black mineral is seen in the ditches within the town itself, but no important accumulation was found along the coast. The captain of the Port of Natal, who had lived at the fishing village of Cunhaú in the southern part of the state, recommended a look there. This little port is 50 km south of Natal, to which it is connected by a passable road through Canguaretama. The channel into the port is too shallow for boats larger than 100 tons.

The black sands were found to lie along the channel on both sides, and on the beaches north and south of the village for several kilometers. There is a big dune deposit between the sea and the old sea cliff. The tonnage in this dune is very large, although the average grade of the sand is low.

Ilmenite and zircon make up 85 pct of the heavy mineral content. Monazite and rutile are present in insignificant quantities.

The ilmenite analyzes only 56.2 pct  $\text{TiO}_2$ , and carries 17.8 pct FeO and 21.6 pct  $\text{Fe}_2\text{O}_3$ .

## Description of the Minerals

The minerals of the Brazilian beaches are as follows:

Mineral	Pct of Heavy Mineral
Magnetite	Trace to 3
Ilmenite	35 to 75
Monazite	1 to 20
Zircon	5 to 35
Rutile	$\frac{1}{2}$ to 5
Sillimanite	$1\frac{1}{2}$ to 5
Kyanite	$\frac{1}{2}$ to 2
Corundum	Trace to $\frac{1}{2}$
Spinel (several varieties)	$\frac{1}{4}$ to 5
Staurolite	Trace to 3
Others	Trace to 3

The "others" include garnet, tourmaline, andalusite, hornblende, etc. Garnet is abundant in the beaches around Guarapary, and andalusite in the sand at Guaratiba. Spinel is abundant only at Barra do Itabapoana. The accessory silicates in all Brazilian beaches are much less abundant than they are in the Florida sands, for example, in which staurolite and tourmaline make over 25 pct of the "heavy mineral" (ref. 11, p. 1593).

The ilmenite, except at Ponta da Fruta, and probably part of that in the Natal deposits, is of the arizonite variety, having the theoretical formula  $\text{Fe}_2\text{O}_3 \cdot 3 \text{TiO}_2$ . In another article the writer (ref. 6, p. 1043) has discussed the name arizonite as applied to the beach mineral found in India, Brazil, Florida, and other places. The Brazilian mineral analyzes from 2.5 to 6 pct or more of FeO. The iron content runs from 30 to 34 pct.

Comparative analyses of various "arizonites" are as follows:\*

	$\text{TiO}_2$	Fe	FeO	$\text{Fe}_2\text{O}_3$	$\text{Cr}_2\text{O}_3$	$\text{V}_2\text{O}_5$
Indian (Quilon)	60.4	24.7	9.55	24.64	0.17	0.36
Floridian (Trail Ridge)	64.8	21.9	3.4	27.5	N.D.	N.D.
Brazilian—500 ton shipment from Guarapary in 1942	53.9	25.12	6.68	28.5	0.05	0.45
Itabapoana—average sample	57.0	27.0	3.60	34.8	0.08	0.22
Carapebus	58.06	25.8	5.92	30.46	0.03	0.44
Guaratiba	61.4	24.3	2.5	31.90	0.07	0.15

\* All of these analyses were made in the analytical laboratory of the pigment plant at Baltimore, Md., of E. I. du Pont de Nemours and Co.

The "penalty in use" to pigment manufacturers because of the lower  $\text{TiO}_2$  content and the higher ferric iron of Brazilian ilmenite is serious, and reduces its competitive value appreciably.

A factor contributing to this lesser attractiveness of Brazilian ilmenite is the absence of altered grains, higher in  $\text{TiO}_2$ , which boost the average quality in both the Indian and Floridian sands. The brown opaque grains which are weakly or almost non-magnetic in Indian and Floridian sands assay 75 to 80 pct  $\text{TiO}_2$ , whereas in Brazil most of the brown grains are nothing but limonite, and only boost the objectionable content of ferric iron. The altered grains in Florida are described by Creitz and McVay.<sup>13</sup>

The Brazilian zircon is badly stained to a brownish color with organic matter, but this coating calcines off readily. Analyses showing 64 and 65 pct  $\text{ZrO}_2$  were made on laboratory concentrates indicating that the material is of acceptable quality. The usual trouble was experienced in reducing the  $\text{TiO}_2$  analysis of the zircon concentrate to acceptable limits, but this problem plagues producers from all sources and the Brazilian is no more difficult to clean than others.



The rutile is a good quality, and analyses of concentrates indicate that a grade of 95 pct TiO<sub>2</sub> could be attained regularly.

The monazite is low in ThO<sub>2</sub> as compared with Indian but carries as much of the other oxides. Analyses are as follows:

Source	Oxides of the Cerium Group	ThO <sub>2</sub>	P <sub>2</sub> O <sub>5</sub>	CeO <sub>2</sub>
Barra do Itabapoana	64.43	5.01	17.85	28.67
Barra do Itabapoana	64.50	5.23	15.57	28.81
Sai		6.89		30.04
<sup>a</sup> Travancore, India <sup>a</sup>	60.36	10.22	26.82	31.90
<sup>b</sup> Travancore, India <sup>d</sup>	61.73	8.65	26.50	Not Reported

<sup>a</sup> and <sup>b</sup> Analyses. <sup>a</sup> and <sup>b</sup> are reported by Houk,<sup>14</sup> table 4, p. 6 of the reference.

<sup>c</sup> Described as "sand from Travancore, India".

<sup>d</sup> Isolated from a concentrate from Travancore, India. No analyses as the content of cerium is reported as Ce<sub>2</sub>O<sub>3</sub>.

No concentrates were made of the corundum, nor of the aluminum silicates, although this problem should not be difficult since other gangue minerals of medium specific gravity and low magnetic permeability are present only in moderate amounts.

In grain size, the Brazilian sand is fairly coarse, as compared with Indian, and particularly with Floridian. Analyses shown in table I are typical.

Table I. Screen Analyses<sup>a</sup>

Ilmenite

Mesh	Barra do Itabapoana <sup>b</sup>						Guaratiba B-6	Cunhaú, Rio Grande do Norte
	S-1	V-2	E-3	T-2	P-9	B-7-12		
+48	11.92	12.89	55.78	10.53	8.64		32.1	
+60	6.30	8.71	1.22	7.93	8.91	4.6	22.4	18.73
+80	51.37	41.30	29.20	57.57	49.20	59.6	32.7	36.08
+100	10.87	17.36	9.65	10.63	14.87	19.6	9.6	29.04
+150	16.17	17.06	8.58	11.01	15.60	13.9	3.2	15.09
-150	3.35	2.69	1.37	2.27	3.00	2.3		0.99

Mesh	Zircon			Monazite			
	Barra do Itabapoana		Guaratiba	Mibra Commercial Shipment <sup>c</sup>	Barra do Itabapoana		Guaratiba
	B-2-29	B-7-14	B-6		B-7-13	B-2-26	B-6-11
+60		2.37	10.10	46.09	0.39		0.59
+80	24.0	25.72	11.47	37.59	16.14	15.07	2.76
+100	33.5	25.22	34.87	13.19	24.02	34.40	21.65
+150	38.2	40.70	37.50	2.85	42.91	46.88	70.07
-150	4.1	6.24	6.11	0.27	16.54	3.64	4.92

<sup>a</sup> Tyler standard sieves were used.

<sup>b</sup> Letter designates the particular deposit, number is the individual composite made.

<sup>c</sup> This analysis implies the concentration process used fails to recover the fine monazite.

### Concentration Problems

A number of "semi-works" tests on concentration of the Brazilian sand were conducted. Flowsheets following the Florida practice (ref. 6, p. 1063) are practical, in spite of the high tenor of the crude ore in heavy minerals. Preliminary gravity concentration in spirals produces a high-grade heavy mineral concentrate with over 90 pct recovery of these minerals. Electrostatic separation on machines of the Carpenter type separate conductors (ilmenite and rutile) from nonconductors, which are all the rest of the minerals. Ilmenite is separated from rutile on magnets of the induced roll type. Zircon is concentrated readily by respiraling the nonconductors, followed by final cleaning of the semi-

concentrate on electrostatic machines and on magnets.

The monazite can be recovered by a combination of gravity and magnetic methods, as in Travancore, and some encouraging but very preliminary tests were conducted on flotation of the monazite, for which credit should be assigned to J. C. Detweiler, Chief Chemist of the Humphreys Gold Corp., which operates the Florida deposits.

The only sand that gave trouble in concentrating was that from Sai, Barra do Riacho, and the delta of the Rio Dôce. The difficulty with these sands is because the ilmenite is mixed with much altered magnetite which has about the same magnetic permeability as the ilmenite.

### Acknowledgments

The writer made many friends in Brazil, and owes much to many of them. In his professional work he was aided greatly by Snr. Mario Pinto, of the Brazilian Department of Mines, and Dr. Glycon de Paiva, of the Brazilian Geological Survey. So many men, from general manager to office boy, in the company called "Industrias Quimicas Brasileiras Duperial, S.A." were helpful that it is difficult to mention one individually. Especial thanks go to his field assistants, Paul H. Klaustermeyer and the late David H. Graham, Professor Rui Franco of São Paulo University, and above all to his mentor and constant companion, Armando Lobo, of Belo Horizonte, whose knowledge of the country and friendly relations with one and all, wherever the party went, were of the greatest help. Mr. Emerson Brown, the U. S. Minerals Attaché in Rio; Vincent Burian and Johannes Holter, of Vitoria; and Boris Davidovitch, President of "Mibra," deserve the writer's sincere thanks for much help.

### References

- W. Melzer and G. H. Chambers: Zirconia in Brazil. *Footprints* (June 1941) 14, No. 1, 17-21 and 24-26.
- Ortho Leonardos: *Mineração do Rutile em Goyaz. Mineração e Metalurgica*. (Sept.-Oct. 1937) 194-199.
- G. H. Chambers: Brazilian Rutile Goes to War. *Footprints* (1942) 15, 3-8.
- Sylvia Froes Abreu: Titanium on the Coast of Espirito Santo. *Ministerio Agricola, Estação Experimental*. (1933) 64 pp.
- Jose Miranda: Areias Ilmeniticas no Brasil. *Mineração e Metalurgica*. (1943) 7, No. 40, 195-198.
- J. L. Gillson: Titanium. Chap. 47, pp. 1042-1073. 2nd Ed. (1949) Ind. Min. and Rocks. AIME.
- A. I. De Oliveira and D. H. Leonardos: *Geologia do Brasil*. 2nd Ed. Trans. Rio (1943) Serviço de Informação Agricola.
- A. K. Lobeck: *Geomorphology*. 350 (1939) McGraw-Hill Book Co.
- Ransome: U. S. Geol. Survey. *Bull.* 540 (1912) 152.
- C. W. Cooke: Scenery of Florida. Florida Dept. of Conservation. *Geol. Bull.* 17 (1939).
- J. H. C. Martens: Beach Sands Between Charles-town, S. C., and Miami, Fla. *Bull. G.S.A.* (Oct. 1935) 46, 1563-1596.
- Norman H. Fisher: The Heavy Mineral Deposits of the East Coast of Australia. *Trans. AIME* (1949) 181, 391; *Min. Tech.* (Nov. 1948) TP 2455.
- E. E. Creitz and T. N. McVay: A Study of the Opaque Minerals in Florida Dune Sands. *Trans. AIME* (1949) 181, 417; *Min. Tech.* (July 1948) TP 2426.
- L. G. Houk: Monazite Sand. U. S. Bur. Mines. *I.C.* 7233 (Feb. 1943) 19 pp.
- J. L. Overholt, G. Vaux, and J. L. Rodda: The Nature of Arizonite. *Amer. Mineralogist* (1950) 35, 117-119.

# Kaolin Production and Treatment in the South

by Paul M. Tyler

**Y**EAR after year, the kaolin industry of the United States has been setting new production records and making better products. High-grade paper, pottery, and rubber clays are produced in this country mostly in the South. Georgia alone contributes over 70 pct and South Carolina almost 20 pct of the total domestic output. Residual kaolin is mined in North Carolina, highly plastic but naturally sandy Tertiary (Eocene) potting clays are worked in north central

PAUL M. TYLER, Member AIME, is Consulting Mineral Technologist and Economist, Kensington, Md. AIME Tampa Meeting, November 1949.

TP 2857 H. Discussion (2 copies) may be sent to Transactions AIME before July 30, 1950. Manuscript received May 20, 1949.

Florida, and good white clays are produced in several other states, but the main sources of kaolin or china clay have been numerous deposits in the Tuscaloosa (Upper Cretaceous) formation. This formation of generally sandy sediments is called the Middendorf member in older geologic reports and corresponds in age with some of the New Jersey clays. As shown in fig. 1, it crops out almost continuously in a generally southwesterly direction across South Carolina and Georgia and into Alabama. Clay is mined from this formation in all three states but the principal producing centers lie within about 10 miles of a straight line drawn between Aiken, S. C., and a point about 10 miles south of Macon, Ga.

The white kaolins of the South were recognized and used prior to the Civil War but suitable treatment processes were not introduced until World War I when imports, chiefly from England, were curtailed. Although imports of high-grade clays were resumed after 1918, the domestic industry managed to treble its prewar production record during the early 1920's and has continued to grow. Whereas the 1909 to 1913 average total production in the United States was only 132,104 short tons valued at \$705,352 f.o.b. mines, the output in 1948 was 1,568,848 tons worth \$19,756,738.

Paradoxically, it seems in retrospect that the early failure of the American industry to meet foreign competition was due to the superior quality of our sedimentary clays in their natural state. Kaolin, of course, is the principal decomposition product of feldspars which originate in acidic igneous rocks such as granite, aplite, alaskite, granodiorite, quartz porphyry, etc. English china clays occur in residual deposits and before they can be marketed they have to be treated to remove accompanying quartz, mica, and other impurities. Notwithstanding the relatively crude methods employed, the final product is a beneficiated clay which is subject to a certain amount of technical control as to quality and uniformity. Although the naturally concentrated deposits in Georgia and South Carolina contain some of the finest crude white kaolin in the world, even it can be made better by suitable treatment.

In recent years well over half of the high-grade

china clay produced in the United States has been used in making paper. Some qualities of paper clays are still produced by the dry process, or air flotation, but the paper industry's specifications have grown so exacting that wet processing was adopted and more refined methods had to be perfected. Notwithstanding notable advances in clay-preparation technology during the past decade, or possibly because these advances have implemented and encouraged technologic changes in consuming industries, demand has grown for products of higher uniform quality than can be obtained from even the best natural deposits without rigidly controlled fractionation. Largely as a result of the wide adoption of machine coating for paper, the clay industry has been obliged not merely to eliminate virtually all mineral impurities but also to segregate the clay substance itself into narrow particle-size ranges.

By extraordinary coordination of sales effort and production technology, several Georgia companies manage to market a wide variety of specialized joint products but the commercial success of many producers depends upon their mining only the best parts of their deposits and then skimming the cream of this almost pure clay in order to obtain a maximum yield of kaolinite finer than about 2 microns in maximum particle size and possessing low viscosity as well as the more familiar attributes of suitable color and brightness, or reflectance.

To the casual visitor from another mineral industry, the kaolin mines and plants may appear to be

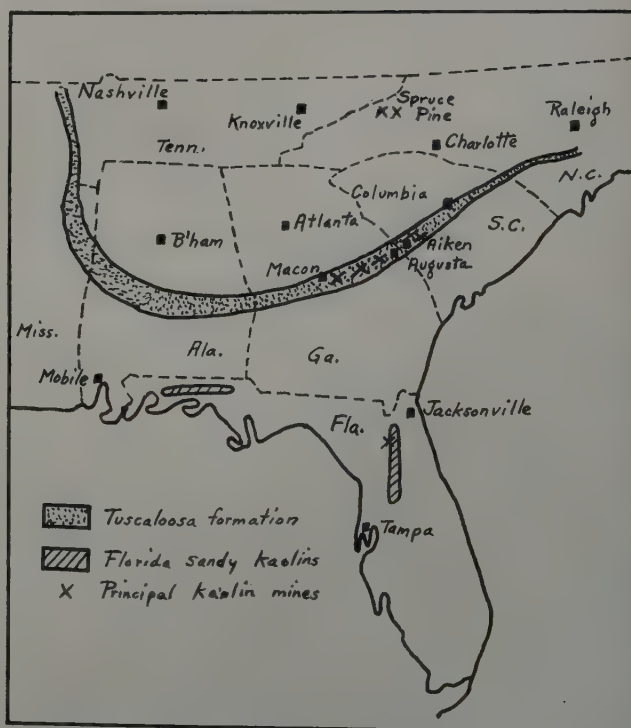


Fig. 1—Sketch map showing generalized outcrop of Tuscaloosa formation and other kaolin-bearing beds in southeastern U.S.



A general view of the kaolin industry of the country is given with special emphasis upon wet methods of beneficiation. Whereas dry milling procedures have become well-standardized, wet treatment technique differs widely and each company has its own flowsheet. Owing to the policy of secrecy in this highly competitive industry, plant descriptions are ruled out but methods and equipment employed by major producers are discussed for the principal treatment steps, crushing, blunging, grit removal, fractionating the clay, bleaching, filtering, and drying.

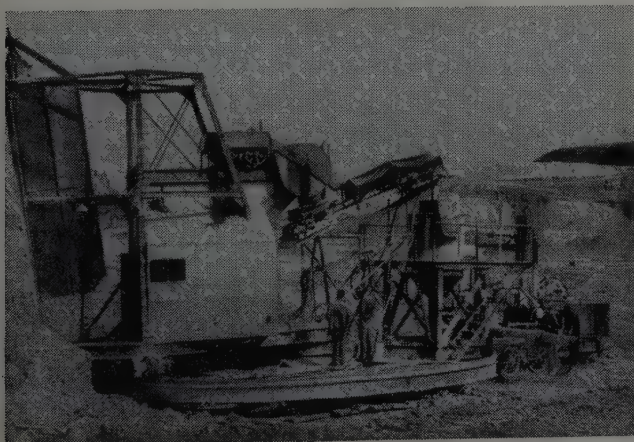
inefficiently operated and very wasteful of natural resources. However, it must be realized that the industry is in the throes of rapid development. Fundamental information is lacking on the nature and behavior of the several kaolin minerals. Consumers, too, often are confused as to specifications and cannot agree on suitable standards for making tests. Aided by new tools of modern research, some of the gaps in our knowledge are being filled, but certain of the difficulties in clay beneficiation are probably inherent. Many of them stem from the fact that colloidal or semicolloidal particles of the desired mineral have to be separated from other minerals and then sized to meet specifications that are not universally understood even in terms of performance in the consuming industries.

Apart from the emphasis on selectivity and avoidance of contamination with waste, kaolin mining does not differ greatly from other types of open-pit mining. Current practice in South Carolina and Georgia has been described in another paper.<sup>1</sup> Fig. 2 is a view in one of the large South Carolina pits. The only completely novel development is in Georgia. This is a continuous mining machine which digs the clay from the bank and delivers a dispersed slip or slurry to a pump and pipeline. This machine is



**Fig. 2 (above)—Mixing clay before loading into trucks.**  
Barden Pit, J. M. Huber Corp., Graniteville, S. C.

**Fig. 3 (below)—Continuous mining machine: crawler planer and mobile blunger.**  
Georgia Kaolin Co., Dry Branch, Ga.



really a combination of a shale planer and a blunger mounted on caterpillar treads and carefully coordinated. It is a great labor-saver but suffers from the disadvantage that it cannot be used in spotty deposits because it is less selective than a small power shovel or, as in earlier days, hand mining. Fig. 3 shows this machine ready to start a new cut.

Southern clays, as mined, are almost always rather acid. Before they can be completely dispersed the pH must be raised to around 6.5. Deflocculants are also added to stabilize the slip. Careful mining is necessary not only to avoid off color clay or excess grit but also to choose such portions of the clay in the bank as the drilling records indicate can be treated satisfactorily. Inclusion of even a small amount of a certain kind of clay, possibly bentonitic, will sometimes "poison" an entire batch. While methods may be worked out for correcting this, the best way to avoid trouble in the plant is to try to keep out the crudes that cause trouble.

Dry milling practice is becoming standardized. It is the only practice now employed in South Carolina where the principal product is rubber clay and is also used in Georgia for refractory kaolins, most potting clays, and to some extent for paper filler and other clays except paper-coating clays. The typical flowsheet is quite simple. Essential features are the large storage shed, the "Payloader" for reclaiming the crude clay from the concrete floor of the shed, the new clay shaver or "slicer" (described later) for shredding the moist clay, the oil-fired semidirect rotary drier (usually a Ruggles-Coles), two 5-roller, high-side Raymond mills with built-in whizzer attachments for fine grinding, and the Bates (St. Regis) valve bagger. For further details, the reader is referred to another recent paper.<sup>2</sup>

### Clay Treatment

Virtually the entire North Carolina output of kaolin is used in pottery, mainly table ware and high-grade floor and wall tile. The geology of these primary deposits and recent mining and treatment practice has been summarized by Stuckey<sup>3</sup> and flowsheets and descriptions of several plants have been published by Hubbell.<sup>4</sup> Mica is a remunerative by-product constituting 5 to 10 pct of the mine-run crude. It is recovered by tabling and froth flotation.

The Florida clay, intermediate in properties between a true kaolin and a ball clay, characteristically occurs below or only slightly above groundwater level. It is used entirely for pottery, electrical porcelain, and high-grade tile. The numerous known deposits, only two of which are currently worked, have recently been described.<sup>5</sup> The clay-bearing sands (containing less than 15 pct clay as mined) are stripped of overburden and then washed by hydraulic giants in front of a suction dredge which floats on the pond or artificial lake. The slurry, carrying about 12 pct solids, is neutralized and may be pumped to an Akins classifier floating on a barge in a separate pond as shown in fig. 4. Most of the associated white sand goes into this pond and can





**Fig. 4—Akins classifier on barge in sand pond.**  
United Clay Mines Corp., Hawthorne, Fla.

be reclaimed. The classifier overflow or clay slip is further purified before reaching the filter presses. The filter cake is often quite brown but the dry clay is almost white and fires white.

Present-day Georgia practice for washing kaolin (fig. 5) does not appear to be developing a standard pattern. The exchange of ideas which characterizes technologists in other mineral industries is lacking. Published research deals with properties of products not with processes employed in obtaining the products. A legitimate criticism is that many clay operators used to try to solve their problems by cut-and-try methods. This condition is being remedied and ideas are being borrowed from other process industries. The steps in preparing high-grade kaolin for market are few in number. Although some features are unique most of the basic operations have a counterpart in other mineral engineering fields. The



**Fig. 5—Clay washing plant.**

J. M. Huber Corp.,  
Huber, Ga.

following discussion is general in order to avoid disclosure of conditions at individual plants. Only when the information is generally known by a company's competitors can the equipment or methods used in a particular plant be identified.

The obvious starting point for any wet treatment process, of course, is a water suspension of the clay. Since hydraulic mining is not practiced in Georgia, the first step is to crush to sizes that are small enough to break down readily in the type of blunger equipment installed. A blunger is any device that will disintegrate the clay and disperse it in water so that sand and other impurities can be dropped out. Mechanical purification may be followed by chemical treatment to eliminate iron staining and, in the case of paper-coating clays, the kaolin particles themselves must be segregated into suitable size fractions. Finally, the various fractions must be thickened, filtered, and (usually) dried.

Dry clay is easily crushed. Formerly, it was common practice to air-dry the material in a storage shed, but this involves a good deal of labor and is inconvenient as well as costly. Clay as mined ordinarily carries over 20 pct moisture and will clog any ordinary jaw, gyratory, or roll crusher. Many small to medium-sized plants use a specialized grinding pan known as a "slicer." The perforated bottom plate of this machine carries 32 to 80 knives, each in a separate slot like the blade of a carpenter's plane. As the plate revolves the action is similar in principle to that of cabbage cutters used in the home kitchen for making cole slaw. On rainy days, if the

clay is unusually sticky, a little dry clay may be fed to absorb excess moisture. This machine handles up to 30 or more tons an hour, depending upon the diameter and number of knives, and reduces large lumps to less than 1-in. across. Figs. 6 and 7 show two types of these machines and fig. 8 shows raw clay being dumped into a machine from a payloader.

One of the largest companies stage-crushes lumps up to 6 ft in diam to a similar small size in two sets of Eagle crushers. This installation can handle 100 tons or more per hour. Powerful curved knives mounted on horizontal shafts chop the clay and feed it into the crushing zone. Similar machines are seen at brick-making plants. At other large plants, secondary crushing is done in the blungers. One company accomplishes the initial reduction with a combination of one corrugated and one smooth roll, another prefers a single-knobbed slugger roll opposed to a traveling breaker plate, still others use double rolls one of which is equipped with sharp cutting knives.

Blunger designs likewise vary greatly. As previously indicated several companies feed rather large lumps to the blunger which thus becomes a wet secondary crusher in addition to dispersing the clay in water with suitable electrolytes to form a creamy liquid or "slip." Horizontal blungers of the log-washer type may be used for this purpose; instead of paddles, however, they carry knives or curved beater arms. In another familiar type, beater

arms are mounted on a rapidly revolving vertical shaft. The vortex-type blunger seems to be especially effective providing the raw clay has been crushed fine enough beforehand. This type may be compared to a wet cyclone in which the settlings at the bottom of the cone enter the suction of a centrifugal pump and are recirculated in a tangential stream in the upper part of the chamber but below the overflow. Small lumps of clay are thus disintegrated partly in the pump and partly by abrasion as they are pressed against the walls of the settling chamber by the whirling action of the slurry. The blunger used in the continuous mining machine employed by the Georgia Kaolin Co. at its Dry Branch mine is of this type. It delivers a smooth slip which is pumped to the sand-removal plant. Pebble-mill disintegrators would seem to have certain theoretical advantages but apparently are not

**Fig. 6—View from top of 80-knife clay slicer.**

Champion  
Paper and  
Fiber Co.  
plant, Sand-  
ersville, Ga.



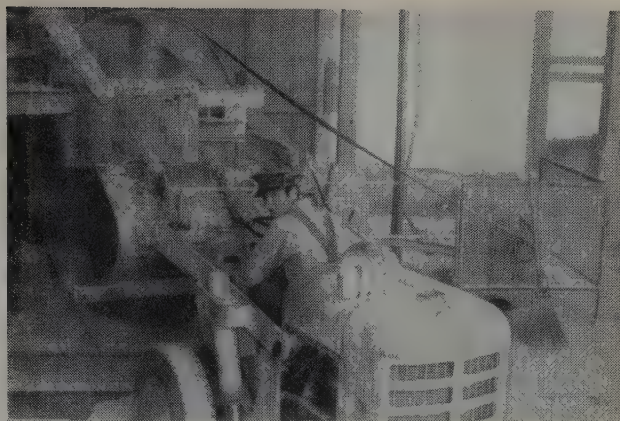


used in Georgia plants, possibly because they might pulverize some grit.

Edgar Brothers Co. and the Georgia Kaolin Co. blunge their clay at the mines and deliver the slip, after removing grit, to their respective treatment plants several miles away through pipelines. The former company pumps material carrying 20 pct solids while the latter handles a 40 pct slip without difficulty. Other companies deliver raw clay to their plants in trucks or, in one case, by narrow-gauge steam train.

In accordance with English china clay practice, several companies conduct their slip through a series of sand boxes and mica troughs to remove grit. The density of the slip averages between 20 and 30 pct solids. Retention time in the system is about 30 min. In order to sluice out grit at intervals, usually once a shift, two or more series of troughs must be provided. The settlings are run to waste except at the extreme end where a separation of the coarsest clay fractions may be made. It is generally considered uneconomical to reblunge and resettle the clay to recover that entrapped in the sand settlings. Sand drags (flight conveyors) are used in two or more plants (fig. 9). Vibrating screens to remove tramp wood and to retain particles 200 mesh or over are installed at the discharge end of the drags. Several plants employ screens to remove the bulk of grit; hydroseparators are used for grit removal and also for fractionation. The material in the hydroseparators overflow may be fine enough for paper-coating clay, in which case the underflow is reblunged, resettled, and the second overflow containing coarser clay fractions is recovered for paper filler and other uses.

Two companies, The Georgia Kaolin Co. and Edgar Brothers Co., are licensed to use the Bird continuous centrifuge. The general design of this machine is well-known and certain of its applications have been sufficiently described.<sup>5</sup> Recent improvements have simplified construction, lengthened the period of operation between shutdowns for repairs, and facilitated controls without modifying basic principles. Its action is in harmony with Stokes' law except that the separatory force may be as much as 1000 times gravity. For certain uses, the clay passes through the Bird machine only once, thereby removing fine grit and possibly some of the coarsest particles or flocs of clay. To produce quality



**Fig. 8—Dumping raw clay into slicer from payloader.**

National Kaolin Products Co., Aiken, S. C.

paper-coating clays, however, the slurry is passed to a second centrifuge which rejects all particles over, say, 2 microns. Clean-cut separations may be made as low as 0.5 micron, if required.

The quantity of grit removed varies at different mines and in different parts of the same mine but the average in Georgia wet-treatment plants may be as much as 15 pct of the dry weight of raw clay treated. This compares with a rejection of about 1 pct at many South Carolina dry-treatment plants where only certain high-grade clays are mined and where there is probably some pulverization of quartz. In both states, the bulk of the grit is composed of extremely fine siliceous sand. Mica, however, may comprise as much as 30 pct of the total and minor amounts of other minerals may be identified. Recovery of the fine mica has been considered, appears practicable, but has not yet been accomplished commercially. Although presumably somewhat sericitic it possesses a fairly good sheen and general appearance. One reason for the retention of mica troughs at several plants, apart from their low initial capital cost, is the traditional argument that while they are prodigiously wasteful of labor and good clay, they do remove mica effectively. Mica flakes more nearly conform to Stokes' law in a slowly flowing stream than in a rising current type of classifier. Except perhaps in the Bird centrifuge, fine mica tends to contaminate certain clay fractions. One would suppose that it could be easily and profitably removed by froth flotation but certain operators insist that it is something of a problem.

Since few clays are white enough to be used for coating fine paper, even after being freed from grit, it is generally necessary to bleach them. This is a batch operation conducted in large wooden tanks. Processing data are carefully guarded but it is no secret that bleaching is ordinarily performed in a slightly acid (e.g. pH 4.6), moderately dilute pulp using a small amount of zinc hydrosulphite freshly prepared from sulphur dioxide and zinc dust. Alum has to be added to complete the reaction. The  $\text{SO}_2$  is usually purchased from a single supplier who contributes technical advice. About 2 or 3 lb of the gas are used per ton of clay. Yellow or brown iron colorations (limonite?) yield readily to this treatment whereas pink or red stains (hematite?) are often difficult, if not impossible, to bleach out. Gray tones may sometimes be removed by suitable oxidizing agents but this type of treatment is seldom required on Georgia clays.



**Fig. 7—  
Another  
type of  
clay slicer.**

National  
Kaolin  
Products Co.,  
Aiken, S. C.



Conventional plate-and-frame filter presses, following thickening in concrete tanks, are standard at most clay-washing plants. Due to the high cost of labor and filter cloths, however, many believe that they are on the way out. Thickening offers no problems; if not already acid from the bleaching step, the pH is lowered by adding alum or sulphuric acid to cause flocculation. String filters are employed at one or more plants and several companies have been using continuous rotary vacuum filters of special design. Despite their high initial cost the latter seem to be gaining ground due to their extremely low labor requirements and large rated capacity. Although used principally on paper-filler clay, because this is somewhat coarser grained as well as a larger tonnage item, they are also said to be successful on the best grades of paper-coating clay. For use in the clay industry, however, these machines have to be built to order. Information as to optimum pulp densities, the best kind of weave for the filter cloth (preferably rayon), and types of cake-stripping mechanisms cannot be published.

Filter cakes contain about 30 pct moisture and may be dried in various ways. Proctor and Schwartz tunnel driers with perforated aprons may be employed either ahead of a rotary drier or alone to produce the finished product. Gas heat is preferred but is not generally available so steam pipes are used. The most common type of drying equipment is the indirect-fired rotary drier but a tray-type or a Sargent drier may be observed now and then.

For some uses clay is shipped "bone dry" (under 1 pct moisture); on the other hand some consumers accept material carrying 10 pct moisture. Paper-filler clay usually carries about 6 pct moisture and coating clay about 5 pct. Potting clays may be treated with measured amounts of Calgon or similar dispersant and sprayed into a Buffalo-vac drum drier, thereby relieving the pottery maker from the necessity of adding chemicals to prepare the slip.

In recent years some paper clay has been shipped in tank cars in the form of low-viscosity (e.g. 225 centipoises) slip containing 70 to 72 pct solids by weight. This product is made by agitating the filter cake with deflocculants; the resulting mobile liquid will stand for weeks without segregation. This mode of shipment simplifies operations at the paper mill and any additional cost for chemicals at the clay plant is offset by not having to put the material through a drier. Railroad freight increases, however, have penalized this practice so that it now seems destined for abandonment. The water content adds over one-third to the already high cost of shipping the clay from Georgia to mills in Wisconsin, Maine, or the Pacific Northwest.

A whole paper might well be written on the increased instrumentation and the development of more and more automatic controls in kaolin plants. Throughout all operations, the pressure on labor economy and products standardization have greatly expanded the number and variety of records to be kept. Every car of paper clay needs to be sampled and tested for color, acidity, moisture, grit, particle size distribution, and viscosity. But this is only the final check. Additional tests are made all along the production line, beginning before the clay in the bank is even stripped and continuing throughout the mining, blunging, and succeeding steps. A daily inventory of clay in process is taken by gauging tanks and pipelines. Owing to variable moisture content simple measurements of tonnage or volume

Fig. 9—  
Sand drags.

Georgia  
Kaolin  
Co., Dry  
Branch, Ga.



of raw clay crushed or blunged are not particularly accurate but once the clay is properly dispersed its progress is readily followed by first metering the water added and then registering the flow and specific gravity of the slip at strategic points.

Chemical analyses of raw Georgia kaolins show them to be highly pure. After washing and bleaching the principal impurity is about 1.75 pct of  $\text{TiO}_2$ . The question naturally arises whether the brightness of clay might not be further enhanced by removing this impurity. The measure of brightness in the paper-clay trade is the percent of light reflected at a certain spectral wave band. Tested in a color analyzer, the best clays show excellent reflectance at the red end of the spectrum but rather poor reflectance in the blue and green range. Bleaching, by eliminating much of the effect of iron, raises the reflectance materially and enough work has been done to show that removal of the titania would raise it even more. Under certain circumstances this can be accomplished by froth flotation but there may be other ways. On some crudes, however, even the laboratory method of electrodialysis does not work.

A certain amount of experimentation has been conducted on the froth flotation of kaolins. It is often possible to remove a considerable amount of off-color material from a batch of clay without making any significant improvement in the properties of the remaining clay. However, more experimentation with flotation is definitely indicated, using both anionic and cationic reagents. Electrophoresis, or electro-osmosis as hitherto practiced in Germany and elsewhere, has not resulted in any substantial purification traceable primarily to selective deposition or migration caused by the passage of the electric current. Electrophoresis (also called cataphoresis) usually amounts to simple dewatering; its use with the Bird centrifuge has been patented.

## References

- <sup>1</sup>P. M. Tyler: Kaolin Mining in the South. *Min. Cong. Jnl.* (1949) **35** (6) 31-34.
- <sup>2</sup>P. M. Tyler: Modernizing Dry Kaolin Milling in South Carolina. *Eng. and Min. Jnl.* (1949) **150** (6) 56-58.
- <sup>3</sup>J. L. Stuckey: Kaolins of North Carolina. *Trans. AIME* (1947) **173**, 47-54; *Min. Tech.* (July 1947) TP 2219.
- <sup>4</sup>A. H. Hubbell: Mining and Washing Kaolin in Western North Carolina. *Eng. and Min. Jnl.* (1943) **144** (1) 51-55.
- <sup>5</sup>J. L. Calver: Florida Kaolins and Clays. *Fla. Geol. Survey. I. C. No. 3* (1949) 59 pp. (mimeo.)
- <sup>6</sup>S. C. Lyons and A. L. Johnson: Continuous Centrifuge in the Mineral Industry. *Trans. AIME* (1947) **173**, 252-262; *Min. Tech.* (July 1947) TP 2195.
- <sup>7</sup>G. I. Adams: Origin of the White Clays of Tuscaloosa Age (Upper Cretaceous) in Alabama, Georgia, and South Carolina. *Econ. Geol.* (1930) **25** (6) 621-626.



# Some Properties of Pseudowavellite from Florida

by W. L. Hill, W. H. Armiger, and S. D. Gooch

The physical properties, chemical behavior under thermal treatment, and fertilizer value of fluorine-containing pseudowavellite (hydrous calcium aluminum phosphate) that occurs as phosphate clay admixed with quartz sand in an extensive deposit near Bartow, Fla., were determined by experiments in laboratory, pilot-plant, and greenhouse. When the mineral was heated at 500°C, 80 pct of the phosphorus became citrate soluble. The heated product, however, deteriorates markedly under moist conditions such as obtain in the soil.

EXTENSIVE beds of a soft aluminiferous phosphate lie in the Bartow-Pembroke region of Florida. Although the extent of the deposit is unknown, it is thought that the equivalent of at least five million tons of material containing 25 pct  $P_2O_5$  exists on one large mine property alone, and soft phosphates similar in nature are known to occur elsewhere in the state. This aluminum phosphate, unlike fluorapatite, is rendered fairly soluble in neutral ammonium citrate solution merely by heating it at a temperature below 600°C, a property made the basis of a process patent by the junior author.<sup>1</sup> The possibility of producing an available form of phosphorus from what has hitherto been a worthless natural phosphate prompted the Bureau of Plant Industry, Soils, and Agricultural Engineering to make a brief study of the Pembroke phos-

phate during the war years with a view towards determining the optimal temperature for heat treatment and the reactions involved in the process. The results of this study supplemented by data and information furnished by the junior author are presented in this paper.

**Description of Natural Phosphate:** The phosphate with admixed quartz sand and siliceous matter occurs as a low-grade (8 to 9 pct  $P_2O_5$ ) clay-like material that can be beneficiated cheaply by making a water separation of the very fine material from the sand, floating off the fine material, settling the suspension to a thick slurry, and finally dewatering the slurry by vacuum filtration followed by drying. The product dried at 100°C contains about 16 pct water and 25 pct  $P_2O_5$ , about 1/12 of which is citrate soluble. Partial analyses of representative samples of the crude ore and corresponding concentrate are given in table I.

The concentrate is light yellow in color and even after being dried at 100°C sticks very noticeably to laboratory grinding and screening equipment. Under the microscope the ground material is predominantly light brown crystalline aggregates with index of refraction near 1.61. The principal phosphate mineral constituent was identified as pseudowavellite with admixed phosphosiderite.<sup>2</sup> Pseudo-

W. L. HILL and W. H. ARMIGER are Senior Chemist and Agronomist, respectively, Bureau of Plant Industry, Soil, and Agricultural Engineering, Agricultural Research Administration, U. S. Department of Agriculture, Beltsville, Md., and S. D. GOOCH is General Manager, Pembroke Chemical Corp., Pembroke, Fla.

AIME Tampa Meeting, November 1949.

TP 2860 H. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Sept. 26, 1949.

**Table I. Analyses of Natural Phosphate and Products Experimentally Derived from It in Pilot-Plant Operation**

(Results are on basis of samples as received)

Sample No.	Type of Material	Total $P_2O_5$ , Pct	Citrate-soluble $P_2O_5$ , Pct of Total	CaO, Pct	$Al_2O_3$ , Pct	$Fe_2O_3$ , Pct	F, Pct	Moisture 105°C, Pct	Ignition <sup>a</sup> Loss 800°C, Pct
2334	Ore	8.67		5.40			0.73	0.40	4.05
2335	Concentrate	24.92	7.9	8.92	29.04	3.80	1.19	1.00	15.06
2336	Heat-treated concentrate	28.20	48.5	10.25	32.90	4.42	1.34	0.73	3.59

<sup>a</sup> Exclusive of moisture at 105°C. Result includes a small amount of organic matter and some volatilized fluorine, but only a trace of carbon dioxide.

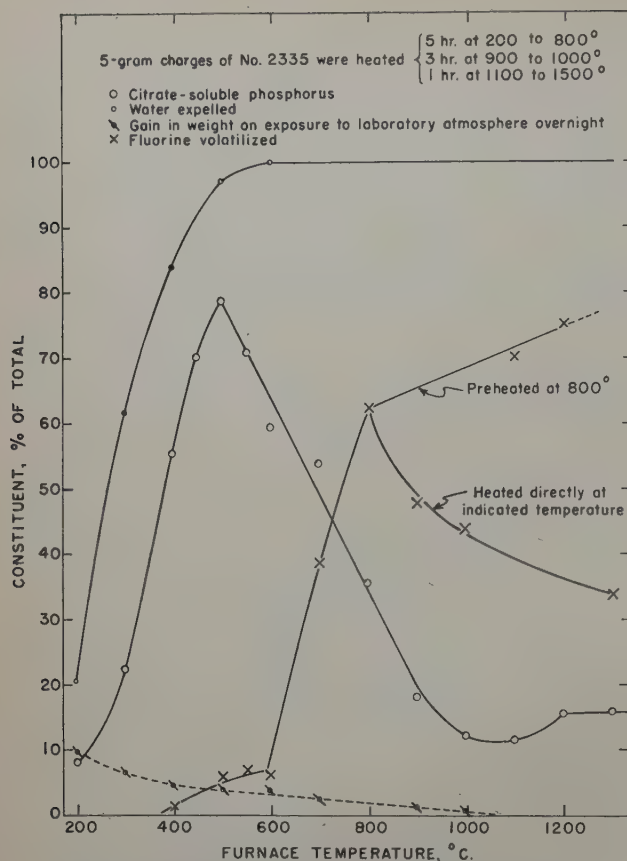


Fig. 1—Effect of ignition on composition and citrate solubility of concentrated phosphate.

wavellite is a hydrous calcium aluminum phosphate having the composition<sup>5</sup>  $\text{CaAl}_3(\text{PO}_4)_2(\text{OH}) \cdot \text{H}_2\text{O}$ \* or

\* The formula given by Larson and Shannon<sup>3</sup> and by English<sup>4</sup> is  $5\text{CaO} \cdot 6\text{Al}_2\text{O}_3 \cdot 4\text{P}_2\text{O}_5 \cdot 18\text{H}_2\text{O}$ . Furthermore, pseudowavellite and deltaite,  $\text{Ca}_2\text{Al}_2(\text{PO}_4)_2(\text{OH}) \cdot \text{H}_2\text{O}$ , are isostructural and probably isomorphous,<sup>6</sup> both being uniaxial and positive with a rhombohedral unit cell. Indices of refraction of three specimens each are: pseudowavellite 1.618 and 1.623, 1.619 and 1.627, 1.622 and 1.631; deltaite 1.641 and 1.650, 1.630 and 1.640, 1.621 and 1.629.

$4\text{CaO} \cdot 6\text{Al}_2\text{O}_3 \cdot 4\text{P}_2\text{O}_5 \cdot 14\text{H}_2\text{O}$ . If in the case of the concentrate (table I) iron and fluorine are disregarded and a figure for water is derived by correcting the total loss on ignition for volatilized fluorine, the composition corresponds with  $3.35\text{CaO} \cdot 6\text{Al}_2\text{O}_3 \cdot 3.70\text{P}_2\text{O}_5 \cdot 18.4\text{H}_2\text{O}$ . From this it is seen that in comparison with the formula derived from a structural analysis for pseudowavellite,<sup>5</sup> the sample is deficient in both calcium and to a less extent phosphoric oxide, but carries excess water.

**Description of Heat-treated Concentrate:** In the process for converting the concentrated phosphate into available phosphate,<sup>1</sup> the filter cake (about 55 pct solids) is heated at about 550°C in a rotary kiln. The resulting nodular product is cooled and ground to approximately 200-mesh. The ground material contains 27 to 28 pct  $\text{P}_2\text{O}_5$ . The citrate solubility varies with the temperature of treatment and with age and storage conditions, which are discussed later in this paper. A partial analysis of a typical sample is given in table I.

The heat-treated material has a somewhat deeper color than the concentrate dried at 100°C, is practically insoluble in water, and yields a suspension with pH 6.5 to 7.5. The freshly prepared product is an amorphous material with the exception of a scattering of stable minerals, mainly quartz. In nodular form it packs to a bulk density of 52 lb per cu ft,

whereas the ground material packs to 42 lb per cu ft. The specific gravity of the powder, determined by the pycnometer method, is 2.56. The freshly prepared material carries practically no water and will remain workably dry after the addition of 20 to 25 pct water.

**Laboratory Experiments:** The concentrate in 5-g charges, No. 2335 (table I), ground to pass an 80-mesh sieve, contained in 50-cc platinum dishes, was heated at a series of temperatures in the range 200° to 1400°C in a well-ventilated muffle furnace for 1 hr or longer. The loss in weight on heating was noted, and the tendency of the heated material to gain weight from the atmosphere was determined by exposing the charge to the atmosphere overnight. The material was then put through an 80-mesh sieve and stored in a screw-cap bottle. Fluorine and citrate-insoluble phosphorus were determined on the heated products. Fluorine was determined by distilling an 0.5-g sample with perchloric acid, having due regard for the very slow rate of release of the last quarter of the fluorine from materials of this type, and titrating the distillate with thorium nitrate solution.<sup>6</sup> Citrate-insoluble phosphorus was determined with the use of 1-g samples in accordance with the official method for determining available phosphorus in fertilizers.<sup>7</sup> The citrate-insoluble residue was washed with 5 pct ammonium nitrate solution in order to minimize the tendency of the phosphate to run through the filter. Despite every precaution to keep the time and other factors constant, however, results of duplicate determinations often differed by as much as 0.8 pct. Contrary to the usual behavior of phosphates the "solubility" of the heated material in neutral ammonium citrate solution increases with the size of the sample (table II). A similar observation on aluminum phosphate from the Connetable Islands was made some years ago.<sup>8</sup>

**Effects of Heat Treatment on Phosphate Concentrate: Citrate-soluble Phosphorus:** The citrate-solubility of the phosphorus increased rapidly with the temperature at which the material was heated (fig. 1) up to 500°C, where the solubility reached nearly 80 pct of the total phosphorus, then fell off with further increase in the heating temperature to less than 20 pct at 900°C and remained between 10 and 20 pct in the temperature range 900° to 1400°C. Strong sintering of the charge began at 900°C. The material appears to be similar to the aluminum phosphate from Redonda Island studied by Morse<sup>9</sup> which showed a maximal solubility at 560°C. It is, however, unlike the calcium-free aluminum phosphate from Connetable Island,<sup>8</sup> which loses substantially all water of hydration at 105°C and attains a solubility around 60 pct that persists up to ignition temperatures as high as 800°C.

**Expulsion of Water:** The water was completely expelled at 600°C (fig. 1) and about 95 pct had been driven off at 500°C where the optimal solubility occurred. Up to this temperature the data

Table II. Effect of Size of Sample on the Result for Citrate-soluble Phosphorus

Sample No.	Type of Material	Total $\text{P}_2\text{O}_5$ , Pct	Citrate-soluble $\text{P}_2\text{O}_5$ in Per Cent of Total Determined on Sample			
			0.5 g	1.0 g	1.5 g	2.0 g
2335	Concentrate	24.92	5.3	6.0	2.8	2.4
2336	Heat-treated concentrate	28.20	44.2	52.4	57.1	58.0

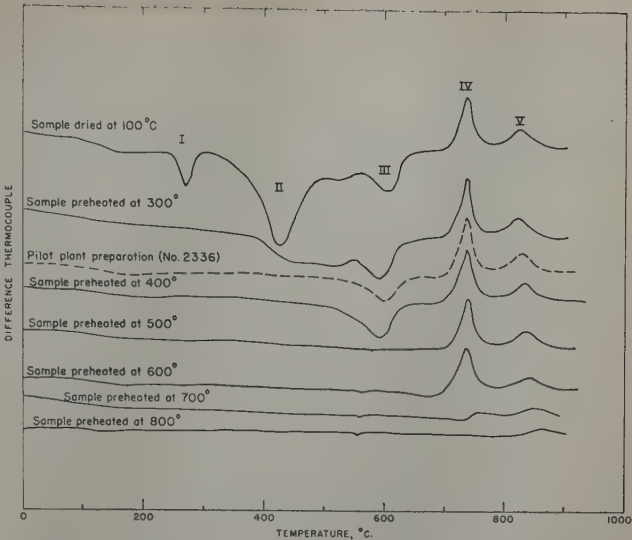


indicate a linear relationship between the citrate solubility of the phosphorus and the proportion of water expelled. Materials heated at the lower temperatures showed a marked tendency to gain weight on exposure to the atmosphere, a fact that is illustrated by the dotted curve in fig. 1. These figures were obtained by exposing the undisturbed charge; larger gains would probably be attained if the charge were stirred several times during exposure.

**Fluorine Volatilization:** Evolution of fluorine began at about 400°C (fig. 1), increased slowly with temperature to 7 pct of the total fluorine at 600°C, then very rapidly to about 63 pct at 800°C, where the material began to sinter, and either increased slowly or dropped off with further increase in temperature depending upon whether the charge was preheated at 800°C or heated directly at the indicated temperature. The small volatilization of fluorine below 600°C can hardly be a factor in the development of citrate-soluble phosphorus.

**Reactions:** Thermal-analysis curves (fig. 2) indicate that five reactions occur within the temperature range 200° to 900°C. Comparison of the heating curves with the dehydration data (fig. 1) shows that the first two reactions involve water loss. The first is presumed to be the loss of water of crystallization (around 200°), whereas the second involves the expulsion of hydroxyl water (around 400°). At a somewhat higher temperature fluorine, which replaces hydroxyl water, is rapidly expelled. This is doubtless the third reaction. The exothermic reactions (IV and V) are presumed to correspond to a crystallization of the amorphous material formed at lower temperatures. The latter view is supported by X ray powder diffraction photographs of the heated materials, which show that the material becomes amorphous at 400°C and that crystallization begins at about 600°C. One of the compounds formed in this crystallization is apatite, which was present in material heated at 800°C and had become a prominent phase in material heated at 1000°C. Finally, it will be noted that the maximal citrate solubility of the phosphorus occurs in the temperature range where the material is almost completely amorphous.

**Deterioration of Heated Material:** The heat-treated concentrate can be cooled by quenching the hot furnace charge in water without affecting the solubility of the promptly dried (100°C) product. Neverthe-



**Fig. 2—Heating curves of concentrated phosphate (No. 2335).**  
Rate of heating was 12°C per minute.

less, the citrate solubility undergoes a gradual decline with time. For example, a batch prepared in August and stored in paper bags contained better than 20 pct of citrate-soluble P<sub>2</sub>O<sub>5</sub>, which had declined to 18 pct by October, and to 15.5 pct by the following March. A similar, though slower, reversion has been observed in heat-treated aluminum phosphate from Redonda Island.<sup>9</sup> In the present case deterioration was observed even in material that was stored in a bottle and sealed to prevent the ingress of moisture. In view of the fact that the increased solubility occasioned by heat treatment is due primarily to the conversion of a crystalline material to the amorphous condition, the loss in solubility in the absence of water might be attributed to a spontaneous crystallization of compounds that are less soluble than the amorphous parent material. This possibility was not investigated.

Although directly comparable results are not at hand, evidence points to an accelerated decline in solubility when the material is kept wet with water. Thus, two laboratory preparations obtained by heating a concentrate at 450° and 600°C, respectively, that contained around 20 pct citrate-soluble P<sub>2</sub>O<sub>5</sub>

**Table III. Yields of Millet in Greenhouse Experiments with Heat-treated Concentrate and Double Superphosphate**

Applied Phosphate	P <sub>2</sub> O <sub>5</sub> , Pct		Average Dry Weight of Plants <sup>a</sup> in Grams per Pot of Total P <sub>2</sub> O <sub>5</sub> Applications of			
	Total	Solubility <sup>b</sup>	50 lb Per Acre	100 lb Per Acre	200 lb Per Acre	400 lb Per Acre
<b>Experiment I on Evesboro sandy loam<sup>c</sup></b>						
Concentrate No. 2335	24.9	7.9			1.1 <sup>d</sup>	
Heat-treated concentrate No. 2336	28.2	48.5	4.0	8.6	11.2 <sup>d</sup>	
Double superphosphate	48.7	96.0	13.1	19.3	19.7 <sup>d</sup>	
<b>Experiment II on Evesboro sandy loam soil<sup>c</sup></b>						
Double superphosphate	48.7	96.0	3.11	3.69	5.63	6.25
Heat-treated concentrate No. 2476	28.9	79.6	1.11	1.77	2.51	3.22
Same reverted by wetting (No. 2477)	28.9	63.7	0.98	1.58		

<sup>a</sup> Differences in average dry weights required for significance are:  
Experiment I—1.75 at 5 pct level (19:1 ratio) and 2.65 at 1 pct level (99:1 ratio).  
Experiment II—for 50 and 100 lb rates, 0.80 at 5 pct level and 1.05 at 1 pct level;  
for 50 to 400 lb rates, 0.85 at 5 pct level and 1.12 at 1 pct level.  
<sup>b</sup> Per cent of total P<sub>2</sub>O<sub>5</sub>; neutral ammonium citrate method.  
<sup>c</sup> Average dry weight of plants without applied phosphorus was 1.3.  
<sup>d</sup> Rate of application was 150 lb per acre.  
<sup>e</sup> Average dry weight of plants without applied phosphorus was 0.44.

(fig. 1), were subsequently moistened, kept wet for 48 hr and then dried at 105°C. This treatment increased the water content (+105°C) of the samples by 3.1 and 1.8 pct, and lowered the soluble phosphorus by about one fifth of the total phosphorus. Accordingly, it is obvious that these wet samples deteriorated within 48 hr to approximately the same extent as was observed over a period of seven months in the dry material cited above.

**Fertilizer Value of Heated Concentrate:** The fertilizer value of a concentrate, two heat-treated concentrates, and a reverted material, in comparison with double superphosphate, was determined in greenhouse experiments, in which millet was grown on soil that responded to phosphate applications. The greenhouse procedure is described elsewhere.<sup>10</sup> In experiment I, planted February 26 and harvested April 9, 1945, millet was grown on Evesboro sandy loam soil (pH 5.0) from the Beltsville Research Center, Beltsville, Md. In experiment II, planted March 7 and harvested April 11, 1949, millet was grown on Evesboro sandy loam soil (pH 4.8) from Beltsville, Md. In both experiments the soil was limed to give a pH of about 6. Four replicates were provided for each treatment. The average yields are given in table III.

The heat-treated material, though greatly superior to the raw concentrate, was in every case inferior to double superphosphate and the differences between yield responses are highly significant. The responses to the heat-treated material in comparison with those to superphosphate are somewhat lower than would be expected on the basis of the citrate solubilities. The response ratios are around one fourth to one third in comparison with a solubility ratio of about one half, a situation that could well be attributed to deterioration of the heat-treated material as it lies in the moist soil. Experiment II provides a comparison of a heat-treated product with a portion of the same that had been reverted by keeping it wet for 48 hr in the laboratory. Although the yield responses to the reverted

material are in both instances lower, the differences are not statistically significant. Apparently deterioration occasioned by the long contact of the heat-treated material with moist soil all but obliterated the effect of any initial difference in availability of the phosphorus to the growing plants.

#### Acknowledgment

The authors are indebted to S. B. Hendricks and J. G. Cady, of the Bureau of Plant Industry, Soils, and Agricultural Engineering, U. S. Department of Agriculture, for the optical examination of the laboratory preparations, and to R. A. Nelson and Cecil Pinkerton, formerly of the Bureau, for the differential thermal analyses and chemical analyses, respectively.

#### References

- <sup>1</sup>S. D. Gooch: Fertilizer. U. S. Pat. 2,395,219. (Feb. 19, 1946).
- <sup>2</sup>C. L. Christ: Private communication to S. D. Gooch.
- <sup>3</sup>E. S. Larsen and E. V. Shannon: The Minerals of the Phosphate Nodules from Near Fairfield, Utah. *Amer. Mineral.* (1930) **15**, 307.
- <sup>4</sup>G. E. English: Descriptive List of New Minerals 1892-1938. 1st ed. 258 pp. (1939) New York.
- <sup>5</sup>E. S. Larsen, 3rd.: The Mineralogy and Paragenesis of the Variscite Nodules from Near Fairfield, Utah. *Amer. Mineral.* (1942) **27**, 281, 350, 441.
- <sup>6</sup>D. S. Reynolds and W. L. Hill: Determination of Fluorine with Special Reference to Analysis of Natural Phosphates and Phosphatic Fertilizers. *Ind. and Eng. Chem.* (1939) **11**, 21.
- <sup>7</sup>Association of Official Agricultural Chemists: Official and Tentative Methods of Analysis. 6th ed. 932 pp. (1945). Washington.
- <sup>8</sup>R. P. Bartholomew and K. D. Jacob: Availability of Iron, Aluminum, and Other Phosphates. *Jnl. Assn. Official Agr. Chem.* (1933) **16**, 598.
- <sup>9</sup>F. W. Morse: The Effect of Moisture on the Availability of Dehydrated Phosphate of Alumina. *Jnl. Amer. Chem. Soc.* (1903) **25**, 280.
- <sup>10</sup>W. L. Hill, F. N. Ward, W. H. Armiger, and K. D. Jacob: Composition and Fertilizer Value of Phosphate Rock-Magnesium Silicate Glasses. *Jnl. Assn. Official Agr. Chem.* (1948) **31**, 381.

#### Correction

In the January 1950 issue: TP 2629 H. *Chromite and Other Mineral Occurrences in the Taştepe District of Eskişehir, Turkey* by Ferid Kromer, p. 108, under the subheading "Mineral Occurrences: Chromite," the last sentence of the first paragraph should read, "However, a new lode, very recently plotted, in the Taştepe mine averages 50 pct Cr<sub>2</sub>O<sub>3</sub>, 4.6 pct SiO<sub>2</sub>, and 14 pct FeO." The author in making the correction added the following information: The production of about 4000 tons which came from this new lode has given the following analysis: 48.19 pct Cr<sub>2</sub>O<sub>3</sub>, 13.36 pct FeO, and 7.82 pct SiO<sub>2</sub>. All the production went to Austria, on the basis of the above analysis.



# An Electronic Tramp Iron Detector for Ore Conveyor Belts

(With Discussion)

by C. M. Marquardt

**Tramp iron and steel moving on a conveyor belt cause small currents to be generated in a coil situated in a strong magnetic field, which are converted to an alternating current and are amplified. The output voltage from the amplifier fires a thyatron with a relay in its anode circuit, which actuates a howler and simultaneously drops a spot of marker material on the belt.**

**T**HE problem of tramp iron removal from moving ore belts is a long standing one. When heavy ore streams are carried on a belt, magnetic pulleys and strong surface magnets fail to remove tramp buried in the ore stream. Tramp iron buried in the ore stream cannot be pulled from the bed by surface magnets and is carried past the magnetic pulley by the falling ore stream.

With the increased use of detachable bits the problem of the detection and removal of tramp has become more necessary and difficult.

Several types of tramp detectors have been developed. It would serve no practical purpose here to review exhaustively the literature on the subject. Methods of tramp detection used are: (1) The magnetic method, wherein the small current generated by the magnetized tramp passing a coil is used. (2) The unbalanced oscillator method wherein the tank circuit of a stable radio- or audio-frequency oscillator is unbalanced causing a change of frequency or a change in plate current due to changes in eddy currents, hysteresis, or dielectric because of the

presence of tramp. (3) The bridge methods wherein the impedance of one leg of an alternating current inductive bridge is changed due to the presence of conducting tramp.

Each of these methods of tramp detection has its field of usefulness. The second and third methods are excellent for oxide iron ores, coal, sand and gravel, grain, etc. However, these latter methods are not suitable for use on ores that contain rich conducting sulphides, since a large piece of pyrite or galena will cause the same detector response as a piece of tramp iron.

At the Caselton, Nevada, plant of the Combined Metals Reduction Co., sulphide ores are treated; therefore, it was necessary that a tramp detector operating on only the magnetic properties of the tramp iron be used.

Tramp iron detectors of this type operate on the principle that if a piece of tramp iron moves near a coil in a high intensity, steady magnetic field, the presence of the moving tramp iron causes the magnetic flux lines through the coil to change. This induces a small current in the coil. In series with the coil is generally placed a relay sensitive to currents of 1 to 3 microamp. When the tramp iron passes near the coil the flux linkages change, causing this very sensitive relay to actuate. Since the contacts of such a relay are very small it must actuate a second relay capable of breaking a larger current

---

C. M. MARQUARDT, Member AIME, is *Electronic Engineer, Combined Metals Reduction Co., Salt Lake City, Utah.*

*AIME Columbus Meeting, September 1949.*

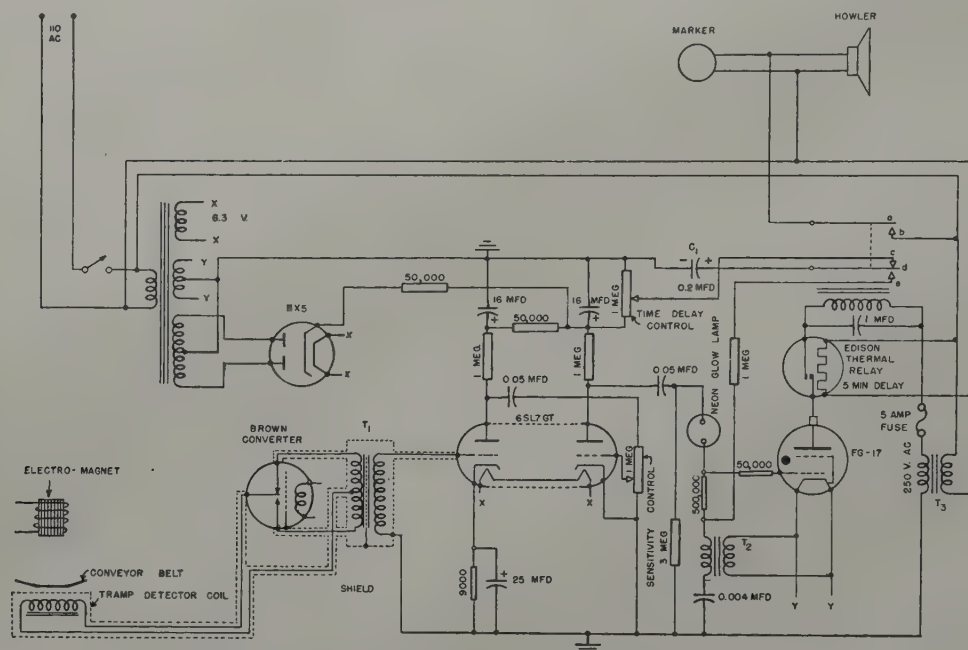
TP 2862 B. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Aug. 17, 1949.

Such a device was employed at the Pioche plant but much maintenance was necessary to keep it in operation. The very sensitive relay was not readily adjustable and was not stable in operation. It was our desire to set the device to detect rock bits but to pass small nails. It was not possible to adjust the sensitive relay to accomplish this end.

Since the belt moves slowly, the current generated by the detector coil is essentially a direct current. It is difficult to construct a simple direct current amplifier that is stable over long periods of time. To avoid this difficulty the circuit shown in fig. 1 was used.

Between the tramp detector coil and amplifier is placed a "Brown converter," which is a special vibrating reed, single-pole doublethrow switch developed by the Brown Instrument Division of the Minneapolis-Honeywell Regulator Co. for use on their "Electronick" recorders and controllers. This device will produce an absolutely square wave 60 cycle current from a direct current source.

If current is flowing from the detector coil it is alternately connected to one end or the other of the primary of the transformer  $T_1$ . This results in an alternating current output which is fed to the al-



**Fig. 1—Circuit of electronic tramp iron detector.**

ternating current amplifier consisting of a single 6SL7 GT electron tube. This tube gives a voltage gain of 3600. The output of the voltage amplifier is fed to the grid of an FG-17 thyratron. This thyratron has the heavy duty relay in its anode circuit and is supplied with anode voltage from the 250 v secondary of transformer  $T_3$ . Transformer  $T_2$  produces bias voltage to prevent the tube from firing when there is no signal from the detector coil.

It will be noted that the grid of FG-17 thyratron is connected to the output of the voltage amplifier through a neon glow tube. This tube will not conduct unless the output from the voltage amplifier is 70 v ac at which voltage the neon glow tube breaks down, applying a high positive voltage to the grid of the thyratron. This prevents minor fluctuations in the magnetic field due to small nails, etc., which give rise to considerable short time voltages from firing the thyratron.

When the thyatron fires, contacts *a* and *b* on the heavy duty relay are made. This sounds the howler and actuates the belt marker. Also contacts *c* and *d* are broken and contacts *d* and *e* are made. When contact *d* is connected to *e* the 0.2 mfd capacitor, *C*<sub>1</sub>, which has been connected across the direct current



power supply of the amplifier through contact *c* is now connected to the grid of the thyatron. This results in a very high positive voltage being applied to the thyatron which keeps it firing for a time period that depends on the setting of the time delay control. This results in an actuation of the howler and the marker for about 1 or 2 sec, regardless of the size of the object passing over the detector coil.

In the installation at the Caselton plant the detector coil is 4 in. long and 40 in. wide and is placed under the belt and very close to it. It has 1000 turns of No. 20 wire and is completely shielded with a copper Faraday shield to prevent electrostatic pick up.

The magnets supplying the steady magnetic field are located above the belt. These are electromagnets

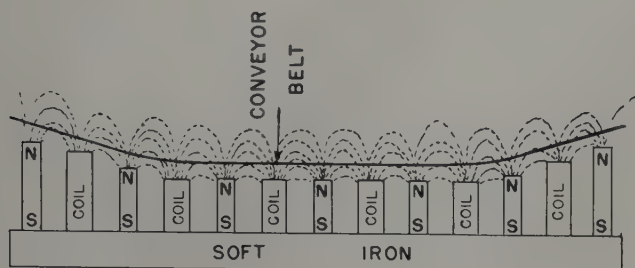


Fig. 3—Marker material hopper and dispenser.

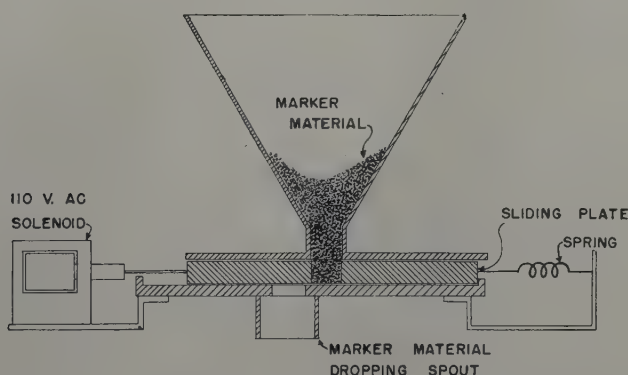


Fig. 2—New coil arrangement of detector.

and are supplied with direct current from a motor-generator set.

For marking the spot on the belt where the tramp iron has been detected white popped perlite is used. Other materials such as clean white sand, powdered coal, chalk, etc., may be used. The marker material is dropped in measured quantities that will make a definite white patch on the belt. The spot on the belt is never more than 2 or 3 in. out of alignment with the tramp being detected.

Through adjustment of the sensitivity control the apparatus is made sufficiently sensitive to detect iron and steel objects the size of rock bits while it does not in general detect small nails. It has been found that a 20 d nail in the bottom of the ore pile, nearest the coil, will sometimes actuate the howler and marker. It depends somewhat on the orientation of the nail relative to the coil. If it is perpendicular to the coil windings it will usually cause an actuation. When the nail is perpendicular to the winds, the most number of turns of the coil are affected by the change in magnetic field.

The Edison thermal relay is used to protect the thyatron. This tube requires a 5 min warm-up period before anode current may be drawn. The Edison thermal relay holds the anode circuit open for 5 min while the tube is warming up.

The apparatus has been in operation over two years and has required little maintenance. To date no tubes have had to be replaced.

The overall voltage gain of the amplifier and input transformer is 36,000. The apparatus is sensitive to current changes in the coil of about 1/100 microamp.

**New Coil Arrangement:** The principal disadvantage of the apparatus is in the arrangement of the detecting coil and magnet. The slightest movement of the coil relative to the magnet causes an actuation of the howler. To avoid this difficulty a new coil arrangement as shown in fig. 2 is being constructed. The coils and magnet will be mounted on a single rigid frame and will not be able to move relative to each other. Having the coils and magnets very

close together will result in greater selectivity. Then also, there will be no dependence on an outside source of direct current power to produce the magnetic field.

**Marker-material Hopper:** Fig. 3 shows in principle the type of marker-material hopper and dispenser used. A sliding plate about  $\frac{3}{4}$  in. thick with a 1 in. hole in it is normally positioned in register with a hole in the bottom of the marker-material hopper. A spring keeps the sliding plate in position. When a piece of tramp passes the detector coil the electronic mechanism causes the 110 v ac solenoid to be momentarily energized causing the sliding plate to be pulled to the left, thus putting the hole in this plate in register with the hole in the marker-material dropping spout. The marker material that fills the hole in the sliding plate is then dropped through the spout onto the moving conveyor belt.

When the actuation is complete the spring pulls the plate into position to register with the hole in the bottom of the hopper again. Bumper blocks prevent excessive travel of the sliding plate and the sharp jar at the end of the travel prevents sticking. The solenoid used has a 32 ounce pull and a travel of about  $1\frac{1}{2}$  in. This arrangement operates excellently when popped perlite is used for the marker material but it is not known if it would operate without sticking if another material were used.

## Discussion

NORMAN WEISS:\* Tramp iron detectors for pro-

\* American Smelting and Refining Co., Salt Lake City.

tecting crushing machinery have been on the market only a few years but have already won favor with operators of crushing plants. In the crushing of sulphide ores the magnetic type of detector is most commonly used but has been criticized by some on the ground that it is difficult to regulate and is not consistent in its ability to single out only those fragments of tramp iron or steel which might damage the crusher.

On the other hand sponsors and supporters of the magnetic detector logically assert that it must be expected to pass through its period of trial and development just as other devices and machines have done in the past. Mr. Marquardt's paper describes a notable development which he has obtained by applying electronic tubes to increase the selectivity of the detector, and by adding a marking device to assist the operator. These changes will shorten the period of evolution.

I shall not comment on the details of Mr. Marquardt's detector, but I should like to point out that his paper expounds in considerable detail the principles of electronics while it gives only passing mention to the principles of ore crushing in which we are particularly interested. It would seem to me that Mr. Marquardt could have added value to his paper by giving us his ideas on the future field for tramp iron detectors in competition with, supplementary to, or working together with magnets. After all, a detector merely detects and manpower must do the job of removal; consequently it is only a temporary expedient, for our ultimate aim is mechanization of such operations.

I should like to mention a case in which I considered that the application of a magnetic detector would have been a mistake, simply because the actual removal of the tramp iron would have been left to the convenience and discretion of the operator. At one of our operations in Central America the ore is conveyed in lumps up to 14 to 16 in. to the primary crusher, and suspended magnets have not been able to give adequate protection to this crusher. When the installation of a detector was suggested I first thought it an excellent idea, but knowing something of the native labor I realized that they would not bother to stop the belt and remove the iron, and that if the detector automatically stopped the conveyor, the natives would simply start it again. It seemed that the only beneficial way to use the detector on this job would have been to have it stop the belt so that it could not be restarted except by the shiftboss after he had found and removed the offending piece of tramp. But this would have involved so much strenuous activity on the part of the shiftboss that the idea was thought impracticable, and was dropped.

HARLOWE HARDINGE:† Some of the control fea-

† Hardinge Co., York, Pa.

tures closely parallel the circuit developments of our "Electric Ear," therefore, I have reason to believe that it can operate continuously without maintenance trouble. As I understand the circuit, it takes care of the difficult condition where materials pass the belt which should not sound an alarm, such as nails and pyrite ore. The means used to prevent a signal, through the use of a controlled voltage regulator, is certainly practical since signals below a definite voltage will not register in the alarm circuit, yet the degree of sensitivity can be controlled.

I only have one suggestion on the circuit and that is that the use of the thyatron tube FG-17 entails the use of a 5 min thermal delay relay; while if a 2050 tube were used, which cannot pass nearly as much current as the one shown in the circuit, it still should be sufficient to operate a sensitive relay of the double-pole-double-throw type to create the time delay used within the sensitive circuit and actuate the howler and marker.

Fig. 2 shows a new arrangement the author proposes to employ but this has apparently not been done as yet.

In these electronic circuits, there is "Many a slip between the cup and the lip" and, therefore, until this new arrangement has actually been put in operation, it might be just as well to reserve judgment as to its practicability, although it does look as though it should work and be an improvement.

I think the means of using a marker to mark where the tramp iron is located on the belt is certainly practical and of use to the operator. I have never heard of this being done before.

Too many practical operators will consider such an arrangement as a gadget, but once they learn how to operate a device of this nature, they are going to save themselves a lot of trouble and their company quite an expense. I can see that Mr. Marquardt's comment on the sensitivity of the operation of the detecting coil and magnet is pertinent. The control at this point of all points should be rugged. It will be interesting to see how his new arrangement works.

C. M. MARQUARDT (Author's reply): The problem of removal of tramp has been enhanced by a better means of detecting the tramp even though the removal remains manual. With the marker on the belt it is a small matter to remove the tramp manually. Insofar as some better means of removing the tramp automatically is concerned I do not see much hope of accomplishing this beyond spreading the ore stream out thinner and slowing it down so the weakly magnetic tramp can be pulled from the stream with magnets, regardless of whether they be electromagnets or permanent magnets. While other means have been considered unfortunately none appear very practical or economical.

The tramp detector is best used as a means of detecting and removing tramp that gets past magnetic pulleys and surface magnets. The amount that gets by is small but it usually is the material that is most damaging to the crusher, such as detachable bits and high manganese steels.

The statement that electronic circuits are tricky must be qualified. When properly engineered and applied there is seldom anything that is less tricky than an electronic circuit. Of the many electronic devices we have constructed and are applying in our operations we have not had any difficulty with the electronic circuits. Such difficulties as have arisen have been in the mechanical devices associated with the device.

A concrete example can be given in connection with the tramp detector. On one occasion the Edison thermal time delay kept opening the anode circuit of the thyatron and thus caused failure to detect tramp. It was found that a heavy electrical load had been put on this particular circuit and the voltage was too low for safe operation of the thyatron tube. The electronic apparatus was doing just what it had been designed to do, protect itself against damage.

This apparatus has been in operation three years and to date not even a tube has had to be replaced. In the case of the previous electromechanical detector that was employed the contacts of the several relays had to be cleaned at least once or twice a month in order to keep the apparatus operating. The difference in maintenance is an ample demonstration of the dependability of the electronic circuit.

The problem of the installation of the new coil and magnetic arrangement is simply one of construction. The strength of a static magnetic field has no bearing on the problem, therefore, from an electronic point of view there is nothing tricky. Since the physics of the problem shows that the change in magnetic flux due to moving tramp will be greater than it now is, it is obvious that the voltage pulse applied to the electronic amplifier will be greater and hence sensitivity will be greater. We anticipate that there will be some instability at very high amplification but this problem is readily corrected by merely turning down the volume control until the desired level is reached.



# Progress Report on Grinding at Tennessee Copper Company

by J. F. Myers and F. M. Lewis

(With Discussion)

The paper reports the development of a large, slow speed ball mill closed circuited with a hydro-scillator. This increased grinding efficiency 28 pct over conventional units.

AS the title indicates, this is a progress report for the first year of experimental operation of a relatively large diameter, slow speed ball mill with 1-in. balls, with and without various classifiers, and including a hydroscillator.

Several years ago the authors came to the conclusion that if any further progress were to be made in our flotation process, an understanding of the chemical and physical reactions and the control thereof, in the grinding circuit, was essential. It may be stated, at this point, that the preparation of a correct feed for the flotation process is still the number one objective of our grinding studies. This progress report deals with grinding "per se," and not with the chemical and physical reactions as the result of grinding.

---

J. F. MYERS and F. M. LEWIS, Members AIME, are Superintendent and Assistant Superintendent of Mills, respectively, Tennessee Copper Co., Copperhill, Tenn. AIME Columbus Meeting, September 1949.

TP 2863 B. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Oct. 4, 1949.

Prior to this time we had taken the naive position, that since grinding had been practiced by the industry for some fifty years or more, it had been well exploited and developed by a host of investigators. Consequently, we had given the matter only superficial attention.

At the start of our study, we were aware of the fact that some difference in opinion existed, but we were not prepared for the controversial evidence that developed nor for the extent to which the experts differed in their viewpoints. There seemed nothing else to do but to tour the continent and decide for ourselves what was right and what was wrong, read all the published literature and visit with anyone who had ideas about the matter. This we did over the next few ensuing years.

Since our study has met with some measure of success, we report, herewith, these data that ultimately they may be of help to other investigators.

In 1944 it became necessary to increase our grinding capacity and at the same time to grind the sulphide portion a little finer. This provided the impetus to put our ideas into effect. As has been previously described, our grinding flowsheet at that time consisted of a 6x12 rod mill, followed by two 5x10 ft, and one 6x12 ft trunnion overflow ball mills, with conventional rake classifiers.

Naturally, we first considered all the possibilities of altering or adding to the existing equipment. It became evident that if any appreciable increase in grinding efficiency could be obtained, it would pay to scrap the existing small mills and classifiers and install one big mill with adequate classification.

Trying to determine just how much more efficient a big mill could be, in which were incorporated all

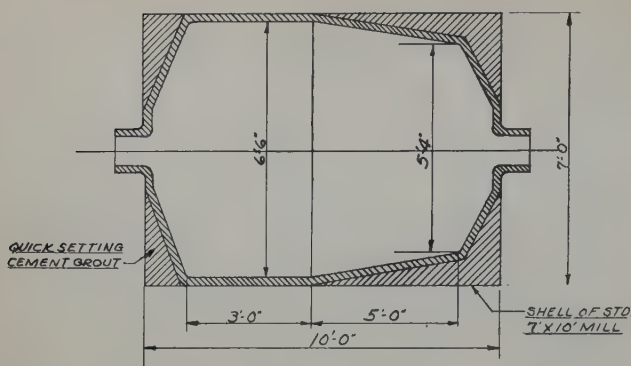


Fig. 1—Cylindrical mill converted to conical type.

the best ideas of operation, certainly brought on a lot of talk and correspondence.

Insofar as we know at this writing, there exists no rule covering the results of various size mills correlated to the other desirable operating features. It does seem, however, that grinding efficiency increases somewhat faster in a grate-type mill than in a trunnion overflow mill, as the diameter size increases. However, this statement holds true only where the grate mill is handling relatively coarse ore with relatively large balls. On fine ball mill feed, say 8 to 14 mesh, with small balls, there is a void in the record, insofar as we were able to determine, on large diameter mills.

The first stage of our study had convinced us that the rod mill was the most economical means of reducing ore from the crusher plant ( $\frac{3}{4}$  in. size) down to 10 or 14 mesh for the ball mill feed. By proper operation and control of the rod mill, we had eliminated any tramp oversize in the rod mill discharge, thus permitting us to go from 2 to 1 in. balls in the ball mills. This change increased the grinding efficiency 9 to 10 pct, as measured by the increase in —200 mesh material.<sup>1</sup> This was a gain which we did not wish to lose in going to one large mill. Our decision to use 1 in. balls eliminated consideration of a grate-type mill, as the evidence indicated that a grate mill cannot retain small balls.

We, therefore, committed ourselves to a trunnion overflow mill, of big diameter with 1 in. balls. To us there seemed to be ample evidence that slow speed offered possibilities of increased efficiency as pioneered by W. I. Garms at Hayden. Also Coghill and other investigators had demonstrated experimentally that slow speed was more efficient. We added slow speed to our specifications.

In 1945, E. H. Rose called to the attention of operators, that in a cylindrical mill, small balls tend to migrate to the feed end of a ball mill and the larger size balls migrate to the discharge end of the mill.<sup>2</sup> W. I. Garms proved that by correcting this horizontal ball migration, the ball mill grinding efficiency increased 6 pct.<sup>3</sup> This was proved by lining a cylindrical mill as shown in fig. 1. This is of course the shape of the Hardinge Tricone mill. Insofar as we know, this is the only case in the wet grinding field where the correction of the horizontal ball migration has been measured.

In the cement industry the horizontal ball migration has long been recognized as important and its correction has been accomplished by first: grate partitions in the mills; second: by Carmen liners.<sup>4</sup> These liners are illustrated in fig. 2. There is reason to believe that Carmen liners would work satisfactorily in a wet grinding cylindrical mill. However,

this would have necessitated our designing and fitting these special liners into a cylindrical mill and it would have introduced another variable with which neither we nor anyone else was familiar, in wet grinding. The Hardinge Tricone was ready for manufacture, so our decision was to use this mill. A diagrammatic sketch of the mill is shown in fig. 3.

**Mill Bearings and Motor:** There seemed to be ample evidence that some 4 pct saving in power input could be obtained by carrying the mill on water lubricated Micarta bearings. These were incorporated in the mill specifications. After a year of operation these have proved to be very satisfactory. It also seemed certain that our old slip-ring motors could be improved.

**Classifier for Closed Circuiting:** On our small ball mills we were never able to obtain a high conventional circulating sand load. There seemed to be considerable question as to just how much, if any, credit we could expect with high circulating loads when the ball mill feed was so fine. We finally concluded that we might get as much as 5 pct credit by proper classification.

By proper classification we had three things in mind: (1) A slime free, or nearly so, classified sand that would keep finished size material out of the

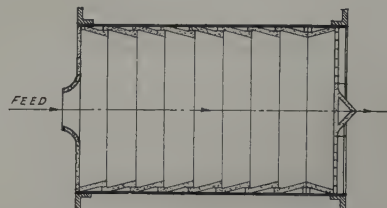


Fig. 2—Carman liner mill.

ball mill and thus prevent so-called overgrinding, and the creation of reagent consuming surface in the flotation machines. (2) If the mill were going to major on lateral grinding (balls rolling or sliding on each other and not cascading), which is characteristic of slow speed mills, then it seemed to us that any lubricating effect of slimes between the small smooth balls would be detrimental. (3) From a purely theoretical angle, it did not seem improbable that by improved classification, some increase in differential grinding of sulphides and gangue could be obtained. This would decrease the power input to the ore as a whole.

We did not know just where or how we were going to get such a classifier, as the existence of such a machine was not known. Nevertheless, we started working on the various manufacturers for such a classifier. The Dorr Co. finally offered the hydro-scillator which would make a sand practically slime free. A description of this machine will not be given at this time, but will be saved for a future progress report. A general description of the Hardinge Tricone is given in table I.

Table I. Description of Hardinge Tricone Mill

Outside diameter at cylindrical section.....	11 ft—0 in.
Average mean diameter inside of liners.....	10 ft—0 in.
Rounded corners to equalize ball slippage.....	4 in. rad.
Discharge opening, choice of, inches.....	12, 22 and 33
Micarta bearings, water lubricated, inches.....	12 x 40
Chilled Ni-Hard liners, T.C. design.....	1 3/4 in. face
Three drive pinions, rpm.....	12.6, 13.7 and 15.0
Selection of percent of critical speed.....	51.96, 56.49 and 61.80
Mill length.....	9 ft—0 in.
Synchronous motor at unit power factor.....	500 hp
End liners of conventional ribbed type.....	Ni-Hard
Grinding balls "Moly Cop".....	1 in. diam



Space does not permit recording the details of the preliminary information obtained regarding the value of the various power factors.

In table II we summarize in col. 1 the range of ideas of reputable engineers and manufacturers concerning these values. In col. 2, we show the values we selected in setting up the cost estimate, justifying the big mill installation. The figures shown are the percentages of power reduction required to produce a ton of —200 material.

**Table II. Value of Power Reducing Factors Over Small Mill**

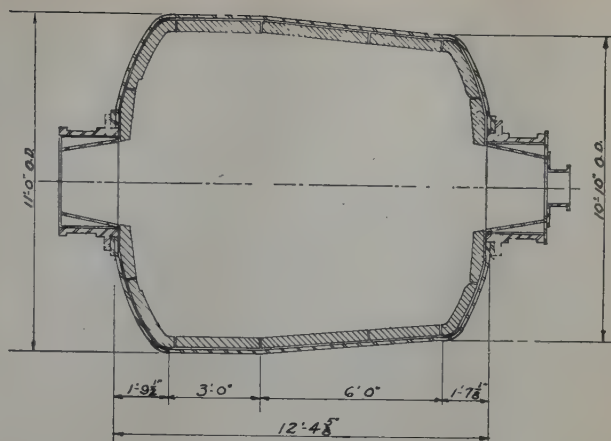
	Pre-liminary Survey, Pct	Authors Estimate, Pct
Increased mill diameter	0—8	6
Control of lateral ball action by slow speed and correction of horizontal ball migration.	0—8	6
Improved motor efficiency	3—5	5
Micarta bearings efficiency	0—4	4
Power reduction by mill		21
Power reduction by classifier	0—5	5
Total		26

It is interesting to note that nowhere in the industry does there exist a big, slow speed mill with small balls. We realized that some degree of uncertainty existed in making such an installation.

The Tricone started operation in August 1948. In the beginning only conventional rake classifiers were available for closed circuiting the mill. These consisted of one 6-ft Dorr FX and two 6-ft Dorr Model D classifiers. By means of pumps, any combination of classifiers and mill could be made.

By October it was clear that the new ball mill, with standard classifiers was capable of equaling the old results at better than 22 pct saving in power. This can be seen in lines 12, 20, and 22 of cols. 1, 2, and 3 of table III. Since this was slightly better than the 21 pct of our estimate, it was quite gratifying.

However, the mill would pull only 308 kw at 13.7 rpm. We did not like the idea of being limited in control of the power input, especially, as we wanted to increase the capacity another 50 tons a day. Space



**Fig. 3—Hardinge Tricone mill.**

does not permit recording the multitude of negative test results that followed, in which we attempted to make the mill draw more power.

The delivery of the hydroscillator classifier was not promised until April 1949, so we could get no help from classification until then.

Among other things it was suggested that since the mill had only a 12 in. discharge opening, we might raise the ball level advantageously. We had been carrying a 45 pct volume. It is of interest to record that we added 11 tons more of balls, and the power needle did not budge.

There seemed to be nothing to do but to increase the mill speed, even though we were convinced that we would lose some efficiency. This we did (15.0 rpm, motor power 348 kw), and operated for 37 days under this setup. The results are shown in col. 4 of table III. It will be observed in line 20 and 21 that no finer grinding was obtained by the extra power and the power saving dropped to 15.3 pct (line 22).

It was evident by this time that the fluidity of the ball mass prevented the transfer of power from the shell even though the shell liners were digging into the first two layers of balls next to the shell. A check with the slow speed ball mills at Hayden did not

**Table III. Operating Data**

Line No.	Col. 1	Col. 2	Col. 3	Col. 4	Col. 5	Col. 6
1 Tons per day	1,900	2,050	2,100	2,100	2,000	2,100
2 Total input, kw	473	453	474	505	448	465
3 Total input, kw-hr per ton ore	5.98	5.31	5.42	5.77	5.37	5.31
4 Total input, kw-hr per ton —200	12.91	10.69	10.70	11.34	11.37	10.06
5 Total rod mill input, kw (5 ft 7 in. diam)	163	162	165	157	160	157
6 Total kw-hr per ton ore	2.07	1.90	1.90	1.80	1.92	1.79
7 Total ball mill input, kw (10 ft 0 in. diam)	310	291	308	348	288	308
8 Ball mill, rpm	27.4	13.7	13.7	15.0	13.7	13.7
9 Ball mill, pct of critical	86.4	56.5	56.5	61.8	56.5	56.5
10 Discharge opening, in.	18	12	12	12	22	33
11 Kw-hr per ton ore	3.91	3.41	3.52	3.97	3.45	3.51
12 Kw-hr per ton —200	12.34	9.54	9.68	10.45	9.95	8.80
13 Ball mill discharge, pct solids	75.0	74.3	66.9	73.0	77.2	58.0
14 Classifier overflow, pct solids	38.3	35.0	35.4	33.8	35.5	34.5
15 Rod mill feed, +0.742	5.3	2.1	2.0			
16 Rod mill feed, —200	9.8	9.6	9.7	9.5	9.5	9.7
17 Ball mill feed, +20	2.5	3.8	2.3	7.3	7.0	6.1
18 Ball mill feed, —200	24.4	23.5	23.9	22.4	22.1	22.6
19 Classifier overflow, +65	5.3	4.9	4.6	4.3	6.5	3.4
20 Classifier overflow, —200	56.1	59.2	60.3	60.4	56.8	62.5
21 Bulk concentrate, —200	65.5	65.1	64.9	65.8	57.7	73.0
22 Pct power reduction, —200 ore ball mill		22.7	22.6	15.3		28.7

Col. 1 Rod mill discharge distributed to classifiers of three small ball mills.  
Col. 2 Rod mill discharge to FX, sand to Tricone and Tricone discharge to Model D classifiers.  
Col. 3 Rod mill discharge direct to Tricone and Tricone discharge to Model D classifier.  
Col. 4 Same as col. 2, except Tricone at 15.0 rpm.  
Col. 5 Same as col. 2, except Tricone equipped with lifter bars.  
Col. 6 Tricone in closed circuit with hydroscillator.

indicate that they had ever experienced any trouble by ball slippage. It will be recalled that Hayden uses rough cast balls, 2 in. in diameter.

Our liners were designed purely for protection to the shell. It was suggested that we had no wedging action from the liners to transmit power, as was the principle of the ship lap and wave type liners.

Our next test of general interest was to weld 5 in. wedge bars on to the liners at an angle of 30° to impart a wedging action to the ball mass. The results of this test are recorded in col. 5 of table III. Even after we cut the tonnage to 2000 tons per day, the results were definitely worse, and the power input reached an all time low of 288 kw. It was argued that the bar angles pocketed the balls and reduced the effective diameter. It will be noted that during the test we enlarged the 12 in. discharge opening to 22 in. (line 10) in order to observe the ball action. Our own conclusion was that 30° bars were giving a wedging action but that they reduced about 5 in. of lateral grinding from the ball mass next to the shell where it was most effective.

While the wedge bar test in itself was a failure, nevertheless, it was probably the means of our ultimate success of the project. When we opened the

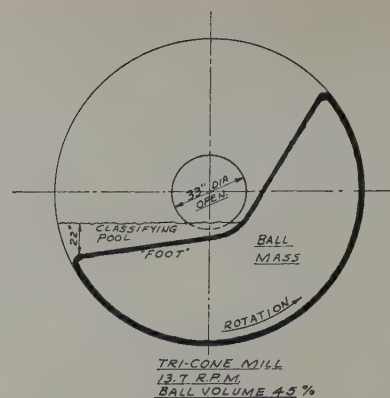


Fig. 4—Ball mill action, Tricone mill.

finer than the top size in the overflow. Aside from this, we were interested in the relative slime-free sand feature as it might minimize overgrinding and thus improve the metallurgy and reagent consumption.

The first ten weeks of varied operation with the mill closed circuited with the hydroscillator were

Table IV. Mill Pool Samples, 12.6 rpm, Discharge 62.3 Pct Solids

Screen	0 in. Down Left 3 ft CL*			10 in. Down 3 ft Left CL			20 in. Down 3 ft Left CL		
	Pet Wt	Pet Sul	Pet Ins	Pet Wt	Pet Sul	Pet Ins	Pet Wt	Pet Sul	Pet Ins
Comp.		40.2	3.4		42.3	2.4		42.6	4.3
28	0.7			2.7	33.0	9.6	2.5	33.4	17.9
35	2.3	18.8	20.0	7.7	36.3	12.5	7.9	37.7	11.4
48	7.2	20.9	15.9	17.4	37.4	10.8	18.3	38.7	9.2
65	17.6	30.4	9.0	34.2	40.4	6.7	35.7	41.3	6.4
100	39.8	41.7	2.3	62.7	44.0	2.5	64.3	44.8	2.5
150	75.2	45.0	1.1	84.1	46.1	2.5	84.6	46.3	1.7
200	85.5	43.5	1.4	90.7	44.2	2.5	91.0	44.4	2.4
—200	14.5	38.6	2.8	9.3	38.9	4.8	9.0	38.6	4.7
Pct Solids	42.7			65.7			68.8		

\* CL: center line.

discharge end to 22 in., we fully expected a lot of the balls to come out, as they would have on most mills. What we saw led us to take off the rest of the trunnion plate, leaving the opening at 33 in. No balls came out, with the ball volume at 45 pct.

What we saw was a surprise to us. In fig. 4 is shown a sketch of the ball mass in action. No cascading of the balls occurs. They simply slide down the slope from the high side to level off in what one friend terms the "foot." There is no impact of balls against the down side of the shell liners or on the balls themselves, they just curl under and start over again. The action is 100 pct lateral grinding, either rolling or sliding, or both.

Insofar as we can see, the mill efficiency is just as good with the 33 in. opening as with the 12 in. opening when the mill is closed circuited with classifiers.

**The Hydroscillator:** The installation was made in April 1949. The laboratory tests indicated that it would increase the grinding efficiency by eliminating critical size material which is prevalent in the sand load of a conventional classifier. By critical size material we mean grains the first three meshes

very discouraging, but by July adjustments began to click. In col. 7 of table III is shown the effects of the hydroscillator with the mill at 13.7 rpm.

It is to be noted that the power saving on the —200 mesh ton basis increased to 28.7 pct (line 22). Under normal conditions the rake sands contain from 2 to 4 pct —200 mesh.

**Classifying Pool in Mill:** Referring again to fig. 4, it will be noted that over the "foot" of the ball charge there exists a classifying pool. This pool is at least 6 ft wide and 9 ft long. Since no balls were falling into the pool to disturb it, we investigated the pool characteristics.

Blocks of wood dropped into the scoop box were picked up by the scoop and deposited in the discharge box in 34 sec. Certainly any slimes or fines that got into the pool were quickly removed.

By means of vacuum lines, samples were aspirated out of the pool at various points under different conditions. From the analysis of the samples withdrawn, while operating, it is obvious that true classification takes place.

Space does not permit recording all the screen analyses of the samples taken. For illustrative pur-



Table V. Mill Pool Samples, 13.7 rpm, Discharge 47.0 Pct Solids

Screen	Ball Mill Discharge			Top at CL*			10 in. Down at CL			20 in. Down at CL		
	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins
Comp.		35.3	10.4		36.2	10.6		38.0	8.6		38.8	7.6
28	0.3			0.4			1.0			1.3		
35	0.4			2.5	22.4	31.5	3.7	24.1	28.5	5.2	29.1	21.8
48	1.6	21.6	32.5	7.4	28.3	20.1	10.3	32.8	15.1	13.5	35.9	11.6
65	5.7	30.9	18.4	19.3	33.8	12.4	25.8	37.6	9.3	30.5	39.5	8.2
100	14.8	38.0	8.9	40.3	38.5	8.0	48.2	41.0	6.1	52.0	41.7	4.9
150	18.1	39.9	6.5	56.0	41.1	6.3	67.4	42.0	5.5	67.5	41.5	4.8
200	13.6	32.9	9.0	69.3	37.6	8.4	77.1	39.6	6.9	78.3	40.0	6.7
-200	45.5	35.1	10.7	30.7	32.6	11.9	22.9	33.9	10.4	21.7	33.6	10.7
Pct Solids	47.0			46.4			58.4			61.1		

\* CL: center line.

poses of the classification effect, tables IV and V are typical of halfway back in the mill at different speeds and mill dilutions.

Summarizing all the data of the pool samples, it is evident that the rate of increase in percentage of solids with depth is rapid at the surface, but decreases continuously until at 10 in. a dilution of approximately the ultimate for the pool is reached. This would indicate that the pool is relatively quiescent in the upper zone, but turbulent next to the ball mass in the foot.

It follows that the phenomenon observed is classification and not thickening. We conclude that: (1) The rate of settling decreases as the percentage of solids increases. (2) The rate of classification decreases as the mill speed increases.

At this writing, we think the turbulence in the lower zone of the mill pool is due to pulp coming out of the ball mass and not due to agitation of the moving balls at the surface of the foot.

**Summary:** There remain many adjustments to make and there are many questions to answer. The effect and operation of the hydroscillator will be left to a future Progress Report; likewise, ball wear, liner wear, etc.

While we have accomplished our objective, insofar as the total power reduction is concerned, there is no evidence as yet to show just how much was contributed by each of the various factors.

Neither is it demonstrated how effective this mode of operation would be on a medium or hard ore. The Bond grindability tests show that the London ore is quite soft, namely 6.04 net g undersize per revolution at 48 mesh. Line 12 of col. 6 in table III indicates that we are producing a ton of -200 mesh material for 8.80 kw-hr as against 12.34 in the old small mills.

In the light of Bond grindability factor, the 8.80 kw-hr is not an impressive figure when compared to known accomplishments of other grinding plants in the industry. It would therefore appear that the work of the small mills must have been very poor. Insofar as we know, the small mills were doing all that was possible for them to do, and we can think of no way that their work could have been improved.

The only explanation that we have at this time is that the Bond grindability tests should have been made at 65 and 100 mesh, that possibly the individual crystals in the ore are easy to separate down to 48 mesh, but that the power required to break the crystals below 48 mesh increases materially.

**Conclusion:** However, we conclude, at this time, that the study of large diameter, slow speed mills with small balls and proper classification offers an

interesting field of investigation. It is a subject beyond the scope of one small group of investigators with just one mill, and one hydroscillator.

### References

- <sup>1</sup> J. F. Myers and F. M. Lewis: Evaluating Grinding Changes at the Isabella Mill, Tennessee. *Western Miner* (Jan. 1946).
- <sup>2</sup> E. H. Rose: Ore Concentration and Milling. *Annual Review Min. and Met.* (Feb. 1946).
- <sup>3</sup> Letters in authors' file.
- <sup>4</sup> C. L. Carman: Grinding in the Cement Industry. *Rock Products* (April 1938).

### Discussion

H. HARDINGE:\* The authors state that so far as

\* Hardinge Co., York, Pa.

they know, the test made by Garms at Hayden is the only case in the wet grinding field where correction of the horizontal ball migration or size segregation has been measured in terms of gain in grinding efficiency. In the early days of the advent of the conical mill, particularly the period from 1906 to 1916, a number of comparative tests were made and it was the result of these tests which was directly responsible for so many conical mill installations. Garms has confirmed these earlier tests by using a mill with less taper in the conical section than was used originally.

With regard to the gain due to classification, might not the 6.1 pct additional gain between col. 3 and col. 6 (22.6 to 28.7 pct) be due in part, at least, to the change in pulp density and classifying effect of the pool within the mill itself rather than to cleaner sands obtained by the hydroscillator? It is noted that prior to lowering the pulp density in the mill, the results obtained with the hydroscillator were "very discouraging." It would be very interesting to see what the result would be if the mill were operated under the new conditions with a standard Model D classifier, of sufficient capacity and over a sufficient period to determine if the gain is peculiar to the combination of low pulp density in the mill with hydroscillator or if the overall result would be substantially the same in either case.

When two changes are made at the same time, it is always difficult properly to weigh the effect of each change.

Note that the authors demonstrated that a piece of wood passed through the mill in a matter of seconds. They also logically assume that slimes or fines would do likewise. Then why would not an increased amount of pulp, within reason, carrying the added slimes or fines from a less efficient classifier, pass through the mill without noticeably decreasing the grinding efficiency?

# Relative Wear Rates of Various Diameter Grinding Balls in Production Mills

The results of wear on marked balls, 4, 3½, 3, and 2 in. diam are given. All balls were forged steel of practically the same chemical analysis and hardness. The results indicate that balls in a given mill for a given length of time will have equal diameter losses, regardless of size.

by D. E. Norquist and J. E. Moeller

(With Discussion)

IN order to determine the relative wear rates of several sizes of grinding balls, groups of 4, 3½, 3, and 2 in. balls were marked individually and were charged, all at the same time, into each of two production mills grinding copper ore. The remainder, and vast majority, of the ball charge consisted of 2 in. diam, and smaller, white, cast iron balls. The original weight of each test ball was determined and recorded. Some of each group of test balls were recovered from the mills periodically, individually reidentified, and reweighed. The test balls were recovered during regular maintenance shutdowns, at approximately thirty-day intervals, so as to avoid disrupting operations. As soon as the weights had been recorded, each marked ball was recharged into the mill from which it had been taken.

D. E. NORQUIST and J. E. MOELLER, Members AIME, are, respectively, Manager, Grinding Media Division, Sheffield Steel Corp., Kansas City, Mo., and District Manager, Sheffield Steel Corp., El Paso, Texas. AIME Columbus Meeting, September 1949.

TP 2864 B. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Oct. 10, 1949.

The test units were mechanically the same and theoretically received equal tonnages of feed.

All of the test balls were of forged, alloy steel composition with only slight variations in chemistry that would not be expected to affect the physical properties. All were heat treated in the same manner to produce a high hardness, as equal as possible for all sizes and as uniform as possible from surface to center. Generally speaking, the hardness of the portion worn from the balls during the test ranged from 62 to 65 Rockwell C. The microstructure was, therefore, predominately martensitic.

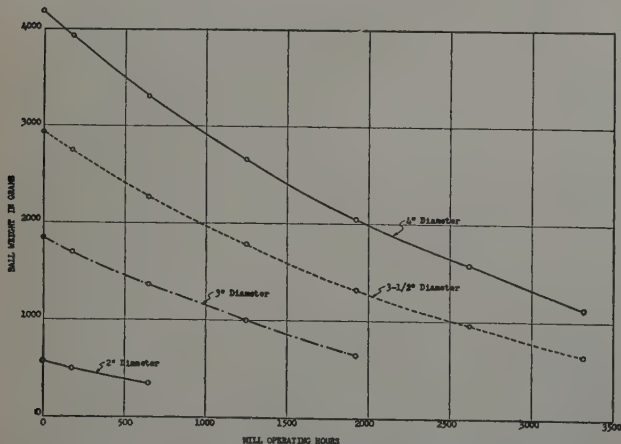


Fig. 1—Average weight loss in each ball size group in mill No. 1.

The method of marking the test balls consisted of forming a small, round hole, 3/16 or ¼ in. diam, into the center of each test ball with a Thomas Metal Master disintegrator, which produces a hole in fully hardened steel without affecting the hardness of the surrounding metal. After the hole was formed, each test ball was individually weighed to the nearest gram and the weight was recorded. The balls were grouped as to size and each group assigned a code letter. The individual balls within the various groups were then assigned a number. A small copper disc was stenciled on one side with the group letter and on the other side with the individual ball number and dropped into the hole in the corresponding test ball. The hole was then plugged with a low-melting-point metal alloy, which could be melted out in boiling water. Half of the test balls in each group were charged into one mill and the other half into another mill, so that all test balls in each mill would be subject to, as nearly as possible, identical conditions for an equal length of time.

Previous testing had indicated that the wear rates on balls of different sizes could not be accurately compared if weight loss, or percentage weight loss, were used as a basis for comparison. Fig. 1 is a graphic record of the average weight loss for each size group in mill No. 1, and it becomes readily apparent that the larger balls lose substantially more weight in a given length of time.

Fig. 2 is a graphic record of the average percentage weight loss for each size group in mill No. 1, illustrating that the smaller balls definitely lose a greater percentage of their original weight in a given length of time.

It was, therefore, decided to convert the actual weights of the test balls to diameter and follow the diameter losses periodically to see if the wear rates

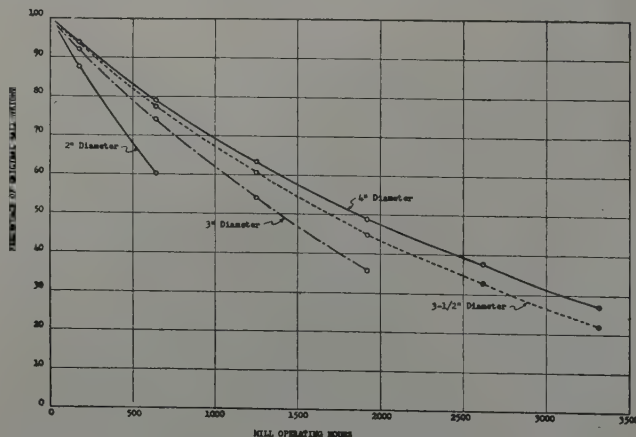


Fig. 2—Average percentage weight loss in each ball size group in mill No. 1.



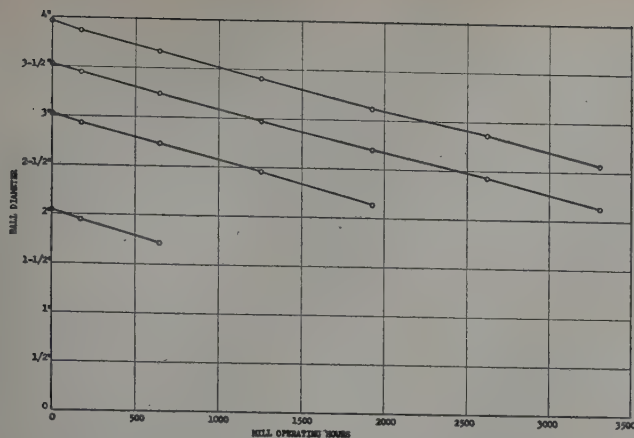


Fig. 3—Diameter loss in each ball size group in mill No. 1.

could be accurately compared on the basis of diameter loss. Table I is an example of the conversion from actual weight to diameter, and diameter loss, for each ball in one size group. The results are shown for each period that the test balls were recovered, until this group of balls had worn down to the point that a sufficient number to arrive at a reliable average could no longer be found in the time allowed for searching. A specific gravity of 7.82 was used for converting weight to diameter. Similar tables were prepared for each size group in each mill.

Table II shows the averages for each test period and the cumulative results for the various test periods on each size group in both mills.

Fig. 3 graphically illustrates the diameter loss for each size group in mill No. 1, as long as sufficient samples could be recovered representing each size group.

Fig. 4 is a graph plotted in the same manner, showing the results obtained in mill No. 2.

Table I. Original 3 in. Diam Mark N1 in Mill 1

Weight Loss in Grams								
Ball No.	Original Wt	1st Period	2nd Period	1st and 2nd Period	3rd Period	Accum. 1—3 Incl.	4th Period	Accum. 1—4 Incl.
1	1,892	152	340	492	375	867	347	1,214
2	1,808	141	327	468	360	828	333	1,161
3	1,888	149	338	487	373	860		
4	1,886	153	337	490	373	863	352	1,215
5	1,896	153	339	492	375	867		
6	1,796	141	325	466	359	825		
7	1,896	154	342	496	374	870		
8	1,799	139	329	468	359	827	328	1,155
9	1,812	137	330	467	362	829	336	1,165
10	1,807	138	329	467				1,173
11	1,808	146	328	474	358	832	335	1,167
12	1,821	140	330	470	367	837		
13	1,892	152	338	490	377	867	347	1,214
14	1,800	141	327	468	357	825	332	1,157
Avg.	1,843	145	333	478	367	846	339	1,180

Diameter Loss in Inches

1	3.044	0.084	0.207	0.291	0.272	0.563	0.319	0.882
2	2.998	0.080	0.205	0.285	0.269	0.554	0.316	0.870
3	3.042	0.082	0.206	0.288	0.270	0.558		
4	3.041	0.084	0.206	0.290	0.271	0.561	0.325	0.886
5	3.047	0.084	0.206	0.290	0.271	0.561		
6	2.991	0.080	0.205	0.285	0.269	0.554		
7	3.046	0.085	0.208	0.293	0.271	0.564		
8	2.993	0.079	0.207	0.286	0.269	0.555	0.313	0.868
9	3.000	0.078	0.206	0.284	0.270	0.554	0.318	0.872
10	2.997	0.078	0.206	0.284				0.883
11	2.998	0.083	0.206	0.289	0.268	0.557	0.319	0.876
12	3.005	0.079	0.206	0.285	0.273	0.557		
13	3.044	0.084	0.206	0.290	0.273	0.563	0.319	0.882
14	2.994	0.080	0.206	0.286	0.268	0.553	0.316	0.869
Avg.	3.017	0.082	0.206	0.288	0.270	0.558	0.318	0.876

### Summary

1. Diameter loss appears to be a reliable basis for comparison of wear rates on grinding balls, regardless of their original size.
2. The method described for marking and testing grinding balls in production mills is probably limited, for practicable application, to balls of about 1½ in. minimum diameter.
3. During the 3rd period, the test balls in mill No. 2, for undetermined reasons, began wearing ap-

Table II. Ball Wear for Mill Nos. 1 and 2 for Each Test Group

Period	Average Losses for Balls Recovered	Mill No. 1					Mill No. 2				
		Operating Hours	4 In.	3½ In.	3 In.	2 In.	Operating Hours	4 In.	3½ In.	3 In.	2 In.
1	Weight loss in grams		255	181	145	70		235	170	135	60
	Diameter loss in inches	176	0.082	0.074	0.082	0.088	163	0.075	0.070	0.075	0.076
	Diameter loss per 100 operating hours		0.047	0.042	0.047	0.050		0.046	0.043	0.046	0.047
2	Weight loss in grams		617	480	333	152		628	477	340	150
	Diameter loss in inches	471	0.215	0.213	0.206	0.225	467	0.213	0.215	0.207	0.216
	Diameter loss per 100 operating hours		0.046	0.045	0.044	0.048		0.047	0.046	0.044	0.046
1 and 2	Weight loss in grams		872	661	478	225		863	647	475	213
	Diameter loss in inches	647	0.297	0.287	0.288	0.317	630	0.293	0.285	0.282	0.298
	Diameter loss per 100 operating hours		0.046	0.044	0.045	0.049		0.047	0.045	0.045	0.047
3	Weight loss in grams		656	493	367			728	536	399	
	Diameter loss in inches	609	0.260	0.253	0.270		609	0.291	0.282	0.292	
	Diameter loss per 100 operating hours		0.043	0.042	0.044			0.048	0.046	0.048	
1 through 3	Weight loss in grams		1,528	1,154	846			1,591	1,183	874	351
	Diameter loss in inches	1,256	0.557	0.540	0.558		1,239	0.584	0.567	0.574	0.571
	Diameter loss per 100 operating hours		0.044	0.043	0.044			0.047	0.046	0.046	0.046
4	Weight loss in grams		613	468	339			697	510	375	
	Diameter loss in inches	668	0.285	0.288	0.318		674	0.334	0.331	0.354	
	Diameter loss per 100 operating hours		0.043	0.043	0.048			0.050	0.049	0.053	
1 through 4	Weight loss in grams		2,141	1,622	1,180			2,288	1,693	1,256	
	Diameter loss in inches	1,924	0.842	0.827	0.876		1,913	0.918	0.897	0.928	
	Diameter loss per 100 operating hours		0.044	0.043	0.046			0.048	0.047	0.049	
5	Weight loss in grams		459	348				479	349		
	Diameter loss in inches	697	0.254	0.263			699	0.281	0.289		
	Diameter loss per 100 operating hours		0.036	0.038				0.040	0.041		
1 through 5	Weight loss in grams		2,600	1,969				2,767	2,033		
	Diameter loss in inches	2,621	1.096	1.091			2,612	1.200	1.185		
	Diameter loss per 100 operating hours		0.042	0.042				0.046	0.045		
6	Weight loss in grams		446	646				442			
	Diameter loss in inches	698	0.301	0.305			698	0.317			
	Diameter loss per 100 operating hours		0.043	0.044				0.045			
1 through 6	Weight loss in grams		3,036	2,293				3,248			
	Diameter loss in inches	3,319	1.396	1.397			3,310	1.520			
	Diameter loss per 100 operating hours		0.042	0.042				0.046			

Note: The losses for only the balls recovered in a given period were used to determine averages; and the losses shown for the accumulated periods may not, therefore, total the same as those for the individual periods.

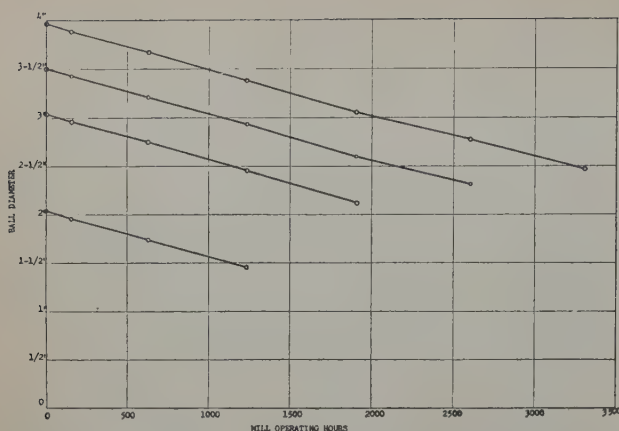


Fig. 4—Diameter loss in each ball size group in mill No. 2.

preciably faster than those in mill No. 1. The average percentage increase in wear rate for each test period was:

Test Period	Faster Wear in Mill No. 2, Pct
1	— 2.2
2	Equal
3	10.0
4	13.4
5	9.5
6	4.7

This indicates that an attempt to evaluate the quality of grinding balls by comparing the consumption of one type of ball in one mill to that of a different type in a second mill, may result in extremely misleading information.

4. A substantial reduction in wear rates occurred during the 5th period on the balls in both test mills. Subsequent investigation revealed that grinding media consumption per operating hour and per ton of ore ground decreased substantially throughout the entire plant during that month, probably due to changes in the abrasive properties of the ore.

5. The indicated life of the test balls in the test mills, assuming discharge at  $\frac{1}{2}$  in. diam, is approximately as follows:

Ball Size, In.	Days, Continuous Operation
2	128-133
3	213-226
3 1/2	278-298
4	317-347

6. The length of time required to completely replace a ball charge is generally assumed to be considerably less than these figures indicate but can be quite accurately determined if properly calculated.

7. The tabulated wear rates of the alloy, forged steel, test balls are not, of course, necessarily indicative of what the wear rates would be in other mills, particularly since the test balls were mixed in with a charge of cast iron balls and the circulating load contained a quantity of cast iron grit.

### Bibliography

<sup>1</sup> E. W. Davis: Fine Crushing in Ball Mills. *Trans. AIME* (1919) 61, 250.

<sup>2</sup> T. K. Prentice: Ball Wear in Cylindrical Mills. *Jnl. Met. and Min. Society of South Africa*. (January-February 1943).

<sup>3</sup> F. C. Bond: Wear and Size Distribution of Grinding Balls. *Trans. AIME* (1943) 153, 373.

<sup>4</sup> W. I. Garms and J. L. Stevens: Ball Wear and Functioning of Ball Load in a Fine Grinding Ball Mill. *Trans. AIME* (1946) 169, 133; *Min. Tech.* (March 1946) TP 1984.

<sup>5</sup> T. E. Norman and C. M. Loeb, Jr.: Wear Tests on Grinding Balls. *Trans. AIME* (1949) 183, 330; *Min. Tech.* (May 1948) TP 2319; *Trans. AIME* (1948) 176, 490; *Met. Tech.* (April 1948).

### Discussion

E. W. DAVIS:\* The conclusions by the authors of

\* Univ. of Minn., Minneapolis.

this paper support the results presented in a paper by T. K. Prentice.† He found that the diameter loss of

† T. K. Prentice: Ball Wear in Cylindrical Mills. Abstracted and reviewed by E. W. Davis. *Trans. AIME* (1946) 169, 147; *Min. Tech.* (July 1944) TP 1736.

grinding balls appeared to be constant, irrespective of the diameter of the balls, or stated otherwise, that the rate of wear of any ball is proportional to its surface or the square of its diameter.

In the experiments conducted by Mr. Prentice, the mill was operated at a comparative low speed and there is some evidence to indicate that in mills operated at higher speeds the rate of wear approaches the cube of the diameter of the ball.

It would be interesting to know the per cent of critical speed at which these mills were operated and perhaps some of the other operating conditions.

T. E. NORMAN:‡ The authors are to be congratulated

‡ Climax Molybdenum Co., Denver.

for the careful and painstaking work they have done on a relatively long series of wear tests. Their data provide further confirmation that grinding balls tend to lose diameter at a constant rate and thus wear (lose weight) in direct proportion to their surface area, irrespective of their weight or size. This is further evidence that the grinding forces in a ball mill are uniformly distributed over all ball surfaces.

We are pleased that the authors' findings on rate of ball wear agree with our observations<sup>5</sup> and also those of Prentice.<sup>2</sup> For a long wear test such as the authors have described in their paper, it is evident that relative wear rates should be based on the average diameter loss for each type of ball. These average diameters can be accurately determined by calculation from ball weights and densities. Where tests of shorter duration are run, we feel that the calculations can be somewhat simplified by using the factor of weight loss per unit of ball area as a basis for comparison. This factor is in direct proportion to diameter loss, so may be used provided ball areas do not change too greatly during the wear test. Most of our wear tests have involved periods of 24 to 250 hr duration. On these we have found the comparisons on the basis of weight lost per unit of ball area (area at the start of the test) to be just as accurate, within the limits of experimental error, as comparisons based on loss of diameter.

The authors have developed a method of marking which has a minimum influence on the wearing surface of each ball. This method is readily usable so long as the balls have some other distinguishing feature, such as size or soundness, which allows them to be picked out readily from the rest of the balls in the mill charge. Where, however, tests are run on balls which are similar in size and other physical characteristics with those regularly charged into the mill, then we believe it is desirable to use a more obvious distinguishing mark such as one or two notches cut in the surface of each test ball. These notches cover a greater surface area than a small drilled hole, but in spite of this they have not measurably changed the rate of wear on the groups of balls we have studied to investigate this matter.



# The Colmol—A Continuous Mining Machine

by C. H. Snyder

The paper deals with details of construction of the Colmol, including improvements in design that will be incorporated in new models. These improvements are results of problems encountered and worked out in experimental operation with the unit. Also included is a descriptive plan of operating where roof and air conditions are of the worst. A summary of production results from the machine is presented.

THE various units used in conventional mining are well-built. Generally the universal undercutter, driven by approximately 50 hp, weighs approximately 25,000 lb, and is a very strong piece of machinery. The drilling machine is not as husky as an undercutter, generally is driven by approximately 15 hp, weighs about 9000 lb, but is a well-constructed, sturdy mechanism. The loader, usually 60 hp, weighs approximately 24,000 lb, and certainly is a strong, rugged unit. The working parts of all three of these sturdy conventional units have been

C. H. SNYDER, Member AIME, is President, Sunnyhill Coal Co., Pittsburgh, Pa.

AIME Columbus Meeting, September 1949.

TP 2865 F. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Oct. 25, 1949.

made of tough steel alloys for some time. With the use of three machines having a combined horsepower of approximately 125, with a combined weight of approximately 60,000 lb, and with the possible inherent advantage of single purpose machines performing a single function, the use of considerable explosives has been required to complete the conventional mining cycle.

It was therefore our thought, upon conception of the first Colmol,\* a machine which was to do the

\* Trade mark applications have been applied for, for the word "Colmol," identifying it as a continuous mining machine.

work accomplished by the three conventional units

plus explosives, that it must be a rugged mechanism. Now, after building several units, after going through many and varied experiments, and after the expense of some million-odd dollars, we find we have observed and learned nothing that contradicts our original thinking. A continuous miner, if it is to give satisfactory performance, must be a rugged piece of machinery.

Ruggedness has been, and is constantly, in the minds of our engineers in designing the Colmol. Improvements where they have been required have been in the direction of more weight, more horse power, better alloys, bigger hydraulic pumps and motors; in short, always, toward more torque, greater strength, more ruggedness. Now not only the finest steel alloys are utilized but more than 100 Timken roller bearings are used.

The following is some pertinent data on models being built and to be built:

The low unit: Cutting height, 30 to 42 in. By changing the upper gear case the maximum cutting height can be increased to 61 in., giving a variable cutting height of from 30 to 61 in. Width of cut, 9 ft 6 in. Width of body, 75 in. over tracks. Height of body, 27½ in. Overall length, 23 ft. Driven by three 50-hp motors or a total of 150 hp. Weight, 26 tons.

The intermediate unit: Cutting height, 45½ to 72 in. Width of cut, 9 ft 8 in. Width of body, 77 in. over tracks. Height of body, 43 in. Overall length, 26 ft. Driven by three 70-hp motors or a total of 210 hp. Weight, 35 tons.

The high unit: Will have a cutting height of 64 to 88 in. Will be driven by three 90-hp motors or a total of 270 hp, and will weigh 92,000 lb.

The extra high unit: Will have a cutting height of 80 to 120 in. Will be driven by three 135-hp motors or a total of 405 hp, and will weigh 135,000 lb.

While ruggedness, weight, strength, torque are vital in a continuous miner, likewise foolproof design is important. Removing coal from the solid and loading it with one machine is a difficult task. Any machine designed to do this job, day in and day out and under greatly varying conditions, must be very powerful.

Because of the human element involved in operating the unit and because of clay veins, rock conditions, sulphur balls and similar materials and conditions existing in coal seams, which the machine may come upon without warning to the operator, some provision should be made to shut off immediately this terrific power before the unit can be damaged. Since all functions of the Colmol are performed by hydraulic or fluid drives, any sudden excessive strain or load results in the hydraulic oil going over a relief valve and causes the particular function under such load to stop. Therefore, without exerting a sudden or excessive load on electric motors or gears the function is stopped, the operator is warned, and investigation can be made. The stall or relief points are adjustable for different conditions existing in different seams. With this overload arrangement it is extremely difficult to abuse the machine. Thus, we believe we have provided not only ruggedness but comparatively foolproof ruggedness.

A continuous miner such as the Colmol, with its 3 to 5 tons or more per minute production, will obviously replace several conventional mining units. With probable daily production of 300 tons per shift in the low model and up to 800 tons per shift and more in the larger models, the mine operator will be very dependent on the production of each machine. When one Colmol is not in operation a shift's production will be greatly affected. With many mines operating two or three shifts per day, a continuous miner is likely to receive as much use in one year as many underground units now receive in two or even three years.

Such a machine must be so designed that its parts may be easily accessible and quickly replaced. With this in mind we have provided for: (1) Replacement of all bits by one man in 4 min. (This is accomplished through a novel bit holding arrangement.) (2) Replacement of hydraulic pumps and motors underground in operating height and on a unit basis. (The hydraulic pump or motor may in each instance be removed and replaced simply and quickly and without removing other gears or drives. Work may be resumed quickly and repairs may be made in the mine shop.) (3) Replacement of gear reduction units or drives on a similar unit basis.

Thus we believe we have provided for quick resumption of production and lowest possible maintenance costs.

So much for the design and construction—let us delve into the inside story of the Colmol. What have been our problems? What have we done to correct them? Where does the Colmol fit in future mining? When will machines be available?

Now, the Colmol advances into the solid on caterpillars and under its own tractive effort. In our earlier experiments we encountered considerable cat slippage. We finally resolved this difficulty with

bit design. We have learned that the best bit, the one which will produce the greatest tonnage and best size of product, must provide for both cutting and then wedging or breaking the coal between it and the adjoining bit. We found that changes in wedge design could control the depth of the kerf and that depth of the kerf had a direct relation to resistance to forward movement, that is, to tractive effort required of the machine. Now, by breaking the coal between the bits, we have eliminated all track slippage and reduced hydraulic pressure required to drive the cats from its former 900 to 1200 psi to under 500 psi. However, we spent tens of thousands of dollars learning this fact.

The unit had, for a long time, a tendency to climb. It would start into the coal cutting exactly even with the bottom but would, in a matter of several feet, climb as much as several inches from the bottom. Since the kerfs between the heads on the bottom are cut out with the dozer blade and since the use of such a blade for such a purpose was apparently new, we deduced that the tendency to climb was a result of the upward resistance of these kerfs to the forward movement of the machine. After many headaches and costly experiments we finally learned, more by accident than design, that the tendency to climb was due to location of the center of weight in the body of the machine. We had long known that the center of weight including the heads (which incidentally weigh some 7 tons) was forward of center of the cats. What we did not realize was the now very obvious fact that in actual operation the greater part of the head weight was supported by the blade and the breaker heads and that therefore the actual center of weight when operating was well to the rear of the center of the cats. Thus, the back grousers were penetrating more than the front and while the machine was apparently level, it was actually inclined upward. The line of cut into the solid simply followed an incline upward parallel with the line of cat travel. This was corrected by proper computation and correction of center of weight by eliminating the proper percentage of head weight supported by the blade and heads while operating. Now by tilting the heads forward or backward or by raising or lowering the heads above or below cat level (all functions which are performed hydraulically) the operator can cut on the level, up, or down.

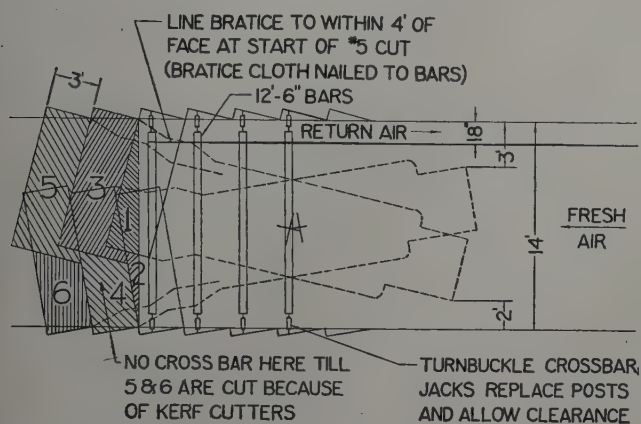
In the operation of the machine under actual floor and roof conditions, we encountered difficulty where rolls existed in the entry or room. Often these rolls will start up or down in the direction of the machine's travel. In many cases they will go on a line substantially perpendicular to the machine's travel. We have now provided 4 hydraulic jacks, one at each end of the cats, so that the operator may, from his operating position, raise the machine by jacking up the front or rear or both and quickly throw a timber, cap pieces or other material under front or rear or under one side to provide for a travel parallel to the coal-seam bottom.

There have been many minor difficulties. As an example, we encountered a condition of overheating in a cone drive bearing which was caused by the unusual angle of the drive and the fact that the drain was so situated that oil could not be drained from the lower end of the case and around the bearing. Sludge was permitted to accumulate at this point. A drain was quickly provided at the lowermost point of the case and the condition remedied. Some mine men after viewing the Colmol in



operation say, "that certainly mines the coal and I can see widespread usage of the machine under good roof or where no gas is encountered, but what am I going to do where my roof conditions require cross bars within 4 or 5 ft of the face and where I have to carry line brattice up close to the face?" The following information details plans for using the Colmol which will permit timbering and carrying air up to within 4 or 5 ft of the face without undue loss of time (figs. 1 and 2).

In the alternate angle cut from center (fig. 3), it is estimated that in completing a left hand and right hand cut which we term a cycle, 29 sec are required in backing and swinging (nonproductive time) which will permit the machine to mine a 3-ft cut in 4 min. The average forward advance is computed at 18 in. per min. The amount of coal in each cycle (combined left and right hand cut) based on a 5-ft



**Fig. 1—Plan for bringing fresh air to face, using turnbuckle jacks and cross timbers over the machine.**

seam is 12.4 tons, requiring a total mining, back up, and swing time of approximately  $4\frac{1}{2}$  min, which would result in a production of slightly under three tons per min, based on total elapsed time.

When one considers that the Colmol will advance a 14-ft entry or an 18-ft room a distance of 10 ft in about 10 min, an advance by conventional means requiring several hours, and when one considers that no damage is done to the roof by shooting, and still further when one realizes that the operator is back some 20 ft from face, it seems logical to assume that less timbering will be required in any given area.

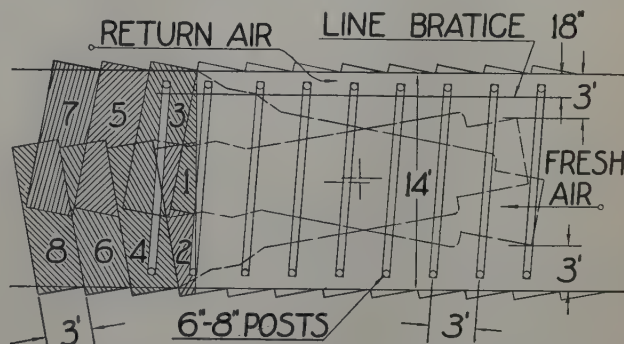
Full scale production appears to be eight to twelve months away. The Jeffrey Manufacturing Co. plans to produce seven experimental Colmols of the low and intermediate sizes; the first of which will be completed in the next few days, and the last of which will be available within six to seven months. Our reason for so limiting the number is that we feel the gaining of a much wider experience in different coals and under different mining conditions is wise before full scale production is begun.

After gaining such experience with these limited number of units, a rugged brute of a machine, which will be comparatively foolproof and which will require a very minimum of down time and maintenance cost, will be offered to the industry. These units should result in extremely high production and low face costs.

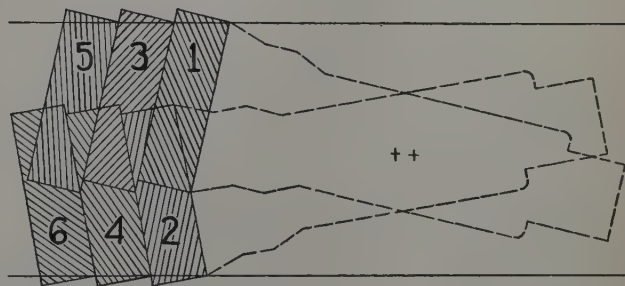
Present existing intermediate transportation facilities will not permit the maximum use of the

Colmol's productive capacity. Actual time studies show that under average haul, using shuttle buggies, the Colmol is able to operate only 30 to 50 pct of the time.

At present the first experimental model is operating in the Upper Freeport seam of coal at Reedsville, Preston County, West Virginia, mining an average of 50 in. of coal. The machine and a five man face crew on a 7-hr shift, when operating conditions were ideal, produced over 400 tons. During this shift the Colmol drove a single  $9\frac{1}{2}$ -ft breakthrough of 80 ft in 90 min, producing 128 tons of coal. The daily average production to date is 225 tons per shift, which means an approximate advance of 112 ft in a standard 12-ft entry. All work being



**Fig. 2—Plan for bringing air to face, using 6 or 8 in. posts and cross timbers over the machine.**



**Fig. 3—Method of mining with alternate angle cut. Numerals indicate progressive steps of advance.**

done at present is development work on a three entry system, where shuttle car haulage is generally over 250 ft. The delays encountered waiting for shuttle car return limit the productive time the Colmol operates in a shift. In room work this down time should be steadily decreased, permitting the Colmol more productive operating time.

Some new means must be provided for carrying the coal away from the face to the main transportation system. We are working on this problem as, we are sure, are others. We are happy to report that we have made considerable progress on paper and on models. We should soon be able to disclose a satisfactory answer to this problem—a new method which should permit some 70 to 80 pct Colmol operating time.

To gain the ultimate productive capacity with a continuous miner, new mine layouts will be desirable. During a recent visit with several of the larger operators it was indeed interesting to note that mine operators are ready and willing to consider new and different mining methods. Such an attitude on the part of mining men bids well for the ultimate success of continuous miners.

# The Young Mining Engineer in the Coal Industry

by M. D. Cooper

UNDERGRADUATES in mining engineering may be prepared for work by giving them sound instruction in the courses generally considered essential to the profession. The industry is not deeply concerned about the details of those courses. The average man in the coal industry does not wish to insist upon a rigid program. Therefore, he differs little from those in the teaching profession who evidently are not unanimous in their opinions, or all college catalogs would be alike. For the good of the profession, it is just as well that there should be differences in regard to details. It appears that students graduating in mining engineering from the accredited institutions receive similar instruction.

---

M. D. COOPER, Member AIME, is Director, Mining Engineering Education, National Coal Association, Pittsburgh, Pa.

*This paper is not a statement authorized by any organization within the coal industry, but is the consensus of opinion of a number of persons of authority in the industry and at various localities.*

*AIME Columbus Meeting, September 1949.*

*TP 2876 J. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Oct. 10, 1949.*

---

It is taken for granted that the graduate will have a good understanding of English, mathematics, mechanics, electricity, chemistry, physics, geology, and surveying, in addition to his major courses in mining. Somewhat belatedly, industry hopes he will have had at least an introduction to the subject of labor relations, the importance of which is only too clear at present.

The coal industry expects, of course, that students in mining engineering will be taught the strictly mining subjects by men who have had practical experience in the mines and who keep themselves well informed in regard to current methods.

While the undergraduate is subject to the control of members of the teaching profession, industry expects him to be trained in certain ways that are not a part of his textbooks, but can be made an inseparable part of his development by the skillful supervision of his teachers.

Of the desired characteristics, dependability is of the utmost importance. Probably most employers would overlook certain short-comings if the young graduate demonstrated that he was thoroughly dependable. If he always appeared at the right place at the right time with the proper equipment, he would soon be well established as a welcome member of his organization. The graduate who gets a reputation for being undependable will have little opportunity for advancement.

Closely allied to dependability is loyalty. Athletic teams and social groups in college tend to develop loyalty which may well be carried over into industry. This does not mean that the graduate has to be satisfied with customary practices. The average manager is glad to see the graduate make constructive criticism as long as he demonstrates his loyalty at the same time. It is important that his loyalty

keeps him alert and ready to take helpful action for the benefit of his organization, and especially to stand with it during times of stress.

With or without an introduction to labor relations in college, the graduate is expected to develop ability in this most important field. Beginning with himself, he will find it essential to deal agreeably with his immediate associates. Getting along in friendly fashion with his own small group will be a great help as his responsibilities increase and he is required to deal with larger numbers of persons. On a higher scale, his interest in his community may grow at the same time by voluntary work in any one of a great number of useful activities.

Industry expects the graduate engineer to be a mature man at the time he gets his first job. Supposing that he has better than average intelligence, industry expects him to continue to grow intellectually and to fit himself for responsible jobs when they are offered to him. For this reason, employers are apt to look over his college record to see what he did that would indicate his fitness for leadership. There is interest in knowing what he did beyond the requirements.

As evidence of his mental growth, it is expected that the graduate will do independent thinking; that he will not take too much for granted. When he reads a report, he should develop the ability to see whether the subject is new or whether it is just a description of an old method that has been superseded by something better.

For the same reason, the graduate should be able to accept conditions that have been arrived at by sound experience rather than cling to something else that seems better in theory. In this connection, it may be remarked that the ability to operate successfully a personal budget will be noteworthy, as it may be assumed that a man who knows how to conduct his own affairs will be prepared to assume the larger responsibilities of industry.

Membership in AIME will indicate to the employer that the graduate is interested in the mining industry as a whole. Therefore, it is good evidence of something more than a local outlook.

Quite apart from college training and mental ability, the newly employed graduate will be expected to be willing to do hard manual labor for a time. This will give him an understanding of the actual conditions of work done by those he supervises later. He will gain their confidence and be able to see that the work is carried on in a safe and efficient manner. Part of this experience may be acquired in his summer vacations during his undergraduate career. Such work would make a favorable impression on a prospective employer, especially if the graduate showed a willingness to continue it until he was prepared for something better.

To summarize, the man in authority in the coal industry will not quarrel with the professor of mining engineering over details of curriculum. He will be pleased if the school sends him graduates who possess a good foundation in the courses studied, and who may be depended upon to do their work faithfully and intelligently. Such men will be ready when the time comes to assume their places as leaders of an essential industry.



# Effect of Waste Disposal of the Pebble Phosphate Rock Industry in Florida on Condition of Receiving Streams

by Randolph C. Specht

**A two year study was made of the waste disposal of the pebble rock phosphate industry. Solid slimes are impounded in large settling areas and the process water is re-used. Clear effluent was not found to be toxic to fish or animal life in field tests or in controlled experiments.**

**Survey:** The pebble rock phosphate mining and washing operations in Florida (fig. 1) are along the river basins of the Alafia to the west and the Peace to the east. The elevation of the area varies between 40 and 170 ft above sea level. Effluents from the operations find their way into one of the two river systems (fig. 2), the Peace River which flows into Charlotte Harbor and the Alafia into Tampa Bay, both along the Gulf of Mexico.

There are no major industries along these rivers below the phosphate operations. On the Peace River, the town of Arcadia, approximately 50 miles below the phosphate operations, obtains its water from the

---

RANDOLPH C. SPECHT, Member AIME, is Professor of Chemical Engineering, Research Engineer, Engineering and Industrial Experiment Station, College of Engineering, University of Florida, Gainesville, Fla. AIME Tampa Meeting, November 1949.

TP 2878 H. Discussion (2 copies) may be sent to Transactions AIME before August 31, 1950. Manuscript received March 8, 1949; revision received Oct. 31, 1949.

---

river. No other communities along the rivers use them as a source of water supply. Small communities, farms, and ranches are located along the rivers and both rivers drain vast swamp areas.

Complaints had been received at the office of the Chief Sanitary Engineer, of the Florida State Board of Health,<sup>1</sup> that both rivers were turbid at various times and that fishing in the Alafia River was "not as good as it used to be." Tests made by another agency on the dissolved oxygen content of the Alafia River revealed that the oxygen content, at times, was comparatively low.<sup>2</sup> However, it was not shown that the phosphate wastes had any effect upon the oxygen content of the stream.

The river waters are normally brown in color due to drainage from vast swamp areas and are very shallow, from 2 to 6 ft in depth. Turbid waters have been noted in streams coming from the present phosphate operations and also from streams along which there are no operations at the present time. It was learned from "old timers" that, when many more companies than at present were operating along both the Alafia and Peace Rivers forty or fifty years ago, no attention was paid to prohibiting the disposal of slimes along the river basins. During dry periods now in some of the river swamps where drainage ditches have been dug, it is indicated clearly that phosphate slimes were disposed of directly into the river swamps for many years past. Places have been observed where there were alternating strata of phosphate mud, leaf mold, and quartz sand.

The former practice of indiscriminate disposal of wastes accounts for some turbidity, particularly after heavy rains and washouts and during clearing of land and digging of drainage ditches. During a six months' period in which a large tract of land was being cleared and drainage ditches dug, a small stream (Six Mile Creek), which previously had been clear, showed a turbidity of 600 ppm and remained turbid until the clearing was completed; after which the stream again flowed clear.

An interview with the operator of the municipal water plant, which is about 50 miles below the phosphate operations on the Peace River and which used this river as a source of supply, revealed that there had been no taste in the water for the past ten years, other than that occasionally encountered from increased growth of algae.

The operations of each of the cooperating com-

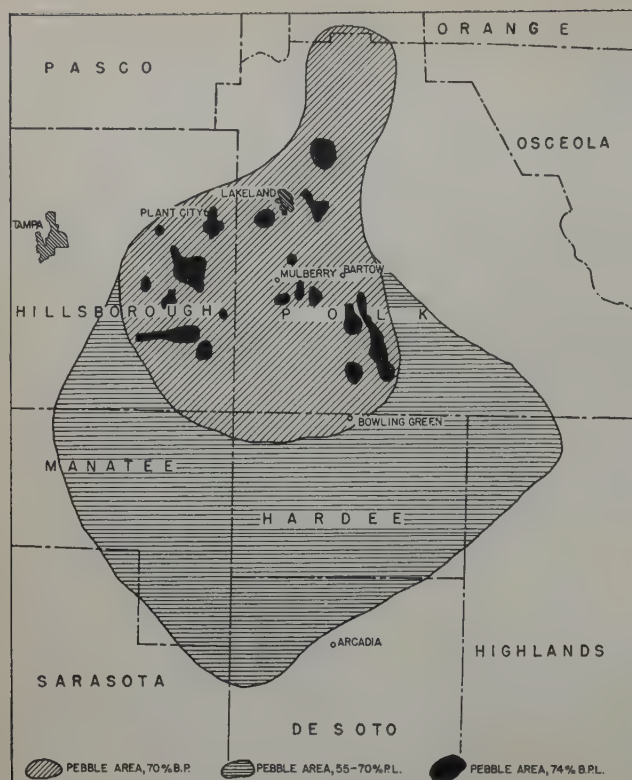


Fig. 1a—Map for insert in fig. 1b.

panies were visited and studied. A flowsheet of the processes was made and all materials used in the processing recorded. Water consumption data were collected and recorded and a sketch was made of the settling areas and waterways noting the direction of flow. All spillways and effluents were located on the sketches, as well as drainage ditches and streams flowing through and from the operation.

Sampling stations were located on all streams and drainage ditches into which effluents were discharged. These stations were above and below the operation so that the condition of the water downstream from the operation could be compared with that upstream.

No record of a fish census of either the Peace or Alafia Rivers could be found at the State or Federal Agencies.

**Clay Suspensoids:** Suspensoids in the water from mining and washing operations are due to clay in the phosphate deposit, from 15 to 40 pct being clay. The survey has indicated that slimes from mines in the Peace River Valley will settle if given sufficient time and that settling of these slimes is feasible and accomplished in the large areas provided. At some of the operations in the Alafia River Basin, a slime was encountered which did not settle in a reasonable length of time in the areas provided or even in a larger area, and coagulants were used to facilitate removal of the suspensoids.

An examination of the geological formation<sup>8</sup> of these areas does not reveal any unusual difference in the formation or drainage areas that might account for the difference in the physical characteristics of the clay. Prospecting samples taken in the Alafia area are similar in appearance to those obtained in the Peace Valley Section.<sup>4</sup>

A partial chemical and spectrographic analysis was made of a representative semicolloidal slime



Fig. 1b—Florida land pebble phosphate district.

(From U.S.G.S. Bull. 934, plate 5.)

taken from an operation in which the slime settles in the settling areas provided, and from one in which the slimes do not settle readily. The slimes were separated by filtration and dried at 100°C before being analyzed.

The analyses are shown in table I.

Sufficient information was not obtained from the various analyses to determine the cause of the non-settling characteristics of some of the slimes. The chemical analyses indicated that it may be partially due to the presence of free alumina in kaolinite. However, a thermal analysis did not identify a characteristic mineral but rather what appeared to be a mixture.<sup>5</sup> The slimy characteristics of the suspended clay would seem to indicate the presence of hydrous oxide lattices in the form of hydrogels of high bound water content. The nonsettling characteristics indicate the presence of a negatively charged colloid, and its presence was shown in a determination of migration velocity in the classic U-tube conductivity cell used by Burton.<sup>6</sup> An electrophoretic study of the migration velocity of the colloidal particles, using the Mudd cell under a microscope, showed that the nonsettling clay had a greater negative charge than the one which settled. By the addition of a proper electrolyte the charge was reduced and the clays agglomerated and settled.

An examination of the flotation chemicals used and analyses of the water in which the clay was

Table I. Analyses of Semicolloidal Slimes

Elements Analyzed for: <sup>a</sup>	Reported as:	Slimes Which Settled	Slimes Which Did Not Settle
Si	SiO <sub>2</sub>	33.84	27.24
F	F <sub>2</sub>	0.46	0.80
Al	Al <sub>2</sub> O <sub>3</sub>	26.32	31.30
Fe	Fe <sub>2</sub> O <sub>3</sub>	4.18	1.58
Ca	CaO	17.36	20.22
Mg	MgO	1.28	1.25
P	P <sub>2</sub> O <sub>5</sub>	6.06	6.38
	Ignition loss	10.21	11.62

<sup>a</sup> The spectrographic analysis, which did not cover the entire band, showed in addition to the elements listed above, the presence of small amounts of manganese, chromium, vanadium, barium, strontium, and titanium.



suspended revealed that when the clay settled, enough electrolyte, in this case calcium sulphate, was added to the water from the process to give sufficient neutralization of the negative charge to allow coagulation and settling. Where nonsettling slimes were encountered, they could be made to settle by the addition of the proper electrolyte.

The cost of chemical coagulants for treatment of the water, where required, varies with the particular deposit and process. Estimated costs have varied between \$10.00 and \$70.00 per 16-hr day for a flow rate of 18,000 gpm.

The analyses and examination of the suspensoids do not indicate any poisonous or deleterious material in them.

**Turbidity:** The greatest problem of waste disposal in the industry is the disposition of slimes which are removed from the process waters so that the waters may be recycled through the system. When the slimes do not settle completely, turbid waters are recycled throughout the plant and the effect on pro-

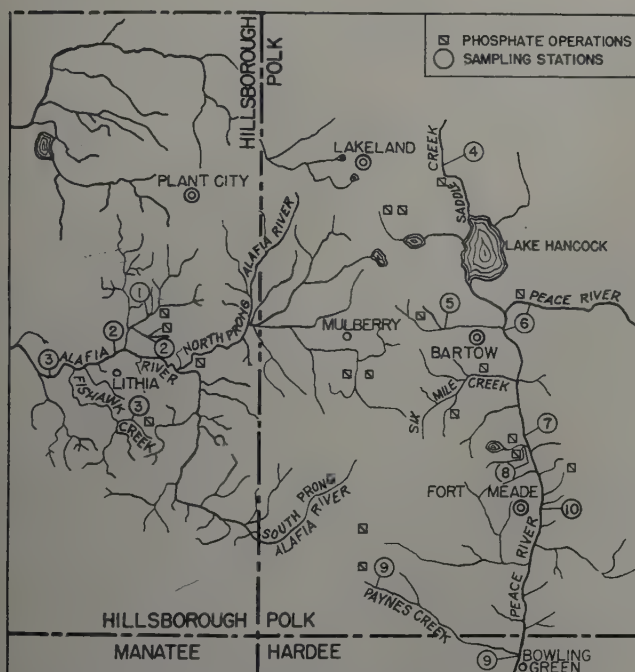


Fig. 2—Map of Alafia and Peace River Basins.

duction is lower quality material and increased costs. It is therefore of considerable importance to have clear water for the washing and flotation operations, and large hydroseparators and settling areas are provided for this purpose. In addition to the interest in economy of operation and production, the industry is also interested in preventing turbid waters from entering the water systems of the State.

At the present time all of the plants recycle the water and during normal rainfall and operation an estimated maximum of 10 pct of the total water used may be discharged from the settling area. During heavy rainfalls, 2 to 4 in. within a few hours, the water falling into the settling areas must be discharged through the spillways to prevent washout of the dams. A heavy rainfall does not mean necessarily that turbid waters will be discharged from the settling areas; generally the settling pond is not stirred up and only clear water is discharged.

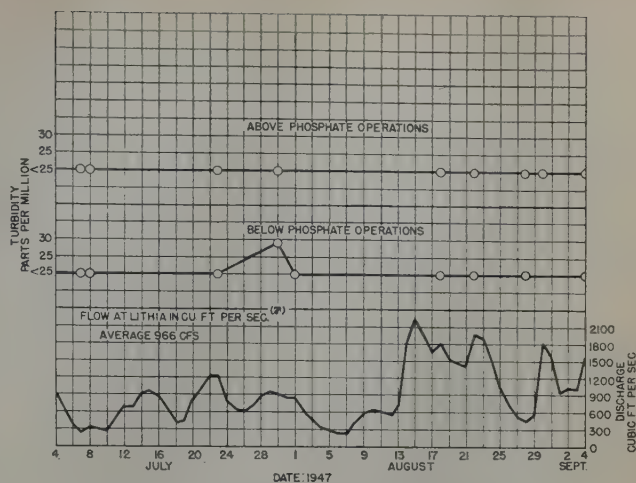


Fig. 3—Turbidity and flow of Alafia River.

As in most industrial operations, emergencies and errors of personnel sometimes result in the discharge of turbid waters.

During the period July 1 to Sept. 4, 1947, samples of water were taken from the water systems of the operations and from the streams above and below the phosphate operations. Turbidities of the samples were measured in the Jackson turbidimeter and recorded. Since the turbidimeter does not give values less than 25 ppm, any value less than this was recorded as "less than 25 ppm." The water in the streams and rivers is brown in color due to organic material, and no correction was made on turbidity for color.

Fig. 3 shows the turbidities of the Alafia River for the period. The flow of the river at the U. S. Geological Survey gauging station at Lithia, is shown on the graph.

One of the operations, at station 1, along this river was abandoned a few weeks before sampling was begun and another, at station 2, on July 15. Several operations remained along the North and South Prong.

On July 30, a turbidity of 29 ppm was recorded on the sample taken from the river below the phosphate operations. On this date the North and South Prong were clear; operations on the branch that was turbid were abandoned and the source of the turbid waters was not located.

It is of interest to note that during the time of heavy rainfall and high waters there was no marked increase in turbidity of the waters in the river.

One of the operations that encounters slimes that are difficult to settle is located on Fishhawk Creek. At this operation lime is used to aid the coagulation and settling of the suspensoids. Normally all the water is recycled throughout the system but in times of heavy rainfall it is sometimes necessary to discharge water into the creek, and the water may or may not be turbid, depending upon many factors.

Fishhawk Creek flows from a swampy lowland area approximately two miles above the effluent of the plant. The creek is very turbid during the rainy season and shows the effects of soil erosion. The flow of the creek above the phosphate operation during the dry season was estimated at less than 1 cu ft per sec; during heavy rains it became a rapidly flowing stream.

Table II shows the turbidity readings obtained from samples taken from the creek at points above and below the phosphate operations on the dates

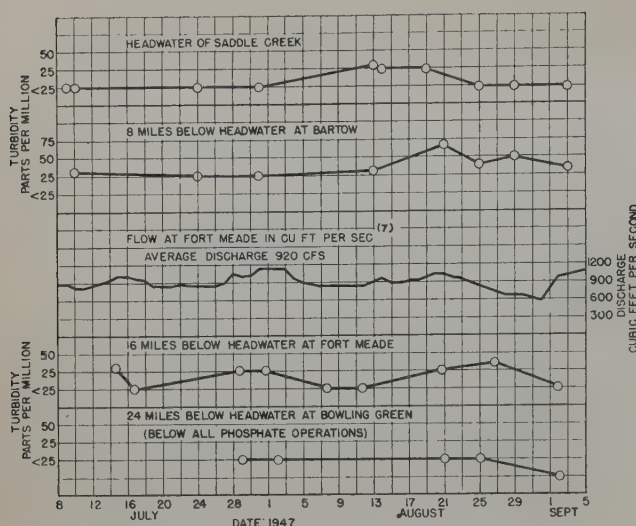


Fig. 4—Turbidity and flow of Peace River.

indicated. It is noted, with one exception (July 23), that the stream above the operation was more turbid than below. There is no doubt but that the industry may add to the total load on the stream but it is to be observed that the present industry is not responsible for the entire condition of this stream.

Samples taken from the Alafia River, two miles below the mouth of Fishhawk Creek, showed turbidities of less than 25 ppm during the same period, thus indicating that the small amount of turbid water from the creek did not materially affect the turbidity of the river.

Fig. 4 shows the turbidity of samples taken from the Peace River Basin and the flow of the river at Ft. Meade. Turbidities are shown for the stream above phosphate operations (Saddle Creek) and at approximately eight mile intervals along the stream. The last station was about four miles below the last operation. The source of a part of the turbidity of the stream, particularly at Bartow, was traced to phosphate operations. There appears to be no direct relationship between turbidity and flow in the stream.

Over 500 measurements were made on samples of the rivers and tributaries during the period of the extensive test. From data obtained it was calculated that with the dilution of the effluent of 1 to 29.5, the receiving river increased in turbidity in an amount equal to 2.5 pct of the turbidity of the effluent; 25 pct of the suspended solids were deposited on the stream bottom; and the remainder, 75 pct, was carried in suspension a distance of at least three or four miles. It would not follow necessarily that this is always the case, because the nature of the suspended matter and the diluting water may be such that there would be coagulation and precipitation in comparatively short distances.

**Dissolved Oxygen and Biochemical Oxygen Demand:** During the period of the test, 519 samples were analyzed for dissolved oxygen (D.O., elementary oxygen dissolved in a water) and biochemical oxygen demand (B.O.D., the amount of oxygen required to oxidize biologically the organic matter in a waste over a stated period of time) by the standard method.<sup>8</sup> All results are reported in parts per million and the B.O.D. as the "five day-twenty degree Centigrade." Normal river water,

into which the effluent or waste flowed, was used as a dilution water for the B.O.D. determination.

In all cases the D.O. content of the effluent of settling areas was found to be higher than that of the receiving river. The B.O.D. content of the river water at Bowling Green, below the operations, was found to be lower than at any point along the stream, except at station 6 where it was the same.

Although the B.O.D. of the effluents of some of the operations was found to be 1 ppm higher than that of the receiving streams, the pounds of B.O.D. added is comparatively small, amounting to only 0.15 pct of the increase in the load in Peace River between Bartow and Ft. Meade, as shown in table III. Many small streams into which no effluents were flowing were found to have a higher B.O.D. than the effluents.

Table II. Turbidities of Fishhawk Creek

Date 1947	Above Operation, Ppm	Below Operation, Ppm
July 1		136
July 2	1,450	190
July 3	3,500	220
July 7		160
July 8		140
July 23	31	63
July 30	170	140
Aug. 18	250	35
Aug. 22		50
Aug. 28		215
Sept. 1	160	
Sept. 4	110	< 25

The results indicate that the dissolved oxygen content of the stream is improved and the biochemical oxygen demand is not materially changed by the effluents of phosphate operations.\*

\* Detailed results of the D.O. and B.O.D. tests will be published elsewhere.

**pH of Streams and Effluents:** The pH of effluents of the operations was found to be substantially the same as that of the rivers. Some of the small streams above the phosphate operations were more acid than the rivers, particularly the streams at the head of Alafia River and at the head of Fishhawk Creek. In Fishhawk Creek the effluent of the phosphate operation had the effect of bringing the pH of the water near to the normal for the river into which it emptied. Table IV shows the minimum and maximum values of pH in the streams. Over 500 determinations were made during the test. The glass electrode was used for all determinations.

Table IV shows a wide variation in the minimum and maximum values of pH from all the operations. However the pH of each individual operation varied only slightly, and the effect of the effluent on the pH of the receiving stream was not measurable except in Fishhawk Creek where the effluent was considered to be beneficial.

**Laboratory Tests on Minnows and Small Fish:** Laboratory tests were made on live minnows which were placed in water containing flotation chemicals and in waters taken from the effluents of various phosphate operations. When flotation chemicals were used, the chemicals were added to water taken from the Peace and Alafia Rivers in an amount such that the dilution would be the same as in a plant, provided all the flotation chemical passed into the effluent of the settling area. This, however, is not the case in any of the operations as is shown later,



Table III. Increase in D.O. and B.O.D. of Peace River due to Effluent from Phosphate Operations

	Flow <sup>a</sup> in Cu Ft per Sec	D.O., Ppm	D.O., Lb per Day	B.O.D., Ppm	B.O.D., Lb per Day
River at Bartow	683	2.74	9,750	1.4	5,175
Effluent No. 7 (16 hr day)	3.3	6.82	12	2.1	3.5
Effluent No. 8 (16 hr day)	12.2	5.88	36	2.4	1.5
River at Ft. Meade	920	4.24	21,050	1.6	8,110

Increase in flow due to combined effluents, 4.3 pct  
Increase in D.O. due to combined effluents, 0.15 pct  
Increase in B.O.D. due to combined effluents, 0.15 pct

<sup>a</sup> Flow of river based on average flow for the period July 1 to Sept. 4, 1947.

All tests were made at 19.5° to 20.5°C (67° to 69°F). Glass jars were used as containers. Water was aerated continuously and kept within the temperature range by placing the jars in a constant temperature water bath. The test water was not changed during the test. The fish were fed periodically with ground puppy (dog) food.

Minnows used for the tests were obtained locally from a swamp area and were the *Gambusia affinis holbrooki*, *Molliensis latipinnis* and *Heterandria formosa*. One small bream was used in one of the tests.

The concentration of flotation chemicals used in the test waters is shown in table V.

**Results:** When minnows were placed in the test water containing the chemicals shown in table V, there was a complete kill after 48 hr. When the fuel oil, tall oil and kerosine were emulsified in the test solution by mechanical means, the kill was reduced to 6 hr, probably due to thorough dispersion of the tall oil in the water. When only amine acetate was used in the tests, the minnows lived for nine days, the duration of the test.

When samples were taken from (1) phosphate float water, and (2) from a flotation water settling area and used for a test, there were no kills of test animals during the 25 day period of the test. In a test using water from a settling area having a turbidity of 50 ppm, there were no kills during a 14 day test.

Table IV. pH of Rivers and Streams

	Mini- mum	Maxi- mum
Alafia River at head (station 1)	5.9	6.9
Alafia River at Lithia (station 2)	6.1	6.9
Alafia River (station 3)	6.8	6.8
Fishhawk Creek above phosphate operation	4.8	5.6
Fishhawk Creek 2 miles below operation	5.8	7.1
Fishhawk Creek phosphate effluent	6.1	6.6
All phosphate effluents in Alafia River Basin	6.4	7.1
Peace River at head (station 4)	6.0	6.4
Peace River at Bartow (station 6)	6.1	6.3
Peace River at Ft. Meade (station 10)	6.4	6.8
Peace River at Bowling Green (station 9)	6.3	6.4
All phosphate effluents in Peace River Basin	6.1	7.4

**Conclusions from Laboratory Tests:** The results of the laboratory tests indicate that tall oil is the most toxic of the chemicals used at the dilutions tested. Amine acetate in the concentration used was not found to be toxic. The change in pH due to the use of caustic and sulphuric acids was not sufficient to affect noticeably the test animals. The effect of fuel oil and kerosine alone was not studied, because neither was observed in the wash waters.

Water from effluents of settling areas did not appear to be lethal to the animals tested. This was also true of waters taken directly from the flotation process that had not been diluted with other plant water or mixed with water containing the slimes. (A common method for removal of oil is by entrainment in flocculated suspended material.)

Toxic material used in the flotation process does not appear to be present in the water in amounts sufficient to be lethal to the animals tested.

**Tests for Toxic Material in Effluent Water:** The laboratory test on minnows indicated that the only likely toxic material used in the processing was the tall oil. Waters from the flotation process, however, were not found to be toxic to the test animals.

Laboratory tests and analyses<sup>9</sup> were made for concentration of tall oil on flotation waters, floated mineral, and sand rejects. The tests showed that if there was any tall oil in the flotation water it was well below the minimum lethal dose of 1 ppm.<sup>10</sup> Tests made on the sand rejects showed that some of the resin soap adhered to it, but the amount found was only a trace. The floated mineral, however, gave tests that indicated that practically all of the converted tall oil was held by the mineral, as would be expected, since the tall oil was used to aid in the floating of the ore.

Table V. Concentration of Flotation Chemicals in Water Used in Tests

	Lb per Gal
Fuel oil	0.000665
Tall oil	0.00016655 <sup>a</sup>
Caustic (20 pct)	0.000133
Sulphuric acid (60° Be)	0.0002
Kerosine	0.00004
Amine acetate (5 pct solution)	0.00001

<sup>a</sup> Equivalent to 20 ppm.

Waters from all flotation processes and settling areas were found to contain excess soluble calcium that would react with any of the tall oil, or the sodium resinate formed, and precipitate the insoluble calcium soap. Therefore, further assurance was made that no toxic resinate was present in the effluent waters; the insoluble calcium compound being nontoxic to fish,<sup>10</sup> apparently because it is not available for absorption through the gill membrane.

**Test on Live Fish:** Fish used in the tests conducted in the waters in, or from, the settling areas of phosphate operations were supplied through the courtesy of John F. Dequigne, Chief Fisheries Biologist of the Game and Fresh Water Fish Commission of the State of Florida, who consulted and assisted throughout the entire test.

**Test Animals:** The fish used in the tests were of the families of fresh water fish normally found in the streams and lakes in the State. The fish were seined from a fresh water lake and kept in captivity at the Game and Fresh Water Commission's hatchery near Eagle Lake for approximately one month before the tests were made.

The size and identification of the test animals were recorded, along with the temperature of water, pH, turbidity, and feeding schedule. Fish used were: bass, blue gill, black spotted sun fish, shell cracker, channel cat, black crappie, warmouth, speckled bull-head, golden shiner and sucker.

Fish used in the test were delivered to the site of

the test in the commission's tank truck. The water in the tank was aerated during the time the fish were in transit.

**Traps:** During the entire test the fish were kept in traps placed in the test waters. The traps were made by covering a cypress wood frame with 18 mesh plastic screening wire. The traps were 6x3x3 ft, entirely closed except for a 10x10 in. hinged opening on one side. The traps were immersed in water to a depth of approximately 3 ft, allowing the side containing the opening to be near the water level.

**Conditions and Results of the Tests\*:** All tests

\* Details of the laboratory tests on minnows and the tests on live fish will be published elsewhere.

were conducted in the same manner but at different operations. Traps were placed in settling areas, recycling water ditches, and in the effluent at five different operations. A control test was made in a stream containing no phosphate waste water. In all the tests except one, the pH of the water was between 7.2 and 7.4, the temperature at 77° to 80°F, and the turbidity less than 25 ppm. In one test the pH varied between 8.75 and 9.7, the turbidity between 78 and 130 ppm, and the temperature between 74° and 77°F.

In the 96-hr test on live fish, which is that recommended for weak toxicity,<sup>11</sup> there was a 100 pct survival in all the tests and in the control. The fish were left in the traps for an extended period beyond the test time limit and continued to live with but a few fatalities that were due to the effects of cannibalism, parasitism, and other aggressive forces normally following prolonged confinement.

It was observed during an extended period of the test where the pH and turbidity were high, that the fish lived in the cooler water that had been treated with lime, even though the pH was beyond the optimum range for fish<sup>12</sup> and the turbidity high.

The effect of turbidity on the plankton in the rivers was not investigated since the turbidity was shown to be practically constant at points above and below the effluents of the phosphate operations during the time of the test.

During observations of the tests, many minnows and live fish were seen in the water outside the traps. Of the fish seen, those identified were shiner, bass, warmouth and cat.

**Conclusions:** The waste disposal of the soft pebble phosphate rock industry in Florida during the year 1947 amounted to over six million tons of quartz sand and clay. This material was deposited in large settling areas provided by the individual operators.

Clear effluents from the operations were not found to be toxic and fish lived for 30 days in waters having a turbidity of 80 to 130 ppm.

Provided the wastes from the operations do not cause unreasonable turbidity in the receiving rivers, the wastes from the soft pebble phosphate rock industry are not considered to cause pollution. It is believed that the clear effluent waters from the operations are beneficial to the rivers.

**Acknowledgment:** Funds required for the study were supplied by the following phosphate companies: American Cyanamid Co., The American Agricultural Chemical Co., Coronet Phosphate Co., The Davison Chemical Corp., International Minerals and Chemical Corp., Pembroke Chemical Co., Swift and Co., and Virginia-Carolina Chemical Corp.

The author is grateful for the permission obtained from the companies to publish the findings of the

study and to the managers and personnel who generously assisted.

The tests on live fish could not have been made without the assistance of John F. Dequine, Chief Biologist of the Game and Fresh Water Fish Commission of Florida, who with his staff supplied the fish for the test and aided and advised throughout the tests.

The data on D.O., B.O.D., and pH were collected by Frank Seely, Assistant Research Engineer. S. L. Besvinick and W. E. Herron, Jr., Graduate Assistants, aided in the collecting of data and in making some of the analyses. The data on flow were obtained from A. O. Patterson, District Engineer, U. S. Geological Survey, Ocala, Fla.

Thanks are due David B. Lee, Chief Sanitary Engineer, Florida State Board of Health, who was responsible for the starting of the study, and to him and his staff for valuable information.

Acknowledgment is given to W. B. Hart, who loaned us his personal copy of "The Evaluation of the Toxicity of Industrial Wastes Chemicals and Other Substances to Fresh-water Fishes," from which valuable information was obtained; Elmer Higgins, Chief Biologist of the U. S. Fish and Wildlife Service, for study of data on fish tests and advice; Kenneth W. Coons, Head, Department of Chemical Engineering, University of Alabama, for information on tall oil toxicity; to Harry Gehm, Technical Advisor, National Council for Stream Pollution, for information on tall oil and tests; to G. R. Grantham, Professor of Sanitary Engineering, for review of the data on D.O. and B.O.D.; and to D. L. Emerson for identification of algae and diatoms and helpful criticisms.

The writer is indebted to R. A. Morgen, Director of the Florida Engineering and Industrial Experiment Station, where much of the work was done; to W. H. Beisler, Head, Department of Chemical Engineering; A. F. Greaves-Walker, Head, Nonmetallic Minerals Section; to fellow members of the staff for assistance, criticism and review; and to the many others who have given aid and assistance.

## References

- <sup>1</sup> David B. Lee, Chief Sanitary Engineer, Fla. State Board of Health. Private communication.
- <sup>2</sup> John W. Wakefield: Stream Pollution Survey of Alafia River. Fla. State Board of Health Rept. (Nov. 22, 1945 to Jan. 31, 1946).
- <sup>3</sup> Richard A. Edwards, Prof. of Geol., Univ. Fla. Private communication.
- <sup>4</sup> E. W. Green, Mgr., Min. Separation, North American Co., Lakeland, Fla. Private communication.
- <sup>5</sup> R. D. Walker, Res. Engr., Eng. and Ind. Expt. Station, Univ. Fla. Private communication.
- <sup>6</sup> E. F. Burton: Colloid Symposium Monograph. (1926) IV, 132.
- <sup>7</sup> A. O. Patterson, Dist. Engr., U.S. Geol. Survey, Ocala, Fla.
- <sup>8</sup> American Public Health Assn.: Standard Methods for the Examination of Water and Sewage. 9th ed.
- <sup>9</sup> National Council of Stream Improvement, Aquatic Biology Res. Rept.: Proposed Methods of Analysis for Resin Acid Soaps.
- <sup>10</sup> Willis M. Van Horn, Res. Assoc., The Institute of Paper Chemistry. Private communication.
- <sup>11</sup> Hart, Doudoroff, and Greenbank: The Evaluation of the Toxicity of Industrial Wastes, Chemicals and Other Substances to Fresh-Water Fishes. p. 123.
- <sup>12</sup> M. M. Ellis: Industrial Wastes and Fish Life. Proc. First Ind. Wastes Conf. Purdue Univ. (1944) pp. 126-134.



# Sillimanite in the Southeast

by Kefton H. Teague

**Attempts to locate domestic supplies of sillimanite have been unsuccessful until recently. This paper describes recent discoveries of sillimanite-bearing schists in the Southeastern States, with emphasis on geology of the deposits, mineral composition, origin, and reserves. Statements concerning possible methods of mining, milling, and utilization of sillimanite are given. Results from refractory tests on a laboratory scale are summarized.**

SINCE 1917, when synthetic mullite was first used in spark plug porcelains, there has been a continued search for natural sillimanite. During the recent war this search was intensified, not only for sillimanite but also for any materials which could be used satisfactorily as a substitute for the imported kyanite from India. The Indian massive kyanite does not decrepitate during calcination as does our domestic bladed kyanite, thus a coarse grog can be obtained from it.

---

KEFTON H. TEAGUE is Associate Geologist, Tennessee Valley Authority, Atlanta, Ga.

AIME Atlanta Meeting, October 1947.

TP 2879 H. Discussion (2 copies) may be sent to Transactions AIME before Aug. 31, 1950. Manuscript received Dec. 1, 1947; revision received March 27, 1950.

---

Some of the most promising domestic materials suited for high temperatures are limited in quantity. Massive topaz from Chesterfield County, S. C., showed promise of possessing desirable refractory properties; but the action of combined fluorine, liberated during calcination, upon equipment and surroundings, as well as a limited supply of ore and the presence of impurities discouraged its use. Residual lumps and boulders of massive kyanite, similar to the material imported from India, occur in Georgia. This material, like the massive topaz, is of limited quantity. Andalusite from California and dumortierite from Nevada are satisfactory high-temperature refractory materials. Their uses have been restricted to specialized products since the

location and character of the deposits are such that large production is impractical.

Sillimanite, like kyanite and andalusite, has the theoretical chemical composition of  $Al_2SiO_5$ . It has a hardness of 6 to 7; vitreous, silky to subadamantine luster; specific gravity of 3.2 to 3.3; and is gray to bluish gray in color. Under the binocular microscope, the crystals are transparent. The mineral may occur as dense, fibrous mats (fibrolite) composed of fibrous, sometimes radiating hair-like crystals in schist which are associated with igneous intrusions. Deposits of this type have widespread distribution, but none discovered to date appears to have commercial possibilities. Sillimanite also occurs as bundles of crystals disseminated in biotite schist, as noted in the Hart-Elbert-Madison County, Ga., area and in some of the South Carolina and North Carolina deposits. This type of material has been called prismatic sillimanite by the U. S. Bureau of Mines. Deposits of this variety appear to offer commercial possibilities.

When heated above  $1650^{\circ}C$ , sillimanite expands about 6.5 pct and is converted into a mixture of mullite ( $3Al_2O_3 \cdot SiO_2$ ) and vitreous silica. This is a stable mixture which has a specific gravity of about 3.15.

**General Geology:** Sillimanite (fibrolite) is a mineral common to the pre-Cambrian metasediments of the Southeast. It occurs throughout the crystalline schists which lie between the Fall Line on the southeast and the established Paleozoic rocks on the northwest (fig. 1). In addition to the metasediments, this area contains pre-Cambrian igneous

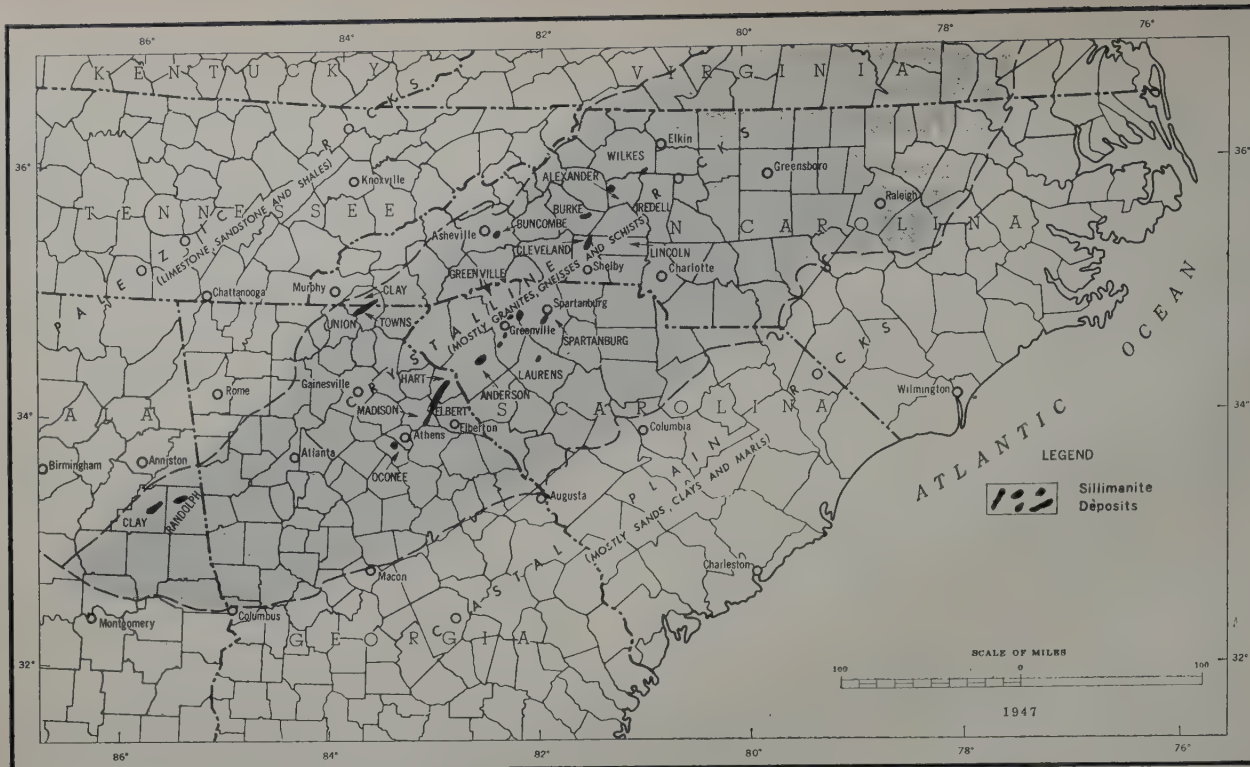


Fig. 1—Distribution of the known sillimanite deposits in the Southeastern States.

gneisses and many granites of probable Paleozoic age. The metasediments, in most cases, appear to be older than the later granites. Schists are intruded by the granites and the effects of granitic injections are exhibited in the schists where coarse recrystallization is superimposed upon the normal grain size at the contacts between the schists and granites or pegmatites.

The widespread sporadic occurrence of sillimanite, as fibrolite, is due undoubtedly to the action of hot solutions from later granites, pegmatites, and quartz veins upon a rock which was originally high in alumina.

In the Davy Mountain area of Georgia and North Carolina, much of the sillimanite is of the fibrolite variety. Other deposits here appear to have been formed from the alteration of kyanite. The fibrolite occurs in a quartz-muscovite schist which contains more or less flake graphite and is associated with bladed kyanite.

Sillimanite, found in the Cliffside-Elkin belt in North Carolina, is mainly of the fibrolite variety; however, at a few places, bundles of crystals weighing up to a pound may occur (fig. 2). This sillimanite-bearing schist occurs in a low southeastward dipping shear zone ( $20^\circ$ ) which has a rather constant strike of N.  $20^\circ$  E., as contrasted with the regional strike of N.  $60^\circ$  E. The rocks consist mainly of sheared granites, granite gneisses, and quartz-biotite-muscovite schists. The shear zone contains numerous stringers of pegmatite and quartz veins which are postshear in age. It is believed that this sillimanite had its origin in mobile solutions emanating from these small pegmatites and has replaced quartz, biotite, muscovite, kyanite, and garnet.

The better-known deposits of sillimanite in Greenville, Spartanburg, and Anderson Counties, S. C., and in Hart, Elbert, and Madison Counties, Ga.,

occur in biotite-muscovite schist as roof pendants over younger granite. The sillimanite-bearing schist, as well as the younger granites, has been intruded by later pegmatites and quartz veins. In this area biotite is the dominant type of mica present in the schist; it is older than the sillimanite and occurs as inclusions in the sillimanite crystals. Along contacts with granites and pegmatites, the biotite shows effects of contact metamorphism in that it has been recrystallized into coarser flakes. Sillimanite crystals are larger and more abundant near the schist-granite contact, suggesting that contact metamorphic processes were active in the formation of the sillimanite. Also, near the granite-schist contact, coarse flake graphite occurs locally in granite, probably representing a residue from the assimilated sediment.

All of the sillimanite deposits found to date exhibit more or less secondary sericitization. Inasmuch as sillimanite is considered a high-temperature mineral formed in a deep zone, the later alteration to sericite is ascribed either to hydrothermal solutions which affected the sillimanite in the last stage of igneous injection which produced the sillimanite or to those solutions which came from later granites and pegmatites that were injected into the schist.

Locally, tourmaline occurs in the sillimanite schist, in which case it, like the sillimanite, exhibits similar lineation and relation to enclosing schist minerals. Such occurrences also suggest the high-temperature origin for sillimanite and its direct relation to igneous activity.

**Previous Work and Distribution:** In 1943, Smith<sup>1</sup> reported sillimanite of possible commercial quantities occurring in South Carolina. Furcron<sup>2</sup> identified sillimanite submitted from Davy Mountain, Towns County, Ga., in the same year. Hudson,<sup>3</sup> in 1943 and 1944, studied numerous outcrops in Georgia





**Fig. 2—Bundles of sillimanite crystals disseminated in a biotite-muscovite schist, Burke County, N. C.**

Rule 6 in. in length.  
(Photo by Charles E. Hunter)

and South Carolina in which he discovered sillimanite. In September 1944, the writer, with A. S. Furcron, discovered a zone of sillimanite schist extending from Madison County northeastward across Elbert County into Hart County, Ga. Charles E. Hunter,<sup>4</sup> TVA, found sillimanite at various places in North Carolina, and in 1945, with Dr. W. A. White of the University of North Carolina, mapped a sillimanite schist zone which extends from Cliffside northeastward across North Carolina to the vicinity of Elkin. In 1944, W. T. McDaniel, Jr.,<sup>5</sup> TVA, reported sillimanite in mica schist from Randolph and Clay Counties, Ala.; however, the extent and richness of the Alabama deposits have not been investigated.

The writer has found, from investigations in Georgia, South Carolina, and North Carolina, that sillimanite in various forms and amounts occurs in almost all of the metamorphosed sedimentary mica schists and gneisses which have been affected by granite intrusions; however, the zones of major concentration are limited. Hudson regarded the sillimanite as occurring in a belt trending from Talbot County, Ga., N. 45° E. for a distance of 200 miles to Spartanburg County, S. C. Also he gives the maximum width of this belt of sillimanite-bearing rock as 15 miles; however, most of the sillimanite in this belt is of the fibrolite variety. If such generalities as to distribution are to be made, then, as has been pointed out previously, all of the metasediments should be included. The writer considers it best to record each deposit as a separate and independent unit.

**Exploration and Reserves:** In 1945, the U. S. Bureau of Mines prospected by diamond drilling two sillimanite properties in Georgia, one in Hart County and one in Elbert County; and three in Greenville County, S. C. Results of this prospecting have been published by the U. S. Bureau of Mines.<sup>8</sup> Also in 1945, the A. P. Green Fire Brick Co. of Mexico, Mo., prospected by auger and core drilling four properties in Hart and Elbert Counties, Ga. Results from four holes of a total of about 80 are included in the report referred to above.

In August 1945, the Georgia Department of Mines, Mining and Geology and the TVA dug a total of nine pits and trenches on the J. I. Jenkins property in Hart County, Ga. (see northeastern part of aerial photograph, fig. 3). These trenches were placed over a distance of about 1100 ft and a width of about 200 ft. Materials exposed in these openings indicate that about 60 pct of the total is sillimanite-bearing rock. A representative chip sample taken from a trench 176 ft long and representing 109 ft of sillimanite-

bearing rock contained 12 pct sillimanite (fig. 4). Another trench 900 ft southwest from the one mentioned above and along the same zone of sillimanite, with a total length of 141 ft, contained a thickness of 85 ft of sillimanite-bearing rock. This rock contains about 10 pct sillimanite. A 100-lb sample taken from a sillimanite-bearing zone about 660 ft northwest of the ridge prospected, but also on the Jenkins property, contained 15.8 pct sillimanite.

The area extensively prospected in Hart County covers about three fourths of an acre. Maximum relief is about 50 ft. This three-quarter acre area is estimated to contain above local drainage about 610,000 tons of sillimanite ore averaging approximately 10 pct sillimanite. It is estimated that about 85,000 tons of ore would be available per 10-ft interval below local drainage in the area described above. Additional ore of similar content on this and nearby properties will increase the above estimate to more than four million tons.

No estimate as to the amount of sillimanite ore is available for any of the South Carolina or North Carolina areas; however, in South Carolina, the area about two miles east of Pelzer in Greenville County appears to offer the greatest promise. In North Carolina, the Prospect Ridge area southeast of Morganton seems to offer the best opportunity in that State.

Anyone interested in prospecting for sillimanite should not rely too heavily upon the abundance of sillimanite float as an indication of a large underlying deposit. The sillimanite ore has much greater resistance to physical and chemical weathering than does the country rock. Frequently, a series of zones of sillimanite not more than a few feet thick will, upon weathering of the country rock, leave numer-



**Fig. 3—Aerial view showing distribution of sillimanite outcrops on the Dove, Jenkins, Dickerson, etc., properties. Southwestern Hart County, Ga.**





Fig. 4—Specimen showing sillimanite crystals disseminated in biotite schist, Hart County, Ga.

ous fragments of sillimanite in the soil over considerable territory, thus giving the impression that the entire area is underlain by sillimanite-bearing rock. Perhaps the best method of prospecting to overcome this difficulty is by pitting and trenching the deposits.

**Mining:** In all of the sillimanite-bearing areas examined, the deposits are of such character as to permit open-cut mining. Overburden at none of the deposits is excessive. Some writers have suggested that, where sillimanite does not crop out in sillimanite-bearing zones, it is covered by a thick mantle of soil. Actually, this condition is caused by the weathering of nonsillimanite-bearing rock; thus, it is useless to expect to find deposits of sillimanite concealed by deep residual soil. The sillimanite-bearing rocks generally form a definite ridge or hill which rises as much as 30 ft above the surrounding country; but only in the North Carolina deposits is there sufficient relief to permit the development of high quarry faces. In Georgia and South Carolina it would appear that several benches at about 20-ft intervals would be more practicable than one high face.

Any mining of the sillimanite deposits, from necessity, would require more or less selective mining. Granite and pegmatite intrusions in the deposits that exceed 4 ft in thickness probably could be eliminated in quarry, but the *lit par lit* character of granite and sillimanite will necessitate milling some barren rock.

**Milling:** Beneficiation tests of the sillimanite ore on a pilot plant scale, using a flotation process, have been conducted successfully by the U.S. Bureau of Mines Southern Experiment Station Laboratory at Tuscaloosa, Ala.,<sup>o</sup> and by the TVA Chemical Engineering Laboratory at Muscle Shoals, Ala. In addition to flotation, the Bureau employed tabling prior to flotation in an attempt to secure a coarser concentrate.

In general, the following processes were involved in concentrating the sillimanite ore: The ore was wet ground to —28-mesh and froth-floated. The flotation concentrate was dried and further processed by passing it over a magnetic separator. A large amount of the iron oxide present in the concentrate was removed by magnetic separation. In one case, the iron oxide content was reduced from 11.7 pct to only 1.1 pct, and acid leaching brought this to 0.5 pct. The surface iron oxide is that part removed by acid

treatment. It is probable that finer grinding would increase the amount of impurities removed. The acid-washed concentrate shows no iron oxide spots under the binocular microscope; thus, the remaining iron in the acid-treated concentrate probably occurs in small particles of biotite which were not liberated during grinding. Table I gives the screen analyses and table II the chemical analyses of non-magnetic Georgia and South Carolina sillimanite.

The iron content of the untreated concentrate is not great enough to cause any difficulty in its use in refractories; however, acid leaching might be required for specialized products. All ores treated thus far are taken from near the surface and therefore are highly oxidized. Secondary ferric oxide should not occur at or below water level.

**Laboratory Results:** No attempt is made to cover completely the results obtained from laboratory research upon the sillimanite concentrates because that phase of the problem is considered beyond the scope of this paper. It is sufficient to point out that laboratory results obtained upon the ceramic properties of sillimanite indicate that products can be made from sillimanite having original linear shrinkage which ranges from 0 to 0.2 pct when fired to 1600°C. The bricks were reheated 5 hr to 1600°C and met both the reheat specifications of the A.S.T.M. and those of the U. S. Navy for superduty refractories.

The results of the high-temperature load tests indicate that sillimanite brick has excellent load-carrying capacity at high temperatures and, in the alumina-silica class, is probably exceeded only by electrocast corundum-mullite.

Inversion of the sillimanite to mullite proceeds rapidly at 1650°C with a volume expansion of about 6.7 pct and a linear expansion of 1.2 pct. This change may occur at a lower temperature with an increase in time of heating. The temperature of inversion is above that usually encountered in refractory service; therefore, the small expansion at such high temperatures suggests good rigidity and resistance, whereas most refractories show softening and weakening under load.

**Possible Uses:** In the past, there has been no dependable source of sillimanite; therefore, there are

Table I. Screen Analyses of Nonmagnetic Georgia and South Carolina Sillimanite, Pct

Coarser Than Mesh No.	Georgia	South Carolina
28	0.2	0.1
35	11.0	2.3
48	30.5	12.0
65	50.2	32.3
100	71.1	58.1
150	84.7	76.5
200	94.2	89.8

Table II. Chemical Analyses of Nonmagnetic Sillimanite Concentrates, Pct<sup>a</sup>

	Georgia	South Carolina
SiO <sub>2</sub>	39.69	37.59
Al <sub>2</sub> O <sub>3</sub>	57.95	59.75
TiO <sub>2</sub>	0.20	0.24
Fe <sub>2</sub> O <sub>3</sub>	0.99	1.20
Total	98.83	98.78

<sup>a</sup> Analyses by R. H. Stacy, Southern Experiment Station, and P. G. Cotter, Electrotechnical Laboratory, U. S. Bureau of Mines.



no established uses for it. At present, a number of high-grade alumina products are made from several sources, including the crystalline corundum produced in the electric furnace from bauxite, corundum-mullite electrocast refractories made from a mixture of bauxite and fire clay, mullite refractories made from domestic and Indian kyanite, and small amounts made from western domestic andalusite, as well as a little made from South Carolina topaz.

Some of the potential uses for sillimanite are: porcelain for spark plugs, high aluminous refractory bricks (fig. 5), crucibles, saggars of all types, boiler linings, high-temperature cements, linings for indirect-arc and heat-treating furnaces, pyrometer tubes, glass tank blocks, etc.

**Conclusions:** The annual consumption of minerals in the sillimanite group can be estimated only roughly, as many of the producers of these minerals are also consumers and production figures are not available. However, it is believed that the preparation of a high quality concentrate would permit marketing of 15,000 to 20,000 tons of sillimanite per year. Certain outstanding conclusions which have been established to date may be briefly summarized as follows:

1. All of the known domestic sillimanite occurs as disseminated crystals in various types of meta-sediments.

2. Known deposits that appear to offer the best chances for development listed in the order of their importance are: (a) Hart-Elbert-Madison County, Ga., belt; (b) Anderson-Spartanburg-Greenville County, S. C., belt; and (c) Cliffside-Elkin, N. C., belt. Abundance and size of sillimanite crystals in large samples concentrated from each of these areas substantiate this conclusion.

3. The area in southwestern Hart County contains several million tons of sillimanite ore which range from 10 to 12 pct in sillimanite content.

4. Future intensive search for sillimanite, no doubt, will result in the discovery of deposits as good as, and perhaps better than, any discovered to date.



**Fig. 5—Sillimanite ore on left, concentrate in center, and 8-in. brick made from sillimanite concentrate on right.**

(Photo by P. D. Rogers)

5. The sillimanite ore is readily concentrated by froth flotation.

6. Laboratory research upon bricks made from sillimanite indicates that it makes a superior refractory.

7. Possible uses for sillimanite are quite numerous, suggesting that a considerable tonnage could be utilized by various industries.

### References

- <sup>1</sup> Laurence L. Smith: Sillimanite in South Carolina. *Econ. Geol.* (June-July, 1945) **40**, (4).
- <sup>2</sup> A. S. Furcron and Kefton H. Teague: Sillimanite and Massive Kyanite in Georgia (A Preliminary Report). *Ga. Geol. Survey, Bull.* 51, (1945).
- <sup>3</sup> W. C. Hudson: Exploration of Georgia and South Carolina Sillimanite Deposits. *U. S. Bur. Mines. R. I.* 3927 (Sept. 1946).
- <sup>4</sup> Charles E. Hunter and William A. White: Occurrences of Sillimanite in North Carolina. *N. C. Dept. Cons. and Dev. I. C.* 4.
- <sup>5</sup> W. T. McDaniel, Jr.: Personal communication. (March 20, 1947).
- <sup>6</sup> C. Rampacek, B. H. Clemmons, and J. B. Clemmer: Beneficiation of South Carolina Sillimanite Schists. *Jnl. Amer. Ceramic Soc.* (1945) **28**, (8) 197-205.

### Correction

In the March 1950 issue: TP 2815 B. *Radiotracer Studies on the Interaction of Dithiophosphate with Galena* by G. L. Simard, J. Chupak, and D. J. Salley. p. 359, abstract, line 4, reversible should read irreversible. p. 361, table II, line 3, 6 (leached) should read 5 (leached); fig. 4, ordinate, mol/gram should read mol/cm<sup>2</sup>. p. 363, table VII, first subtitle, Galena No. 2 should read Galena No. 2 (unleached), p. 364, third paragraph, line 6, dithiophosphate should read dithiophosphate solely; 4th paragraph, line 17, reference 10 should read reference 11, and reference 7 should read reference 8; References, No. 7, *A. Electrochem.* should read *Z. Electrochem.*

# Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries

by S. A. Falconer

**This paper deals with the practical application of the Dutch State Mines cyclone separator for fine-coal cleaning. The more important operating variables are discussed, and results of a number of continuous-scale tests on various fine coals with a 6-in. cyclone separator are given. Descriptions of the Cyanamid Pilot Plant and the 50 tph Cyclone Separator Plant at the Emma Washery in Holland are included.**

**T**ODAY, as never before, the coal producers of this country are faced with the necessity for obtaining maximum yield of marketable coal of all sizes down to and including the finest. Particular interest is being displayed in a more effective means of recovering the finer sizes which in the past so often have been discarded to waste or treated by inadequate equipment. The reasons for this interest are easily understood. Rising mining costs have resulted in the increased use of mechanized equipment underground for full seam mining. This, in turn, means more refuse in the run-of-mine and

---

S. A. FALCONER, Member AIME, is associated with the Mineral Dressing Laboratory, American Cyanamid Co., Stamford, Conn.

AIME New York Meeting, February 1950.

TP 2884 BF. Discussion (2 copies) may be sent to Transactions AIME before Aug. 31, 1950. Manuscript received Jan. 16, 1950.

---

also more fines in the feed going to the cleaning plant. Stringent antistream pollution laws now make it necessary at least to impound the fines, and usually it is a paying proposition to recover the coal from these fines before sending them to the storage area. Finally, the market and price of the finer sizes has increased, but at the same time specifications have tightened and the demand is for higher grade products.

During the past four years a great deal of publicity has been given to this new process and apparatus for treating fine sizes of ores as well as coal. Excellent discussions of the apparatus itself and the theoretical aspects of the basic operating principles involved in its operation have been presented by

M. G. Driessen,<sup>1-3</sup> M. R. Geer and H. F. Yancey,<sup>4, 5</sup> J. W. Hyer<sup>6</sup> and more recently by D. A. Dahlstrom<sup>8</sup> as well as others.<sup>7, 9-11</sup> In view of this, it has been decided to include in the present paper only a brief description of the cyclone separator and its operating principles. This paper deals primarily with some of the more practical commercial aspects of the process and equipment required for its operation; the variables in the operation of the cyclone, their effect and means for control; and the results obtained with a 6-in. semicommercial size separator on a variety of fine coals, bituminous and anthracite, using finely ground magnetite for the separating medium solids. A description is included of the commercial-size cyclone separator pilot plant installed by the Dutch State Mines at its Emma mine in Holland, where finely ground slate is utilized as medium solids.

**Field of Application:** The Dutch State Mines cyclone separator operates most efficiently on ores and coal in the size range of  $\frac{3}{8}$  in. to about 48 mesh. On some types of feed, effective separation can be made on sizes as small as 100 mesh, but up to the present, this may be considered exceptional.

Depending upon the type of medium used, the specific gravities of separation can be varied within wide limits, up to as high as 2.70, if magnetite is used as medium.

**Early Development:** The development of the Dutch State Mines cyclone as a separator, or cleaner, was the outgrowth of early experimental work by Driessen<sup>1</sup> and others in Holland, just prior to World War II, in the course of which was discovered the effectiveness of the cyclone in reclaiming and thickening the dense medium used in the "Loess" process of



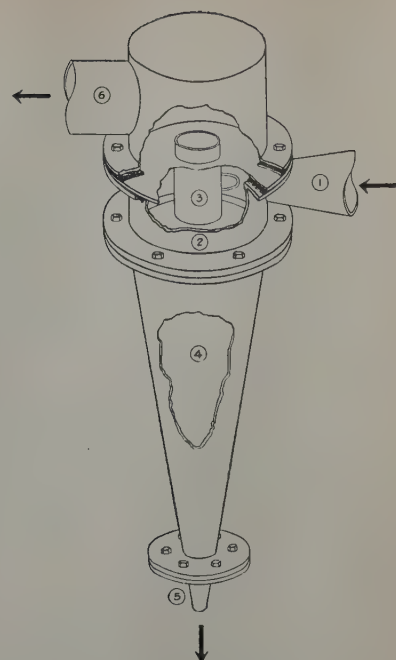
coal cleaning. These early investigations with the cyclone eventually lead to the further discovery that under certain controlled conditions, using correct proportions of finely ground media of appropriate specific gravity, the centrifugal and centripetal forces in a properly designed cyclone are capable of effecting very efficient separations of fine size coal from refuse on the basis of their specific gravity differences.

Of the efficiency of a true sink-and-float method in separating coal from refuse, there can be no question. The industry has been seeking a way to make accurate separation at low cost of all sizes at a predetermined point on the washability curve. Sink-and-float processes, in which gravity is the separating force, employing finely ground suspensions of solids of suitably high specific gravity, approach very closely the properties of a true heavy liquid. Today there are about 80 coal cleaning plants in this country and in Europe using these very efficient processes for treating sizes coarser than about 10 mesh. Unfortunately, these processes are not readily adapted to the treatment of finer sizes, those below about 10 mesh, because of the much longer time required for separation of the particles in a dense medium (or even in a true heavy liquid). Below 10 mesh, the size of installation and cost of coal cleaning plants employing sink-and-float principles to effect separations under static conditions increase in proportion with increasing fineness of the feed. However, if means are supplied to increase the velocity of fall of the heavier fine particles in the separatory fluid, it becomes commercially practical to make separations of these finer sizes on the basis of their specific gravity differences. The Dutch State Mines cyclone separator most effectively furnishes this means and makes possible sharp, efficient separations at particle sizes as small as about 48 mesh. The medium solids used in making the separation may be autogenous (inherent in the material to be separated); or exogenous (extraneous material not inherently present in the feed to be treated).

**Construction and Operating Principles of the Cyclone Separator:** The cyclone separator or "washer" is a simple centrifugal type, utilizing no mechanical means to remove the float-and-sink products. The essential details of construction of a typical separator are shown in fig. 1.

The material to be treated is fed into a suspension consisting of very fine medium solids and water (the specific gravity of which can be considerably lower than the required separating gravity), and the mixture is introduced tangentially under pressure, usually supplied by means of a centrifugal pump, through the feed inlet (1) into the short cylindrical section (2). The latter also carries the central "vortex finder" (3) which guides the upwardly flowing stream of lighter gravity particles or "float" to the discharge outlet (6) and prevents short-circuiting of the incoming feed within the cyclone. Separation is effected in the cone-shaped part (4) by the action of: (a) centrifugal forces which draw the heavier gravity particles to the wall of the cone and eventually discharge them through the bottom or "apex" orifice (5); and (b) centripetal forces which sweep the lighter gravity particles toward the central vortex where they leave the cone through the "vortex finder" (3).

In commenting on certain theoretical considerations involved in the working principles of the cyclone separator, Driessen<sup>2</sup> offers the explanation that:



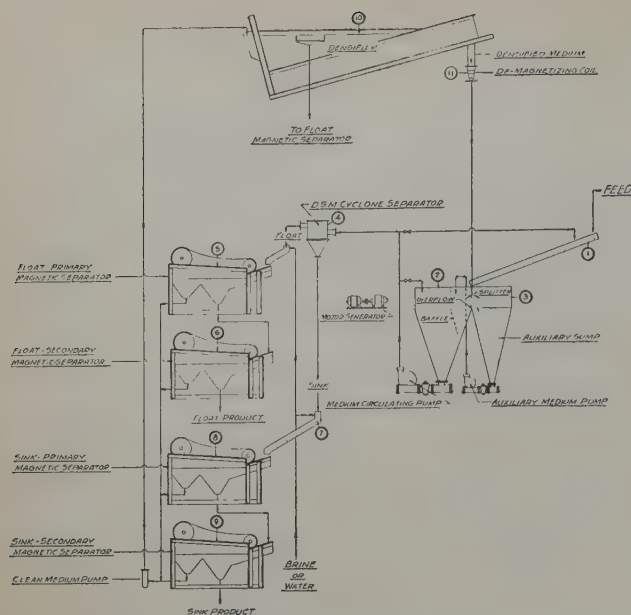
**Fig. 1—Details of construction of a typical cyclone separator.**

In the Cyclone a state of equilibrium appears to exist between the centrifugal force working on the small suspension particles and the flow of the liquid, which acts in the opposite direction. The suspension particles, which are introduced into the Cyclone together with the raw fine coal, are forced towards the cone wall and move down toward the apex. Here a part leaves the Cyclone and another part rises in the central vortex and either leaves the Cyclone at the top or is again forced towards the walls. . . . part of the suspension particles are circulating in the Cyclone, so that it is possible that the concentration, and therefore the specific gravity of the suspension is higher in the Cyclone than the specific gravity at the inlet and outlet . . . (vortex finder). The small suspension particles introduced into the Cyclone, together with the raw coal, form a block for coal particles, whereas the shale particles due to their higher density can readily migrate through the bed.

**Operating Variables:** Although the cyclone separator is fundamentally an extremely simple piece of apparatus, there are a number of variables involved in its operation which influence its capacity and the efficiency of separation. Of these the following have been found to be the most important: (1) diameter of cyclone, and steepness of the conical section; (2) type and fineness of the medium solids used for separation; (3) specific gravity of the medium suspension and solids; (4) size of orifice at apex (sink) discharge; (5) pressure used in introducing feed and medium to the separator; (6) ratio of medium to raw coal feed; (7) tonnage rate and variation in proportion of coal and refuse; (8) diameter of feed inlet; (9) diameter and position of "vortex finder" (outlet for float, or coal); (10) particle size of raw coal feed; and (11) viscosity.

### Testing

**Cyclone Separator Installations in Cyanamid's Pilot Plant:** Following the appointment of American Cyanamid Co. as exclusive technical and sales representatives in the United States for the Dutch State Mines cyclone separator in 1946, apparatus was installed in the pilot plant in Stamford, Conn., for research work and testing of all types of ores and coals on a batch scale or a continuous, semicommercial scale.



**Fig. 2—Dutch State Mines cyclone flowsheet for double cleaning circuit with primary and secondary magnetic separators.**

In this pilot plant, a 6-in. cyclone separator is used for continuous scale testing of ores and coal at feed rates up to about 4 tph of a size range of  $-\frac{1}{4}$  in. x 0. For small-scale batch testing a 3-in. cyclone separator is available. This latter apparatus is capable of handling feeds up to about 10 mesh.

**The 6-in. Cyclone Separator Plant:** In the operation of this 6-in. cyclone separator plant, the material to be tested is either stored in a stockpile just outside the pilot plant or in the small bin inside it. If the feed is known to contain oversize material, it is necessary to remove it before introducing this feed to the cyclone separator. Therefore, the first stage of the operation usually consists of feeding the material either from the stockpile by means of a set of portable conveyors, or from the bin onto a 2x8 ft Tyler Tyrock screen. The oversize from this screen, after sampling, is removed to waste or to storage in case some separate form of treatment is desired.

The undersize of this screen is sluiced by means of medium supplied through a by-pass pipeline into a 4-in. Wemco pump sump. The medium used for this purpose is the overflow from the 4-in. pump sump which flows into the hopper of a 2-in. Wemco pump used to control the level in the 4-in. pump sump. In addition to the overflow from the 4-in. pump sump, the 2-in. pump sump also receives the densified return medium plus the water that is added to dilute the densified medium to the required specific gravity for the cyclone separator feed.

The 4-in. Wemco pump referred to above delivers the  $-\frac{1}{4}$ -in. feed plus the required volume of medium to the cyclone separator. This pump line is equipped with a by-pass so that the required volume of feed and the correct pressure may be maintained in the cyclone separator.

The 6-in. cyclone separator is installed a short distance above a longitudinally divided 2x8-ft Allis-Chalmers Lohead vibrating screen equipped with 8 or 10-mesh woven-wire cloth. The apex discharge (sink) and the vortex discharge (float) are discharged separately onto the divided screen. The oversize products travel along the screen, through

which most of the medium and undersize float or sink is removed by straight drainage. The remaining adhering medium and fine float and fine sink are removed by means of water sprays located near the discharge end of the screens.

A portion of the medium and the separated products which are removed by drainage through the screen is returned to the sumps shown in fig. 2.

The undersize from the section of the screen handling the apex discharge, or sink, together with most of the medium that has drained through the sink side of the screen is delivered by a 2-in. Wemco pump to the first of two 12-in. Dings-Crockett S-F type magnetic separators, which are operated in series. Practically all of the magnetic medium contained in the pulp is recovered in the first magnetic separator; the second magnetic separator acts principally as a scavenger to remove the last traces of medium from the sink product. While tailing from the second magnetic separator is being pumped to waste, timed samples are cut accurately for analyses and tonnage estimating purposes.

The float washings and the adhering portion of the medium draining through the screen handling the float product are treated in a manner similar to that described above for the sink drainage and washings.

The combined magnetic material recovered from the float and the sink magnetic separators is delivered to the sump of a 2-in. Wilfey pump which delivers the medium to a 24-in. Akins Densifier. Sometimes, instead of the densifier, a 6-ft thickener and a 2-in. Dorco type VM diaphragm pump are used. The returned medium, whether from the densifier or the thickener, passes through a 3-in. ac demagnetizing coil before returning to the cyclone feed circuit.

The above-mentioned equipment and auxiliary apparatus are installed in such a manner as to provide maximum flexibility. The cyclone separator is built so that its various parts may be interchanged readily to provide larger or smaller openings of feed, apex, and vortex; or to provide for greater or less slope of the conical section of the apparatus. The above-mentioned description applies both to the flowsheet used in the pilot plant, when screens are employed ahead of the cyclone separator, and to treat the separated products. However, on some types of materials, it may not be necessary to feed the screens ahead of the cyclone separator and there may not be a sufficient amount of  $+10$ -mesh sizes in the feed to warrant passing the separated products over drainage and washing screens. In such cases, the flowsheet shown in fig. 2 is employed. Referring to fig. 2, it will be noted that the feed, together with medium, is sluiced down the launder (1) into a feed sump (2) from which it is fed by the 4-in. pump to the cyclone separator (4). The medium and feed in the sump (2) are maintained at a constant level by means of the sump (3) and the 2-in. auxiliary pump, as shown.

It will be noted that the vortex discharge of the cyclone separator flows directly by gravity to the magnetic separator (5) where most of the medium is recovered for re-use. The tailings from the magnetic separator (5) flow by gravity to the secondary magnetic separator (6). The nonmagnetics of the latter form the finished float product, while the small additional amount of medium that may be recovered in this secondary separator joins the magnetic medium from the primary separator (5). The apex discharge from the cyclone separator (4) flows down a launder (7) to the primary magnetic separator (5).



rator (8). Normally, water is used to dilute this thick apex discharge in order to assist its flow and the separation in the magnetic separator (8). However, in the treatment of potash ores, saturated brine would be used instead of ordinary water at this point. As in the case of the float recovery circuit, the tailing from magnetic separator (8) flows by gravity to the secondary magnetic separator (9) where the last traces of magnetic medium are removed from the finished sink product. The magnetic material recovered by this secondary separator joins magnetic material recovered from the primary magnetic separator (8).

All of the recovered medium from magnetic separators 5, 6, 8, and 9 are joined and pumped to a densifier (10). The overflow from the latter is substantially free of magnetic solids but to avoid any possible loss at this point, this overflow is directed back to the primary magnetic separator (5). The thickened solids from densifier (10) are discharged through a tube surrounded by an ac demagnetizing coil and delivered back to pump hopper (3) or (2). This densified material is, of course, diluted with whatever amount of water or brine is needed to bring the medium solids to the required density for feeding into the cyclone separator.

*The 3-in. Cyclone Separator for Batch Testing:* The 3-in. cyclone separator and auxiliary apparatus for conducting batch tests are shown in fig. 3. A hopper is provided to hold the sample of the material to be tested together with a desired proportion of fluid medium. This conical hopper is connected to the 2-in. Wemco pump which is used to introduce the feed into the cyclone separator. A by-pass on the pump line permits recirculation of the feed and medium directly back to the hopper when desired.

In operation, the feed and medium, after suitable mixing, are introduced into the cyclone separator at whatever pressure desired, the pressure being regulated by the by-pass line. The vortex overflow and the apex discharge from the 3-in. cyclone separator flow back into the hopper. Timed samples of both of these discharges are caught during the running of the test.

With this setup, it is possible to conduct a great number of tests on any given material with relative

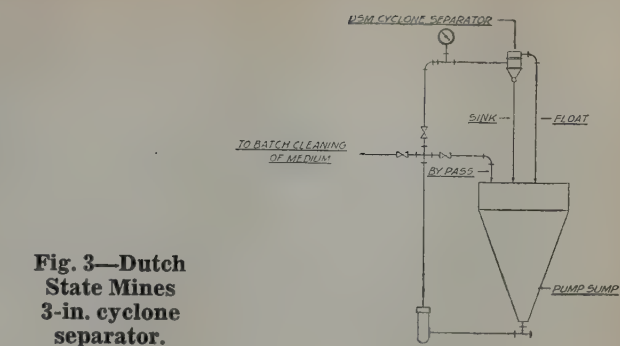


Fig. 3—Dutch State Mines 3-in. cyclone separator.

ease and rapidity. For practical reasons, however, only a few tests can be made with any single charge of a material since it is undesirable to recirculate the separated products for any extended time because repeated passage through the pump degrades the feed.

As previously mentioned, the size of this cyclone separator precludes the treatment of feeds coarser than about 10 mesh. However, this small apparatus does perform a very useful function in that it permits making a large number of tests on a relatively small amount of material. The results of these batch tests provide a reliable guide as to what separations might be expected with larger commercial-size cyclone separators. If results with the 3-in. cyclone are favorable, further testing on a continuous scale with the 6-in. cyclone separator can be justified. When desired, the 6-in. cyclone separator also can be used for batch testing by using a flow scheme similar to that employed with the 3-in. cyclone.

**Results of Cyclone Separator Testing of Samples of Bituminous and Anthracite Coals:** The following are some typical examples of results which can be expected with the Dutch State Mines cyclone separator on bituminous and anthracite fines.

It will be observed that the separations obtained in the various examples reported were made under differing sets of conditions. In each case, the conditions reported are those which were found to be best suited for the particular type of material tested.

Example No. 1

Bituminous coal fines from current mining, Pittsburgh seam.  
Size range tested, ¼ in. x 48 mesh.  
Analysis of raw coal feed.

Size	Pct Weight	Pct Ash	Pct Sulphur
¼ in. x 4 mesh	3.5	11.14	1.45
4x14 mesh	63.4	14.66	1.51
14x28 mesh	18.3	12.90	1.49
28x48 mesh	11.2	14.38	1.74
48x100 mesh	1.9	17.86	2.03
100x200 mesh	1.0	27.20	4.54
—200 mesh	0.7	36.05	2.60

Washability Studies, Sink-and-Float Tests, Raw Coal

Size Range	Float 1.30			Sink 1.30 Float 1.35			Sink 1.35 Float 1.40			Sink 1.40 Float 1.45		
	Pct Wt.	Pct Ash	Pct S	Pct Wt.	Pct Ash	Pct S	Pct Wt.	Pct Ash	Pct S	Pct Wt.	Pct Ash	Pct S
¼ in. x 14 mesh	64.5	4.91	0.94	15.6	7.50	1.60	4.6	11.57	2.86	2.7	16.33	3.78
14x48 mesh	59.2	3.02	0.97	21.5	7.17	1.44	3.8	11.82	2.48	2.6	15.73	3.09
48 mesh x 0	20.7	4.00	0.98	27.9	8.66	1.26	13.7	14.16	1.62	5.2	17.11	2.20
Total ¼ in. x 0	61.4	4.35	0.95	17.8	7.01	1.52	4.7	11.9	2.63	2.7	16.3	3.50

Size Range	Sink 1.45 Float 1.55			Sink 1.55		
	Pct Wt.	Pct Ash	Pct S	Pct Wt.	Pct Ash	Pct S
¼ in. x 14 mesh	1.5	23.55	4.27	11.1	73.87	2.40
14x48 mesh	1.8	21.93	3.85	11.1	67.79	3.67
48 mesh x 0	2.1	19.35	2.95	30.4	53.19	6.22
Total ¼ in. x 0	1.6	22.8	4.05	11.8	70.2	3.02

Cumulative Float on 1.55 sp gr, 88.2 pct weight at 6.00 pct ash, 1.27 pct sulphur.  
Sink in 1.55 sp gr, 11.8 pct weight at 70.2 pct ash, 3.02 pct sulphur.

Operating details of test at 1.55 sp gr separation.

6 in. Cyclone, 15 in. cylindrical section, 3¼ in. conical section.

Pressure, 20 psi.

Orifice feed inlet, 3 in. reduced to 1 in.

Orifice vortex (top outlet), 2 in.

Orifice apex (bottom outlet), 15/16 in.

Feed rate, 1½ tph.

Medium used, "B" Magnetite (—100-mesh, 75 pct—325-mesh).

Specific gravities, medium, feed, 1.185; overflow, 1.14; underflow, 2.15.

Results, Clean Coal, 88.92 pct weight recovery, 5.81 pct ash, 1.34 pct sulphur.

Size Range	Float 1.50			Sink 1.50 Float 1.55			Sink 1.55		
	Pct Wt.	Pct Ash	Pct S	Pct Wt.	Pct Ash	Pct S	Pct Wt.	Pct Ash	Pct S
+10 mesh, 66.1 pct	99.37	5.10	1.27	0.36	25.91	3.61	0.27	33.45	3.10
—10 mesh, 33.9 pct	95.50	5.34	1.40	1.73	23.82	3.49	2.77	52.05	3.60
Total ¼ in. x 0	98.05	5.16	1.30	0.83	24.10	3.50	1.12	49.10	3.58

Refuse, 11.08 pct weight recovery, 69.84 pct ash, 3.82 pct sulphur.

+10 mesh, 52.6 pct	2.69	12.88	2.63	3.24	27.03	3.88	94.07	75.22	2.55
—10 mesh, 47.4 pct	2.67	12.50	2.52	2.30	20.12	3.00	95.03	71.37	5.29
Total ¼ in. x 0	2.68	12.70	2.58	2.79	24.40	3.54	94.53	72.80	3.85

#### Efficiency of Separation

	Pct Weight Yield Coal at 1.55 Sp Gr	Pct Ash	Pct S	Pct 1.55 Sp Gr Sink in Coal	Pct 1.55 Sp Gr Float in Refuse
Theoretical	88.2	6.00	1.27	NIL.	NIL.
Cyclone separation	88.92	5.81	1.34	1.12	5.47

#### Example No. 2

Anthracite culm bank fines.

Size range of feed, 5/16 in. x 0.

Analysis of raw coal feed.

Size	Pct Weight		Pct Ash	
	Ind.	Cum.	Ind.	Cum.
+5/16 in.	0.4	0.4	54.15	54.15
—5/16x3/16 in.	1.5	1.9	57.60	56.87
—3/16x3/32 in.	5.1	7.0	40.30	44.80
—3/32x1/16 in.	8.4	15.4	34.15	38.99
—1/16x3/64 in.	13.7	29.1	33.30	36.31
—3/64x1/32 in.	18.6	47.7	30.80	34.16
—1/32 in. x 35 mesh	17.4	65.1	35.35	34.48
—35x48 mesh	13.5	78.6	35.65	34.68
—48x65 mesh	8.8	87.4	44.45	35.66
—65x100 mesh	5.1	92.5	40.47	35.93
—100 mesh	7.5	100.0	46.45	36.72

#### Operating Conditions

6 in. cyclone 60° cone.

Pressure, 20 psi.

Feed inlet orifice, 1½ in.

Orifice vortex (top outlet), 1½ in.

Orifice apex (bottom outlet), 1.2 in.

Specific gravity feed, 1.57; overflow, 1.29; underflow, 2.21.

Size Range	Screen Oversize Coal				Screen Undersize Coal			
	Pct Weight		Pct Ash		Pct Weight		Pct Ash	
	Ind.	Cum.	Ind.	Cum.	Ind.	Cum.	Ind.	Cum.
Clean Coal								
+5/16 in.	1.7	1.7	15.05	15.05				
—5/16x3/16 in.	8.4	10.1	12.10	12.60				
—3/16x3/32 in.	39.0	49.1	9.10	9.82				
—3/32x1/16 in.	39.8	88.9	9.05	9.47	2.9	2.9	11.32	11.32
—1/16x3/64 in.	8.4	97.3	9.70	9.49	11.9	14.8	11.25	11.26
—3/64x1/32 in.	1.9	99.2	10.70	9.52	19.5	34.3	10.83	11.02
—1/32 in. x 35 mesh	0.4	99.6	11.35	9.52	20.1	54.4	12.16	11.44
—35x48 mesh	0.2	99.8	16.30	9.54	16.5	70.9	13.84	12.00
—48x65 mesh	0.2	100.0	9.40	9.54	11.6	82.5	15.64	12.51
—65x100 mesh					7.2	89.7	19.10	13.04
—100 mesh					10.3	100.0	34.69	15.27



Refuse								
+ 5/16 in.	1.9	1.9	70.94	70.94				
- 5/16x3/16 in.	9.7	11.6	75.13	74.44				
- 3/16x3/32 in.	37.5	49.1	66.15	68.11				
- 3/32x1/16 in.	36.7	85.8	62.86	65.86	5.1	5.1	61.8	61.8
- 1/16x3/64 in.	10.0	95.8	65.62	65.84	17.6	22.7	60.6	60.87
- 3/64x1/32 in.	3.2	99.0	66.12	65.85	25.3	48.0	59.85	60.33
- 1/32 in. x 35 mesh	0.6	99.6	71.18	65.88	21.0	69.0	68.00	62.67
- 35x48 mesh	0.2	99.8	64.64	65.88	13.7	82.7	67.50	63.47
- 48x65 mesh	0.2	100.0	59.02	65.86	7.8	90.5	68.90	63.93
- 65x100 mesh					4.6	95.1	68.45	64.15
- 100 mesh					4.9	100.0	69.60	64.42

Example No. 3

Anthracite silt bank.  
Size of feed, —3/16 in. x 0.  
Analysis raw coal feed.

Screen Analysis, Raw Coal Feed

Mesh	Pct Wt.	Pct Wt. Cumulative	Pct Ash	Pct Ash Cumulative
—3/16 in + 20	47.41	47.41	37.14	
—20 + 35	14.28	61.69	34.76	
—35 + 48	9.16	70.85		
—48 + 65	8.46	79.31	34.33	
—65 + 100	5.68	84.99	38.82	35.80
—100 + 150	3.24	88.23	34.80	
—150 + 200	1.88	90.11	39.15	
—200	9.89	9.89	64.25	57.24
Total	100.00	100.00	38.69	38.69

Operating Details and Results of Separation

Type of cone, 20°.  
Pressure, 20 psi.  
Orifice feed inlet, 1½ in.  
Orifice vortex (top outlet), 1½ in.  
Orifice apex (bottom outlet), 1.2 in.  
Feed rate, 2½ tph.  
Medium used, Magnetite "B".  
Specific gravities: feed, 1.69; vortex, 1.44; apex, 2.24.

Products	Pct Wt. of Product	Pct Weight of Raw Feed	Pet Ash	
			Ind.	Cum.
Raw coal feed		100.0	38.94	
Clean coal (full size range)		68.28	20.81	
Refuse		31.72	77.97	

Clean Coal				
(screen)				
+ 20 mesh (o'size)		33.72	12.98	12.98
—20 mesh x 0	100.0	34.56	28.44	20.81
—20x35 mesh	19.73	6.82	12.27	12.86
—35x48 mesh	15.47	5.35	13.60	12.95
—48x65 mesh	15.86	5.48	15.80	13.25
—65x100 mesh	11.80	4.08	19.73	13.73
—100x150 mesh	7.54	2.61	24.64	14.22
—150x200 mesh	4.64	1.60	33.23	14.73
—200 mesh	24.96	8.62	62.88	20.81

Clean coal with —200-mesh fines removed, 59.66 pct weight recovery in product analyzing 14.73 pct ash. With the —100 mesh sizes removed, the weight recovery of coal would be reduced to 55.45 pct but the ash content would be only 13.73 pct.

Refuse				
(screen)				
+ 20 mesh (o'size)		13.69	78.18	78.18
—20 mesh x 0	100.0	18.03	77.82	77.97
—20x35 mesh	41.36	7.46	77.97	78.10
—35x48 mesh	21.13	3.81	78.75	78.20
—48x65 mesh	16.54	2.98	78.73	78.26
—65x100 mesh	8.88	1.60	78.12	78.25
—100x150 mesh	3.52	0.63	76.29	78.21
—150x200 mesh	1.53	0.28	73.30	78.17
—200 mesh	7.04	1.27	73.33	77.97

Efficiency of Separation Based on Sink-Float Tests on Recovered Coal Sizes

Size Fraction	Pct Weight Recovery		Pct Ash Clean Coal		Sp Gr Separation	Pct Recovery on Total Feed Coal	
	Theory	Actual	Theory	Actual		Theory	Actual
+ 20 mesh	62.39	61.89	12.20	12.24	1.88		13.48
—20x48	67.27	65.72	11.26	12.02	1.94		29.23
—48x65	72.97	70.11	14.52	16.40	2.06		6.92
—65x100	73.76	72.79	21.37	25.42	2.09		7.58
—100	?	82.34	?	53.45	?		
Total + 100	57.57	57.21	12.45	14.38	1.95 <sup>a</sup>	57.57	57.21
Calc. feed				38.69			

<sup>a</sup> Weighted average.

### Example No. 4

Anthracite silt bank material.  
Size range of feed, —3/32 in. x 100 mesh.

#### Analysis Raw Coal Feed

Mesh	Pct Weight Ind.	Pct Weight Cum.	Pct Ash Ind.	Pct Ash Cum.
+3/32 in.	4.22	4.22	22.8	22.8
—3/32x3/64 in.	46.55	50.77	25.7	25.5
—3/64 in.x28 mesh	12.21	62.98	28.6	26.1
—28x35 mesh	27.09	90.07	35.1	28.8
—35x48 mesh	4.77	94.84	35.3	29.1
—48x60 mesh	1.24	96.08	34.4	29.2
—60x100 mesh	1.76	97.84	37.3	29.3
—100x150 mesh	0.50	98.34	38.4	29.4
—150x200 mesh	0.34	98.68	40.0	29.4
—200 mesh	1.32	100.00	55.8	29.8

#### Sink and Float Washability Tests on Raw Coal Feed

Specific Gravity		Float				Sink	
		Pct Weight		Pct Ash			
Sink	Float	Ind.	Cum.	Ind.	Cum.	Pct Wt. Cum.	Pct Ash Cum.
	1.60	42.12	42.12	5.2	5.2		
1.60	1.70	21.40	63.52	12.0	7.5	57.88	47.5
1.70	1.80	6.20	69.72	23.0	8.9	36.48	68.3
1.80	1.90	2.72	72.44	33.5	9.8	30.28	77.6
1.90	2.00	2.20	74.64	42.5	10.8	27.56	82.0
2.00		25.36		85.4	29.8	25.36	85.4

#### Cyclone Separator Tests, Operating Details

Separator, 6 in. cone 60°.  
Orifice feed inlet, 1½ in.  
Orifice vortex (top outlet), 1½ in.  
Orifice apex (bottom outlet), 1.2 in.  
Pressure, 20 psi.  
Feed rate, 1½ tph.  
Medium used, Magnetite.

#### Pct Yield and Ash of Products

Test No.	Sp Gr Medium Fed to Separator	Pct Weight Recoveries						Pct Ash in Products					
		Coal			Refuse			Coal			Refuse		
		+10 Mesh	—10 Mesh	Total	+10 Mesh	—10 Mesh	Total	+10 Mesh	—10 Mesh	Total	+10 Mesh	—10 Mesh	Total
1	1.54	23.61	49.96	73.57	7.47	18.96	26.43	8.5	10.3	9.7	85.1	80.7	81.94
2	1.58	23.21	52.93	76.14	8.25	15.61	23.86	9.2	12.6	11.6	85.3	83.8	84.32
3	1.63	32.31	47.62	79.93	4.11	15.96	20.07	9.7	13.8	12.1	80.5	85.6	84.56

#### Pct Coal and Refuse in Final Products

Test No.	Approx. Separating Gravity	Coal			Pet Coal in Refuse			Pet Coal Lighter Than 1.70 Sp Gr
		Pet Refuse in Coal						
		+ 10 Mesh	—10 Mesh	Total	+ 10 Mesh	—10 Mesh	Total	
1	1.90	3.36	6.60	5.56	1.72	6.56	5.19	4.5
2	2.0	2.96	8.16	6.57	2.24	5.80	4.57	3.4
3	2.05?	1.52	6.76	4.64	10.80	6.40	7.30	4.8

### Example No. 5

Anthracite fines from North Africa.  
Separator 6 in., cone 60°. Size of feed, 5x0.5 mm (approx. 4x30 mesh).  
Pressure, 19 psi.  
Orifice feed inlet, 1½ in.  
Orifice vortex (top outlet), 1½ in.  
Orifice apex (bottom outlet) diameters variable.  
Medium, Magnetite, —100 mesh, 98 pct —325 mesh.  
Specific gravity of pulp medium feed, variable.

Test No.	Pct Weight			Pct Ash			Specific Gravities			Apex Diameter, In.
	Raw Coal	Clean Coal	Refuse	Raw Coal	Clean Coal	Refuse	Feed	Vortex	Apex	
1	100.0	12.8	87.2	27.38	3.19	30.93	1.46	1.15	2.20	0.90
2	100.0	19.5	80.5	29.24	2.63	35.69	1.46			0.85
3	100.0	65.6	34.4	27.51	3.14	73.98	1.46	1.19	2.25	0.80
4	100.0	73.4	26.6	31.28	10.94	87.39	1.66	1.41	2.62	0.80
5	100.0	72.6	27.4	29.73	8.92	84.86	1.66			0.85
6	100.0	72.0	28.0	29.76	8.43	84.61	1.66	1.34	2.40	0.90



# Screen Analysis of Raw Coal

Mesh	Pct Weight	Cum. Pct Weight
+4	9.6	9.6
—4x6	20.7	30.3
—6x12	58.0	88.3
—12x16	4.0	92.3
—16x30	2.7	95.0
—30x50	1.2	96.2
—50x0	3.8	100.0

## Example No. 6

Anthracite silt bank fines.  
Size range of feed, 3/16 in. x 0.  
Screen analysis, raw coal feed.

Size	Pct Wt.	Pct Ash
—3/16x3/32 in.	3.0	23.0
—3/32x3/64 in.	36.0	21.0
—3/64 in. x 50 mesh	52.0	22.6
—50 mesh	9.0	24.2
Total	100.0	22.2

Separator 6 in., cone 20°. Pressure, 25 psi.  
Orifice feed inlet, 2 in.  
Orifice vortex (top outlet), 2 in.  
Orifice apex (bottom outlet), 7/8 in.  
Feed rate, 4/10 tph, Tests 1-3; 3-7/10 tph, Test 4.  
Medium used, Magnetite.

Test No.	Feed Pct Ash	Clean Coal		Refuse		Sp Gr Medium Solids in Feed
		Pct Wt.	Pct Ash	Pct Wt.	Pct Ash	
1	20.03	86.19	11.60	13.81	72.62	1.35
2	21.95	80.10	10.54	19.90	67.91	1.30
3	21.14	83.75	11.20	16.25	72.39	1.42
4	22.94	79.15	11.07	20.85	67.97	1.40

# Sink-and-Float Data on Separated Products

Pct Weight in Products	Size Fraction	Test 1		Test 2		Test 3		Test 4	
		Coal	Refuse	Coal	Refuse	Coal	Refuse	Coal	Refuse
Float on 1.70	+ 10	87.65	5.78	93.34	7.50	89.48	2.46	87.27	13.54
	—10	87.54	9.38	89.72	10.69	88.76	7.41	87.54	19.95
Sink 1.70	+ 10	2.84	1.99	0.99	1.86	1.56	0.64	3.71	1.08
Float on 1.75	—10	0.69	0.63	1.58	1.45	0.80	0.82	0.69	2.48
Sink 1.75	+ 10	2.06	0.63	2.13	3.27	2.00	0.69	2.00	2.35
Float on 1.80	—10	2.07	0.99	1.63	1.43	1.93	0.78	2.07	0.88
Sink 1.80	+ 10	2.25	1.96	1.38	3.68	2.04	2.01	2.41	2.22
Float on 1.85	—10	2.37	1.86	2.16	2.69	2.16	2.31	2.37	2.23
	+ 10	5.20	89.64	2.16	83.69	4.92	94.20	4.61	80.81
Sink in 1.85	—10	7.38	87.14	4.91	83.74	6.35	88.68	7.38	74.96

## Example No. 7

Anthracite fines, current fresh mined coal.  
Size of feed, —1/4 in. x 48 mesh.  
Analysis, raw coal feed.

Size	Pct Weight	Pct Weight Cum.	Pct Ash Ind.	Pct Ash Cum.
—1/4 in. x 10 mesh	51.15	51.15	18.62	18.62
—10x28 mesh	28.72	79.87	23.23	20.28
—28x48 mesh	12.85	92.72	28.24	21.38
—48 mesh	7.28	7.28	33.31	22.25
Total		100.00		22.25

Type of cone, 60° angle.  
Pressure, 20 psi.  
Orifice feed inlet, 1 1/8 in.  
Orifice vortex (top outlet), 1 1/8 in.  
Orifice apex (bottom outlet), 0.765 in.  
Feed rate, 1 1/2 tph.  
Medium used, Magnetite, 80 pct —325 mesh.  
Specific gravities: feed, 1.45; vortex, 1.32; apex, 2.22.

# Analyses of Separated Products

Size	Clean Coal		Pct Ash	Refuse		Pct Ash
	Pct Weights			Pct Weights		
	of Raw Feed	of Product		of Raw Feed	of Product	
—¼ in. x 10 mesh	44.80	53.39	10.71	6.35	39.51	74.44
—10x28 mesh	23.19	27.63	11.02	5.53	34.37	74.45
—¼ in. x 28 mesh	67.99	81.02	10.82	11.88	73.88	74.44
—28x48 mesh	9.99	11.90	14.23	2.86	17.79	77.16
—¼ in. x 48 mesh	77.98	92.92	11.25	14.74	91.67	74.91
—48 mesh	5.94	7.08	25.15	1.34	8.33	69.49
Total	83.92	100.00	12.24	16.08	100.00	74.51

## Example No. 8

Anthracite fines.

On this same feed as example 7, the effect of substituting a cone having an angle of 20° instead of 60°, with all other operating variables the same, resulted in a decreased recovery of coal, but a higher grade product, as shown by the following figures:

Size	Clean Coal		Pct Ash	Refuse		Pct Ash
	Pct Weights			Pct Weights		
	of Raw Feed	of Product		of Raw Feed	of Product	
—¼ in. x 10 mesh	36.38	52.13	7.20	13.61	45.06	50.07
—10x28 mesh	21.11	30.25	7.91	9.55	31.62	53.30
—¼ in. x 28 mesh	57.49	82.38	7.46	23.16	76.68	51.40
—28x48 mesh	8.46	12.12	9.41	5.12	16.95	62.07
—¼ in. x 48 mesh	65.95	94.50	7.71	28.28	93.63	53.37
—48 mesh	3.84	5.50	17.69	1.93	6.37	66.44
Total	69.79	100.00	8.26	30.21	100.00	54.17

**Dutch State Mines Fine Coal Cyclone Separator Plant Using Slate Medium:** For the past two years, the Dutch State Mines have been operating a commercial size plant for recovery of bituminous fines, using 350 mm (13¾ in.) diam cyclone separators.

This plant, which is installed in the Emma Washery, has a capacity of 50 tph of run-of-mine coal of a size range of 8x0.5 mm (0.315 in. x 30 mesh).

The flowsheet employed in this plant is shown in fig. 4.

The fine screen undersize from the main plant is conveyed by elevator (1) to a 5x12 ft Allis-Chalmers Lohead presizing screen (2) where the +8 mm (approximately 5/16 in.) sizes are removed and returned to the main washing plant.

The —8 mm sizes flow by gravity to a mixing sump (3) and are pumped from there to a 5x12 ft Allis-Chalmers Lohead screen (4) equipped with 0.5 mm (approximately 30 mesh) openings. The undersize, or —30-mesh sizes from this screen, is used to furnish the autogenous (slate slime) medium used in the operation of this cyclone plant. In the first step in this utilization, the fines are classified roughly by means of a dam placed across the width of the sloping hopper underneath the above-mentioned screen. The underflow of the dam, that is the coarser portion of the —30-mesh sizes, flows to a 13¾ in. cyclone thickener (5). The apex discharge (heavy solids) of the thickener is sent to the main plant for use as medium in the Baum washers. The vortex discharge from (5) is returned to the desliming screen (4). The overflow from the dam, placed in the hopper under this screen, flows to a slate-slime (medium) surge tank (6). Excess overflow from the latter flows to waste. The underflow is pumped to two 13¾ in. cyclone thickeners, (7).

The vortex discharges from these thickeners flow back to the surge tank (6), while the apex discharge (thickened medium) joins the washings from the refuse screen (10) and eventually finds its way to (12), one of the two flotation machines in the medium regeneration circuit. Here the coal is removed to give a coal-free slate medium, the further disposition of which will be described later.

Returning to the oversize, that is the —8x0.5 mm (—5/16 in. x 30 mesh) of screen (4), this product is collected in a constant level feed box equipped with equally spaced outlets for supplying this feed (together with fine slate medium in correct proportion) to four 13¾ in. cyclone separators (8). The static head is sufficient to introduce the feed and medium solids at the equivalent of 19.1 psi to the separators. Here the coal and refuse solids are separated at a gravity equivalent to 1.45. The vortex discharge, carrying the separated coal (75 pct weight of the cyclone separator feed coal) flows to a 5x12 ft Allis-Chalmers Lohead screen (9) equipped with 30-mesh screen. Oversize of the screen goes to finished product storage. The screen undersize (medium plus any degraded coal that may have formed during the preceding operations) flows by gravity to the feed end of the 5x12 ft Allis-Chalmers Lohead refuse screen (10) where it is used to dilute the dense apex discharge of the cyclone separators (8) before passing over the screen and thus assists in removing the medium solids from the —8 mm x 30 mesh refuse particles.

The undersize from the coal screen (9) and the refuse screen (10) flows to the regeneration flotation machine (12), previously mentioned in the description of this flowsheet, where any —30-mesh coal fines are removed.





rator. Alternatively, if these fine sizes are not removed before the feed is introduced into the cyclone separator, it may be necessary to remove them later from the recovered coal in order to obtain the desired grade.

In the range of  $\frac{3}{8}$  in. x 48 mesh the separations on the various size fractions are uniformly good. Coal of excellent grade is produced and recoveries closely approach theoretical as developed from washability studies with heavy liquids.

These pilot plant tests have shown that magnetite medium is ideally suited for separating bituminous as well as anthracite fines, from the standpoint of control of the operation, simplicity of flowsheet, and ease and efficiency of recovery of the medium for re-use.

The separating gravity has been found to be conveniently regulated and controlled by: (a) the specific gravity of the medium, (b) diameter of the apex discharge (refuse) opening, and (c) pressure at which the feed enters the cyclone separator.

When separating at low gravities, variations in the specific gravity of the medium entering the cyclone cause a correspondingly greater change in the separating gravity, but as the separating gravity increases, variations in the separating gravity of the entering medium have less effect. Although not essential, automatic control of the feed gravity may be advisable.

The gravity of separation is sufficiently sensitive to changes in the diameter of the discharge opening so that an automatically regulated discharge valve often can be justified in the interests of close control of the size of the discharge opening as well as reduction of wear at this point. In this connection, the Dutch State Mines engineers have developed a special type of rubber valve for closely regulating the size of the opening of the apex discharge on the cyclones operating at the Emma mine washery in Holland.

On the basis of their experience in operating cyclone separators with  $\frac{3}{8}$ -25-mesh slate medium, the Dutch State Mines engineers recommend that the specific gravities of both the separatory medium and the separation effected by the cyclone should not differ greatly for maximum efficiency and ease of operation, because of the lesser sensitivity of the medium to fluctuations in feed gravity. On the other hand, experience with a somewhat coarser magnetite medium in the Cyanamid pilot plant indicates that maximum efficiency may be obtained when there is a definite difference between the gravities of the feed and separation, and that in some cases efficiency is decreased by any change in this gravity differential. It is further concluded that the volumetric distribution of the float-and-sink products affect the choice of the feed gravity and the diameter of the discharge opening in the selection of any particular gravity of separation. Where a feed with a full range of sizes is being treated, the matter of medium recovery requires careful consideration. Experience in the Cyanamid pilot plant indicates that it is difficult to drain and wash magnetite medium solids and the separated products from the cyclone separator on a screen much finer than about 10 mesh. If the feed contains only a minor amount of sizes coarser than 10 mesh, it may be desirable to omit any screening operation, and send the separated products directly to magnetic separators, as shown in fig. 2. Under such conditions, a sufficient number of magnetic separators, of proper size, must be pro-

vided to recover and clean continuously the magnetite. The amount to be cleaned will be about three times the weight of the feed solids. That is, if the feed to the cyclone separator amounts to 10 tph, 30 tons of magnetite must be recovered, cleaned, and recycled back to the cyclone separator.

When operating with cyclone separators of metal construction and using ferrosilicon or magnetite media, experience in the Cyanamid pilot plant indicates that considerable wear occurs toward the lower end, particularly at the apex.

In the commercial plant operated by the Dutch State Mines, an automatic, rubber valve is used at the apex discharge. This valve has a life of about three to four weeks. The other parts of the cyclones operating at this plant are sectionalized and are constructed of cheap, easily replaceable cast iron. The section nearest the apex discharge has been found to last about one month. The other parts have a life up to seven months, depending on the diameter of the section.

Although the Dutch State Mines engineers have developed a rubber liner for installing inside the cyclone separators, their present feeling is that for their own particular operations cheap cast-iron cast parts probably will be cheaper than a rubber-lined cyclone which would have a higher initial cost.

In this country, a well-known manufacturer of concentrating equipment has developed 3-in., 6-in., and 12-in. diam commercial type cyclone separators. The two smaller size cyclones have either rubber-lined conical sections or are constructed of hardened Meehanite. For the 12-in. size, both the feed and conical sections are rubber-lined.

## References

- <sup>1</sup> M. G. Driessen: Cleaning Coal by Heavy Liquids. *Jnl. Inst. of Fuel* (Aug. 1939) **12**, 327-341.
- <sup>2</sup> M. G. Driessen: The Use of Centrifugal Force for Cleaning Fine Coal in Heavy Liquids and Suspensions with Special Reference to the Cyclone Washer. *Jnl. Inst. of Fuel* (1945) **19**, (12) 33-45.
- <sup>3</sup> M. G. Driessen: The Use of Hydraulic Cyclones in Thickeners and Washers in Modern Coal Preparation. *Trans. AIME* (1948) **177**, 240-260; *Coal Tech.* (Aug. 1947) TP 2135.
- <sup>4</sup> W. R. Geer and H. F. Yancey: Preliminary American Tests of a Cyclone Coal Washer Developed in the Netherlands. *Trans. AIME* (1948) **177**, 220-235; *Coal Tech.* (Feb. 1947) TP 2136.
- <sup>5</sup> H. F. Yancey and M. R. Geer: The Cyclone as a Thickener of Coal Slurry. *Trans. AIME* (1948) **177**, 262-277; *Coal Tech.* (Feb. 1948) TP 2351.
- <sup>6</sup> J. W. Hyer: Heavy Density Separation—A Review of Its Literature. *Quarterly. Colo. School of Mines.* (1948)
- <sup>7</sup> Heavy-Media Separation Processes and The Dutch State Mines Cyclone Separator Processes for Coal Preparation. Min. Dressing Notes (16) American Cyanamid Co. (Feb. 1948).
- <sup>8</sup> D. A. Dahlstrom: Cyclone Operating Factors and Capacities on Coal and Refuse Slurries. *Trans. AIME* **184**, 331-344; *Min. Eng.*, Sept. 1949, TP 2633.
- <sup>9</sup> Lavage par centrifugation en milieu dense. Essai semi industriel à Götterborn Sarre. Centre d'Etudes et Recherches de Charbonnages de France. Note Technique 49/11 (May 1949).
- <sup>10</sup> Etude du fractionnement du cyclone l'apex. Centre d'Etudes et Recherches des Charbonnages de France. Note Technique 48/11 (Dec. 1948).
- <sup>11</sup> Recovery of the Coking Fraction from Witbank Fine Coal by means of the Cyclone Washer. Fuel Research Institute of South Africa.



# Automatic Controls on Sand Pumps

by William B. Stephenson

The paper describes efficient and effective methods of automatically controlling sand pump installations. Particular reference is made to liquid-level controls actuating variable speed pump-driving units. Included is a discussion of various combinations of equipment, with schematic drawings.

HERETOFORE many sand pump installations in the mining and metallurgical field involved guesswork as to the size of the pump and then simply direct-connecting it to a constant speed motor which was often large enough to do twice the work required. Considerable thought was given to mill location in an effort to avoid as much pumping as possible. Hillside design of mills with gravity flow

W. B. STEPHENSON, Member AIME, is Vice President, Allen-Sherman-Hoff Co., Philadelphia, Pa. AIME Columbus Meeting, October 1949.

TP 2871 B. Discussion (2 copies) may be sent to Transactions AIME before Aug. 31, 1950. Manuscript received Oct. 10, 1949.

of tailing was much to be desired. Power costs were low, construction was relatively cheap, and labor in many places meant nothing in comparison to the values derived from the ores being treated.

Today most of these conditions are reversed. Ores are becoming more complex with lower values per ton of ore treated. Power is more expensive, construction is higher, and labor definitely takes a very important place in the profit and loss statement.

Each piece of equipment incorporated in the design of a new mill is selected to provide highest efficiency based on careful metallurgical analysis of the ore to be treated both in good and poor markets. Each is designed with its flexibility in mind and with provision for adjustments in its performance to meet actual conditions in the operation of the mill.

Sand pumps have taken an important position in mill design. Operators the world over are relying more and more on the performance of a sand pump in the efficient operating of their plant. Many ordinary jobs are being done today with pumps driven through adjustable pitch diameter V belt drives in order that changes in pump speed can be made easily to meet actual operating conditions. Operating costs have long since proved the folly of operating a pump at too high a speed for the volume and head to be overcome. Metallurgical results have shown

the benefits derived from a smooth flow as contrasted with a surging flow from an uncontrolled pump. Construction costs have risen to such an extent that surge bins and other means formerly employed must be omitted from today's flowsheet.

Several installations of considerable size and involving the use of automatic pump speed control have proved the adaptability of such controls since they were placed in service in recent years.

In the following presentation each system discussed and illustrated schematically has been installed in one or more instances and has proved itself reliable and definitely economical from the standpoint of capital investment, operation and maintenance.

**Variable Volume-Head Speed Control:** Probably the most popular, effective and efficient automatic control setup for sand pumping installations is the one shown schematically in fig. 1. Commonly used

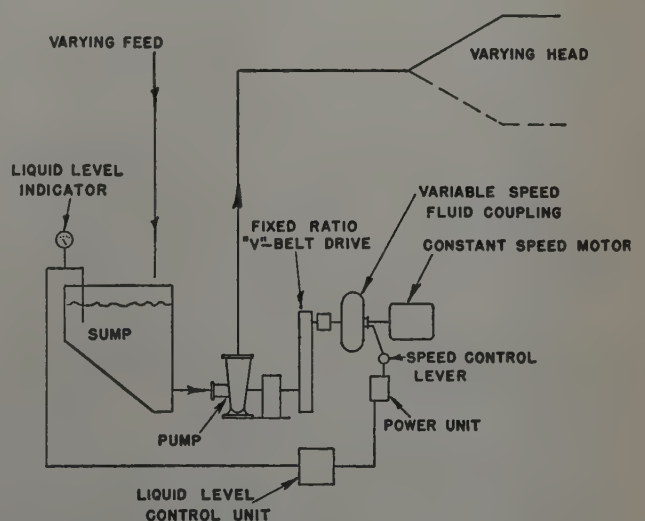


Fig. 1—Schematic arrangement for automatic control of volume and head.

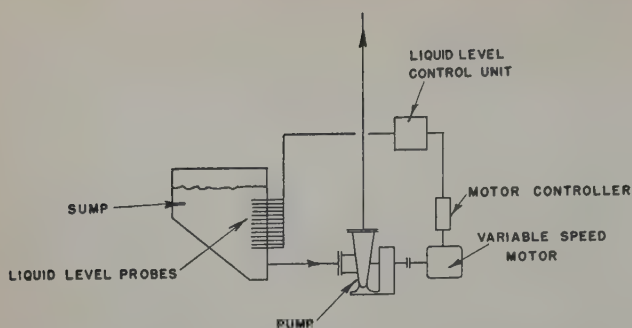


Fig. 2—Schematic arrangement for automatic speed control from liquid level of sump.

in connection with tailing disposal, this control equipment provides for handling a variable volume against a variable head without manual supervision. This type of control is particularly effective for a mill handling a constant tonnage of feed consisting of varying ores, which results in fluctuations in volume of tailing.

In cases where a tailing dam distribution system is employed and this control system is used the amount of pipeline can be changed at the will of the dam operator without concern as to results at the pumphouse.

Fig. 1 shows a conventional centrifugal sand pump being fed with a variable volume of pulp from a sump. The pump in this case is driven from a high speed squirrel cage motor through a variable speed hydraulic coupling and a fixed ratio V belt drive. The output speed of the coupling and consequently the speed of the sand pump are controlled by the positioning of an adjustable lever from the power unit of the liquid level control system. Any change or attempted change in the liquid level in the sump results in an increase or decrease in the pump speed.

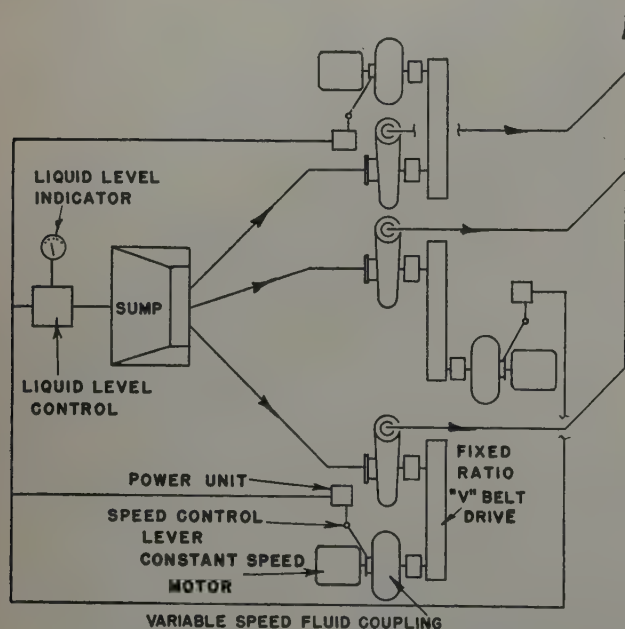


Fig. 3—Schematic arrangement for automatic control of volume and head. Three pumps in parallel.

This liquid level control system involves the use of low pressure air bubbled through a tube submerged in the pulp in the sump. Changes in pulp level create more or less resistance to the flow of the low pressure air. These changes are picked up by the receiving and transmitting instrument which in turn actuates the power unit to move the coupling speed control lever.

Instead of hydraulic couplings, some installations use direct current motors with which a rheostat is automatically adjusted. Wound rotor motors using automatically positioned drum controllers and even wound rotor motors using probes in the sump for liquid level control have been successful. Fig. 2 shows such an arrangement of equipment which does a good job but does not provide the smooth, non-step speed variation of the hydraulic, electromagnetic or direct current units.

Various combinations of pump equipment can be used with any of these types of control. One installation involving the handling of a wide range in

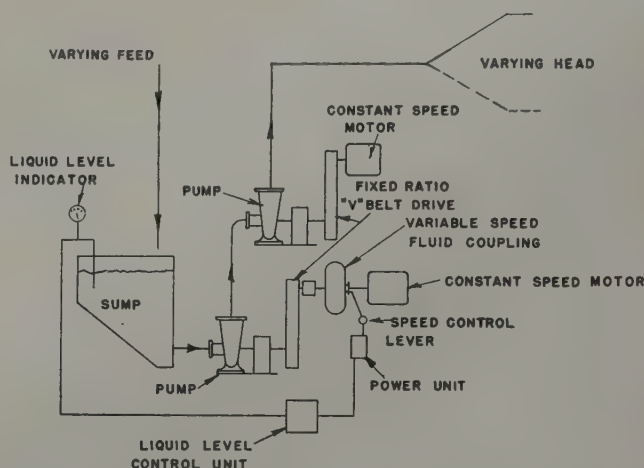


Fig. 4—Schematic arrangement for automatic control of volume and head pumps in series.

volumes uses three 12 in. pumps operating in parallel, each with variable speed hydraulic couplings controlled from a central panel tied in with the liquid level of a common sump as shown in fig. 3.

In cases where the total head on a system is greater than can be handled economically by one pump, two or three pump units connected in series in one pump station can be used. Fig. 4 shows such an arrangement with the first pump in the series driven through a variable speed controlled power unit regulated from the pulp level in the sump.

For slight changes in volume and only minor changes in head, fig. 5 shows the arrangement of equipment where the volume exceeded reasonable capacity limits of one 12 in. pump. Two of the three pumps shown are in constant operation, splitting the volume involved between the two. No. 1 pump is the variable speed unit, utilizing the same arrangement of control equipment as shown in fig. 1. No. 2



pump is a constant speed unit. The total volume handled in this case amounts to about 8000 gpm against a total head on each pump of about 26 ft. The third pump shown is a standby unit for either of the other two and is driven from a wound rotor motor with a drum controller for manual speed variation.

**Constant Discharge Volume Control:** Possibly more closely allied with the chemical field is the problem of pumping a constant volume. In processes where batching is necessary, tanks of considerable size must be pumped out. To put the suction of a centrifugal pump on the bottom of one of these tanks without some control definitely means that the pump will discharge an ever decreasing volume as the tank is emptied.

Fig. 6 shows the schematic arrangement of equipment for automatically maintaining a constant flow through a centrifugal pump as applied in the above case. A rubber-lined pinch type of valve is used on the tank discharge pipe leading to a transfer sump.

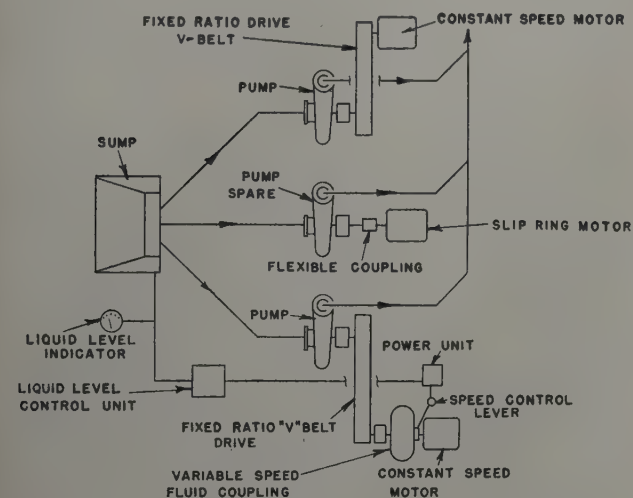


Fig. 5—Schematic arrangement for volume control. Two pumps in parallel, third pump spare.

The pump, taking its feed from the transfer sump, is driven from a squirrel cage motor through a fixed ratio V belt drive and is set, as to speed, to deliver a given volume against a constant head.

A liquid level control unit measures the level in the transfer sump and energizes the pinch valve to open or close, providing an increase or decrease in flow of pulp from the tank to satisfy the requirements of the sand pump.

**Filter Level Control:** Through the use of basically the same equipment as shown in fig. 1 it is possible to control the liquid level of the receiving vessel through changes in the speed of the pump. Fig. 7 shows the arrangement of the equipment in this case.

A good example of application of this arrangement is in controlling the level in a filter. Through control at its maximum point, higher efficiency may

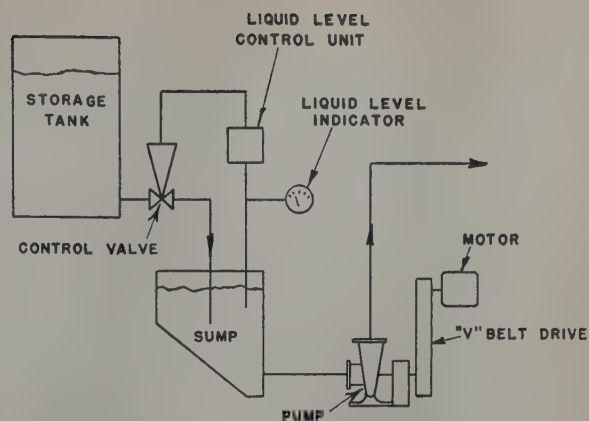


Fig. 6—Schematic arrangement for maintaining constant flow.

be expected. In varying grades of concentrates, effectiveness of filter cloths will have its effect on the level in the filter pan. Each change in level or tendency to change is picked up by the control system and results in an increase or decrease in pump speed.

One installation has gone so far as to direct-connect the suction of the sand pump to the under-flow of a thickener. This particular installation involves the use of direct current motor and a field rheostat positioned by the power unit of the level control. An electronic tube-type converter is used to change alternating current to direct current for the small motor.

**Density Control:** There have been several installations made of equipment designed to control the speed of pumps according to density of the pulp being handled. Most of these are to be found in Florida in the phosphate field and handle large volumes of pulp within not too close density ranges. One particular installation depends on its density control from the load on the thickener rake motor. With an increase in load of the motor, a pinch valve is automatically opened, thus relieving the load on the rake and discharging a variable volume to the sump beneath. At this point liquid level control comes into play and adjusts the speed of the pump to the

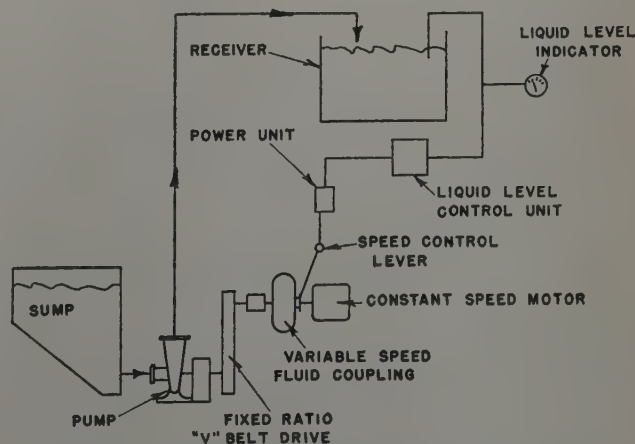


Fig. 7—Schematic arrangement for automatic control of liquid level in receiver.

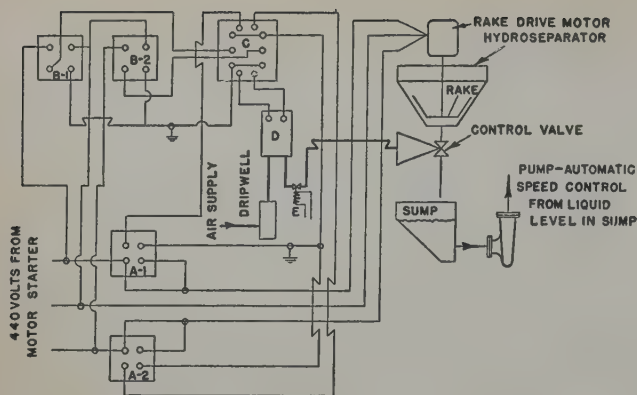


Fig. 8—Schematic arrangement for automatic density control of hydroseparator underflow.

A-1 and A-2—Current transformer  
B-1 and B-2—Potential transformer  
C—Thermal converter  
D—Recording controller  
E—Solenoid air valve (to close on power failure) Air piping shown heavy.

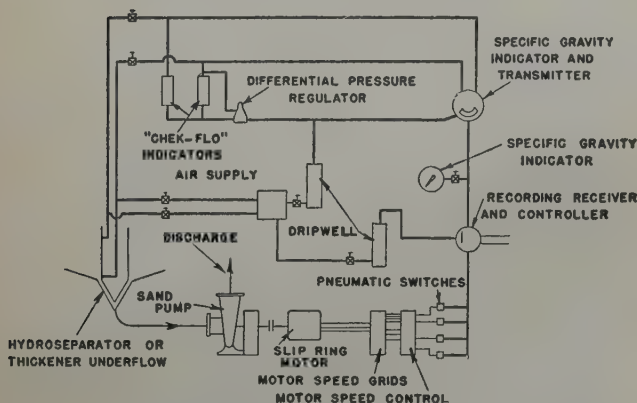


Fig. 9—Schematic arrangement for automatic density pump speed control as applied to "Torque tube" type thickeners and hydroseparators.

volume to be handled. Fig. 8 shows the arrangement of the equipment used in this case.

Another such installation uses a control system involving pressure differential readings continuously taken through taps in the torque tube of the thickener mast near the cone. Changes in these readings mean a change in density of the mass at that point and through control equipment automatically result in changes of pump speed. This arrangement of equipment is shown in fig. 9.

**Control Accessories:** Most of this discussion has been devoted to the control of the speed of a pump. Fig. 10 shows a rather ingenious device recently seen in use in the west in connection with control of the feed to a sand pump. A considerable volume of tailing is being reclaimed, hauled some distance, repulped, and pumped against very unusual heads. No unsurmountable difficulties were encountered except for an unexpected amount of trash which continually plugged the suction, the pump, or the distribution lines.

The device as shown is simple but very effective. Instead of the usual open end suction in a mixing tank, this suction is a capped, perforated pipe in a vertical position. Over the perforation is a close fit-

ting collar which is continually moved up and down. The result is uninterrupted pumping between weekend shut downs during which the accumulated trash in the tank bottom is cleaned out.

On a recent visit to Mexico, another individual development was seen which may be of interest. At this particular plant, thickened concentrates going to the filters are constantly troublesome because of their slimy nature. From our observation, it is doubtful that any automatic control on filter level would ever work at this plant. Filter bags are continually blinding.

The thickener discharge flows by gravity through a pipe with a gooseneck on it connected by a swivel joint to the tank bottom. By means of a system of ropes and pulleys going to the remote filter floor, it is possible for the operator to control the density of the pulp entering the filter. It is amazing to watch the underflow thicken as the gooseneck pipe is raised. The change from a thin pulp in low position to a thick pulp in high position is almost instantaneous. The flow decreases with an increase in density and the pump, running at constant speed, gulps and gasps and discharges in surges, but the operator does have control over his filter level when it is required.

### Summary

Summing up, there is no new piece of apparatus used in most known sand pump installations which are automatically controlled. Systems as developed today, to handle a particular problem, simply involve a combination of thoroughly proven apparatus. To determine what is most necessary to be controlled is the important point and the benefits derived from controlled equipment are numerous.

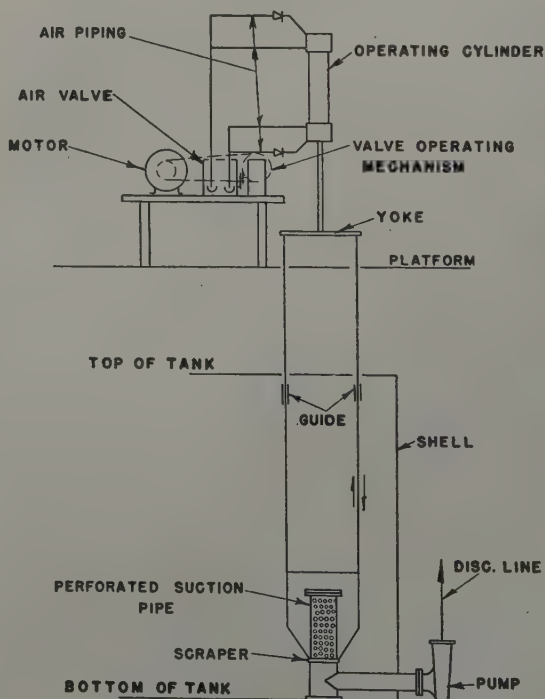


Fig. 10—Schematic arrangement of continuous operating suction strainer cleaner.



# Low-Temperature Coke as a Reactive Carbon

by C. E. Leshner

**T**HIS paper reports a study of the reactivity of 950°F and 1650°F cokes as measured by relative rates of reduction of iron oxides at temperatures up to 2200°F. Previous work cited shows general acceptance of the theory that reduction by carbon is a gaseous reaction, and that kind and character of carbon as well as particle size have measurable effect on the velocity of reaction. As will be shown, the data obtained in this study confirm those conclusions. The work was not designed to examine iron oxide reduction equilibrium, but if reaction velocity be defined as the speed with which "a reaction tends to approach conditions of equilibrium," the data here presented may be considered as a study of reaction rates, and the relative degree of reduction to metallic iron as the measure of reactivity.

Three standardized combinations of Lake Superior brown iron ore with carbon were tested by similar procedures. One combination was a mechanical mixture of carefully sized high-temperature coke (1650°F) with the ore. The second was a mechanical mixture of the ore with Disco\* obtained by

\* "Disco" is the registered trade name of a solid fuel made by the Disco process of low-temperature carbonization of coal.

carbonizing the identical coal at 950°F. The third was an agglomerate prepared by carbonizing the coal and ore at 950°F, premixed in proportions to give as nearly as possible the same relative amounts of carbon and ore as the mechanical mixtures. This agglomerate, obtained by heating the finely divided ore (through 30 mesh) with coking coal through the plastic temperature range so as to form solid aggregates, gives a product in which the oxide particles are impregnated with, and intimately bound together with low-temperature coke.

The agglomerate—ore-Disco—was most active in oxide reduction; the mechanical mixtures of Disco and ore next in order, with coke the least reactive.

**General Discussion:** Carbon exists in many forms and it is well known that the form or nature of the carbon used in reduction of oxides is related to the critical temperature of reduction. Sugar carbon, charcoal, and lampblack are forms of carbon that will reduce oxides at lower temperatures than high-temperature coke, and coke will, in turn, give a lower critical reduction temperature than graphite. There have been many investigations of this characteristic of carbons. Johnson<sup>1</sup> reported a difference of 130°F (70°C) in the critical reduction temperature of zinc oxide as between charcoal 1891°F (1033°C) and Acheson graphite turnings 2021°F (1105°C) with zinc oxide. Bodenstein<sup>2</sup> using charcoal and coke, found a difference of 138°F (77°C) comparing an experimental figure of 2066°F (1130°C) for coke and 1928°F (1053°C) for charcoal, in the reduction of zinc oxide.

He concluded that this is very marked and observed that the "type of carbon merely raises or lowers the temperature at which rapid reaction takes place." Comparing the effectiveness of types of carbon in reduction of zinc oxide, it was found that a "brown coal coke" gave 97 pct zinc elimination at 1832°F (1000°C), as compared with 48 pct with "hard coal coke."<sup>3</sup>

A wide range of metallic oxides was studied by Tammann and Sworykin,<sup>4</sup> who found that the temperature at which decomposition of oxides begins depends on the nature of the carbon used. Carbon in the form of graphite, lampblack, and sugar carbon was investigated. Sugar charcoal will reduce  $\text{Fe}_2\text{O}_3$  to  $\text{Fe}_3\text{O}_4$  at 842°F (450°C) as compared with 1112°F (600°C) for coke, according to Meyer.<sup>5</sup>

Direct reduction of iron oxides by charcoal begins at 1382°F (750°C), but "first becomes intense" at 1652°F (900°C), whereas with coke, direct reduction begins at 1742°F (950°C), and "first becomes appreciable" at 2012°F (1100°C).<sup>6</sup> The total reduction of the sample under certain conditions when heated in a current of CO with charcoal was about 100 pct for limonite and about 77 pct for magnetite. Using coke under the same conditions, the respective percentages were 75 and 47. In a study of processes for sponge iron<sup>7</sup> by the Bureau of Mines, the conclusion was reached that a low-temperature char from noncoking subbituminous coal is the most satisfactory solid reducing agent.

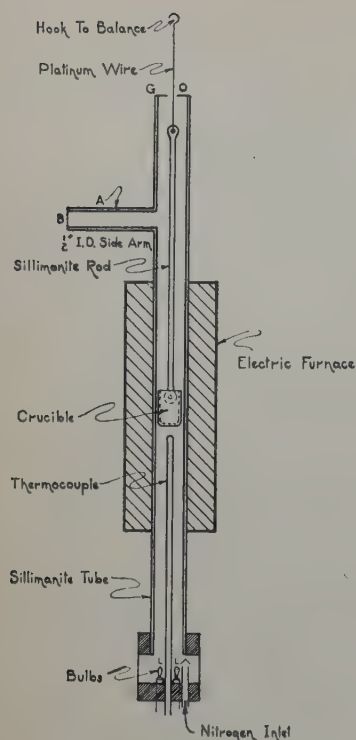
In a critical study of zinc smelting from a theoretical viewpoint Maier<sup>8</sup> concluded that the reduction is by CO, that the reaction between ZnO and CO is intrinsically more rapid than the subsequent reduction of  $\text{CO}_2$  by C, which is limited by diffusion rates, which in part effectively limits the smelting process. Maier said that the operation is improved with the activity of the reducing carbon. An active carbon, he said, is one maintaining a low  $\text{CO}_2$  content in the retort.

**Reactivity of Carbon:** One form of carbon is more potent in reducing oxides than another. A carbon that reacts faster than another at a given temperature is said to be more reactive. Reactivity is measured by several methods, using carbon dioxide, air, or steam as reactants.<sup>9</sup> Sebastian and Mayers<sup>10</sup> have developed a method for the determination of absolute reaction rates between coke and oxygen by a study of ignition points under certain conditions.

These and other investigators have established the relative reactivity of types of carbon. Lignite, charcoal, bituminous coal, cokes in the ascending order

C. E. LESHNER, Member AIME, is President, Disco Co., Pittsburgh, Pa.

TP 2874 F. Discussion (2 copies) may be sent to Transactions AIME before Aug. 31, 1950. Manuscript received June 6, 1949; revision received Jan. 20, 1950.



**Fig. 1—Furnace tube for gaseous reduction tests.**

of temperature of carbonization, are progressively less reactive. The carbon that is the more easily oxidized by  $\text{CO}_2$  is the carbon that will be more useful in smelting zinc, to the extent that the rate of reduction of  $\text{ZnO}$  is dependent on the slower reaction between carbon and  $\text{CO}_2$  for its reactant,  $\text{CO}$ .

Most of the study of reactivity of carbon has been done on coke, from that carbonized at  $932^\circ\text{F}$  ( $500^\circ\text{C}$ ) to  $1832^\circ\text{F}$  ( $1000^\circ\text{C}$ ). In a study of cokes from five bituminous coals, Reynolds and Davis<sup>9</sup> conclude that the reactivity of coke from a given coal decreases with increase in the carbonizing temperature.

There appears to be a clear parallelism between the reactivity of carbons with the gases mentioned, and their rates of reaction in reduction of metallic oxides. Why one form of carbon is more reactive than another is a question yet to be answered. Reynolds and Davis could find no relation between the rank of the coals and the reactivity of the corresponding cokes. Although cokes made at the same temperatures using five different coals showed a wide range of reactivity, the most reactive low-temperature coke combined with carbon dioxide about twice as rapidly as the least reactive coke made at the same temperature. It has been observed that amorphous forms of carbon are more reactive than crystalline forms.<sup>11</sup>

It is known, of course, that low-temperature cokes yield volatile matter when heated above the temperatures of their formation. The gas given off is mainly hydrogen, which, like  $\text{CO}$ , will reduce oxides. The rate of gas evolution<sup>9</sup> after the first hour is fairly constant from  $932^\circ\text{F}$  ( $500^\circ\text{C}$ ) and  $1292^\circ\text{F}$  ( $700^\circ\text{C}$ ) cokes. That is to say, even though a  $932^\circ\text{F}$  ( $500^\circ\text{C}$ ) coke will undergo changes in the direction of high-temperature coke when recarbonized, it will not only react with carbon dioxide more rapidly at  $1112^\circ\text{F}$  ( $600^\circ\text{C}$ ) and  $1292^\circ\text{F}$  ( $700^\circ\text{C}$ ) than at  $932^\circ\text{F}$  ( $500^\circ\text{C}$ ), but it is evolving hydrogen with some carbon monoxide and traces of other gases, throughout the heating period.

**Influence of Particle Size:** The rate at which an oxide is reduced by carbon is not only related to the reactivity of the carbon, but to the particle size of the oxide as well. The finer the particle size of oxide and of carbon and the more intimately they are associated in the reaction zone, the faster the rate of reduction of the oxide by the carbon.

If the reduction is exclusively, or even primarily, a solid to solid reaction, it is exceedingly important that the particles of both oxide and carbon be very small as well as intimately mixed. A flue dust imbedded in low-temperature coke would represent a practical example of such a condition. However, if a solid oxide and carbon do react at all, "the reaction would soon stop since the contact between the (zinc) oxide and carbon would shortly be broken by the vaporization of the products of reaction."<sup>12</sup>

The more generally accepted theory is that oxides are reduced by  $\text{CO}$ , the reaction giving  $\text{CO}_2$ , which in turn is reduced by carbon to  $\text{CO}$ . Thus the oxide is reduced by  $\text{CO}$ ; the  $\text{CO}_2$  by carbon. Two solids therefore—the oxide and the carbon—when heated give as end products the metal and carbon dioxide. To initiate the gaseous reactions, some gas, e.g. air (oxygen),  $\text{CO}$ , or  $\text{CO}_2$  must be present in some quantity, however small. It has been suggested that the solids may have sufficient vapor pressure at elevated temperatures to supply this gas,<sup>13</sup> or that there is always sufficient entrained air to initiate the reduction.

In this conception, the metal oxide is reduced and  $\text{CO}$  is oxidized to  $\text{CO}_2$ . The molecule  $\text{CO}_2$  must in turn be reduced by coming into contact with a molecule of carbon. Under like conditions of temperature and pressure, the time required for reduction of a given quantity of oxide will be governed by the sum of all the times it takes for molecules of  $\text{CO}$  to travel from the solid carbon to the solid oxide, and as  $\text{CO}_2$ , back to carbon; that is to say, the rate of diffusion. The shorter the paths, the more rapid the reaction and since surface films impede reaction, the more surface, the more reaction there will be.

Intimacy of contact and fineness of particle may be had by fine grinding, but an impalpable powder so produced would offer almost a maximum of resistance to gas flow. A quiescent mass of dust may have almost infinite surface and yet be relatively impenetrable to both liquids and gases. Whether

**Table I. Analyses of Materials Screen Analyses**

	Coal	Ore
+ 4 mesh	0	0
4x8	0.5	
8x14	24.0	
14x28	32.8	
+ 28 mesh		0
28x48	16.9	24.0
48x100	11.5	28.2
100x200	9.7	33.5
—200	4.6	14.3
	100.0	100.0

**Analyses of the Cokes, Dry Basis**

Carbonization Temperature	850°F	1652°F
Volatile matter	18.1	2.0
Ash	6.7	11.3
Fixed carbon	75.2	86.7
Sulphur	0.7	0.95
Btu	13,650	12,660



reduction<sup>2, 12-17</sup> be by solid or gas, the particle size is important in determining rate.

**Oxide-Carbon Agglomerates:** The process by which finely divided oxides are "wrapped up" in "balls" of reactive, low-temperature coke was developed from the Disco process of carbonizing coal. In the Disco process, coking coal and finely divided "breeze" are heated and carbonized in revolving kilns or retorts. The coking coal is softened and the plastic mixture, when coked or baked so that it becomes solid and hard, is the Disco "ball." Substituting iron ore for the breeze gives a "ball" in which each particle of ore is imbedded in coke, being coated with carbon.<sup>18</sup>

**Preparation of Standard Samples:** Samples of Lake Superior iron ore, low-temperature coke, high-temperature coke, and an agglomerate of ore and low-temperature coke were used in all tests. The cokes were made from low-sulphur coking coal from the Pittsburgh bed. High-temperature coke (900°C, 1652°F) was made from this sample of coal by the Bureau of Mines in the standard AGA-BM retort.

Minus 30-mesh ore used in these tests contained 59.25 pct of iron, equivalent to 84.65 pct of  $\text{Fe}_2\text{O}_3$ . The ore-Disco was made by carbonizing to 850°F in a batch retort a mixture of 60 lb of ore and 40 lb of coal. The product was crushed and the 4x8-mesh size screened out for testing. The low-temperature coke forming the matrix was made under identical conditions with the 850° coke made from coal alone. Screen analyses of ore and coal as charged were as shown in table I.

The agglomerate contained 67.8 pct ore and 32.2 pct low-temperature coke. The calculated volatile matter in the agglomerate was 5.8 pct and fixed carbon 24.1 pct.

**Experimental Procedure with Iron Oxide:** Two laboratory procedures were followed in the study of reduction of iron oxide. In both, samples of ore and reducing agent were heated in a vertical furnace, suspended in a small capsule or crucible. One procedure was to weigh and record the weight loss progressively; the other to collect and weigh the gases given off progressively during the test. All tests were made in a crucible surrounded by a current of purified nitrogen. The furnace charges were 3 g of the agglomerate or 2.035 g (67.8 pct) of —30-mesh ore and 0.965 g (32.2 pct) of coke. This provided an ample amount of excess carbon which remained in the crucible at the end of the heating period.

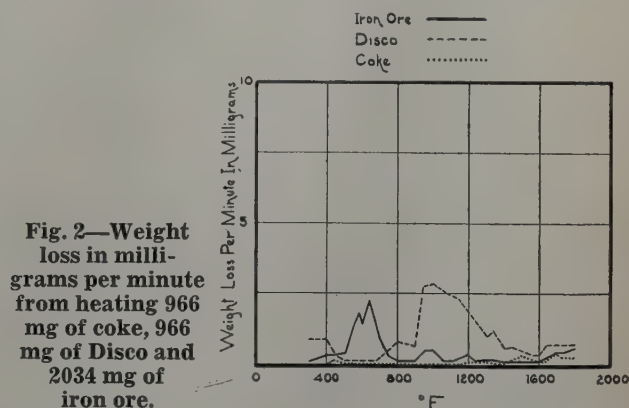
**Apparatus:** The apparatus, fig. 1, consisted of a sillimanite tube 1 in. id and 32 in. long with a side arm at right angles 7 in. from the upper end, supported vertically within a Sentry electric furnace. Temperatures were read with a Brown indicating pyrometer up to 1500°F and with an indicating and controlling Leeds & Northrup potentiometer above 1500°F.

The crucible containing the reactants, ¾ in. od, 9/16 in. id, and 2 in. long, was suspended in the furnace by a sillimanite rod. Connecting the upper end of this rod with one arm of a balance was a platinum wire of such a length as to hang the crucible in the middle zone of the furnace. A slow stream of nitrogen was fed in the bottom of this vertical tube, and allowed to come out around the space between the wire and the edges of the hole in the glass plate, and at the junction of the glass plate and the top of

the furnace tube. A cluster of four small light bulbs was placed in the bottom cool part of this long tube so that by having a hole in the bottom of the balance case it was possible to see down the length of the tube to properly align it before starting a test, and until the crucible reached a red heat, after which the light bulbs were unnecessary.

The tip of the protection tube of the thermocouple was located ⅜ in. below the crucible bottom, when the balance arm was in the horizontal position. When studying the gaseous products, the upper end of the tube was sealed after supporting the crucible, and the gas taken out the side arm of the furnace tube.

In preparation for a test by the weight loss



method, the crucible was weighed empty and weighed again containing a charge of 3 g. The timing of each test began when current was turned on in the Globar furnace. The heating rate was about 10°F per min, reaching 1000°F at 90 min, 1300°F in 120 min, 1800°F in 170 min, 2000°F in 190 min, and 2200°F in 210 min. Thus the weight loss may be correlated with temperature and with time with considerable accuracy.

The weight loss method of testing was used in two ways. The first method was to reach a maximum temperature and hold the crucible at that temperature until there was little or no more loss of weight. At that point the power was shut off, the nitrogen flow increased to insure no sucking back of air. The other weight loss method was to heat to the maximum temperature being studied, and immediately upon reaching that temperature to shut off the power and increase the nitrogen flow in an attempt to cool the furnace and to stop the reaction. In either case, the crucible was cooled before removal from the furnace tube.

The crucible and contents were weighed before and after each test and again after the residue was removed. Crucible plus residue weight was subtracted from the crucible plus charge weight at the start of the test to get overall weight loss, which usually checked very closely with the weight loss as determined while running. The entire residue was ground to —100-mesh in agate mortar, weighed, and a free-iron determination made by the mercuric chloride method.

**Test Results with Weight Loss Procedure:** The data at various temperatures are presented in figs. 2, 3, and 4. Except for the tests in which ore, coke, and Disco were heated alone, the standard crucible charge was 3 g. The total weight loss of the charge materials when heated alone, fig. 2, was insignificant. The iron ore had its greatest loss at between 600° and 700°, when water of hydration or crystallization was driven off; Disco had its highest loss

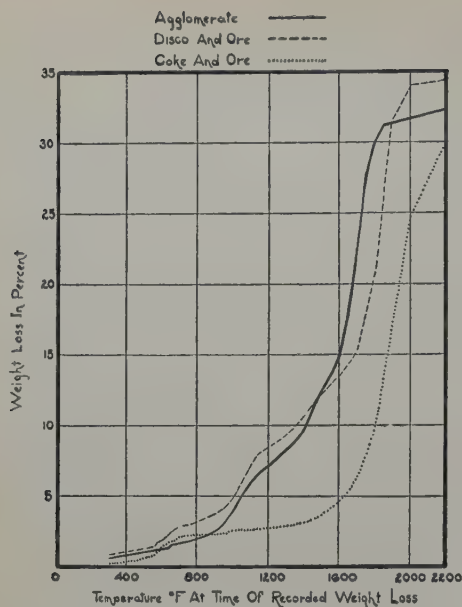


Fig. 3—Cumulative weight losses in percentage of original sample.

between 900° and 1200°, when volatile matter was driven off; and, as was expected, coke alone had very little loss in weight at any point in the test.

Cumulative data for weight losses of the ore with each of the three reactants are given in fig. 3. The cumulative percentages of weight loss for each of the three reactants are closely parallel up to 800°, above which the Disco mixtures lose weight much more rapidly than the coke mixture.

The rates of weight loss expressed in milligrams per minute were calculated from the original data averaged for all tests. These averages were plotted and the weight loss for each 50°F, and 5-min interval, were read from the plots. The figures for weight losses in milligrams per minute are the averages per minute for each such 50° interval. The areas below

each of the three curves in fig. 4 are approximately equal, but the respective rates of weight loss are significantly different. The subsidiary peaks at 600° with the mechanical mixtures arise from the loss of water of crystallization in the ore, which had been driven off the agglomerate when it was made. The peaks at 1100° with Disco are attributed about one-half to loss of volatile matter from the Disco and the remainder to reduction of higher oxides of iron to FeO or Fe<sub>3</sub>O<sub>4</sub> by the hydrogen in that volatile matter.

Similar peaks in both these curves, and absent from the coke data, are recorded at about 1400°. The accelerated rates at this point are attributed to further reduction in the oxides, for as is shown later, there is very little evidence of reduction to metallic iron at this temperature.

The rate of weight loss—a measure of gas evolution which in turn is a measure of the rate of the reduction of the iron oxides—was most rapid for the Disco agglomerate. Starting upward sharply at 1500°, it rose to a peak at 1750°, and then dropped as sharply as it had risen, to a low point at 1900°.

The rate of reduction of ore with Disco mechanically mixed began to rise rapidly at 1600°, reached a peak at 1850°, and the reaction was far from completion at 2200°.

Coke and ore mixed reacted at more gradual rates; beginning at 1400°, the rise in weight losses of the reactants mounted to a peak at from 1950° to 2000°. The reaction was continuing with considerable speed at 2200°F.

There were 24 test runs in which the weight losses were determined. The preceding data are averaged from these tests in which there were no great departures from the averages.

**Gas Recovery Tests:** The second series of tests in the same heating furnace and with the same reactants in which the gases involved were collected and weighed gave significant results with respect to the relative rates of reduction of the ore by different reducing carbons. Reproducible results were ob-

Table II. Material Balances and Weights of Gases Collected, in Milligrams

Run No.	Maximum Temperature Momentarily Reached °F	Weight Gases Collected					Material Balance			
		H <sub>2</sub> O	CO <sub>2</sub>	H <sub>2</sub>	CO	Total Gases	Weight Residue	Residues Plus Gases	Weight Charge	Percent Over or Under
Iron Ore and Coke										
61	1,600	94.5	68.0	0.0	2.5	165.0	2,860.0	3,025.0	3,006.0	+0.63
Avg. 49-58	1,800	104.5	180.3	1.0	29.4	315.6	2,721.5	3,037.6	3,005.5	+1.05
66	1,900	96.0	300.0	1.3	185.0	582.3	2,474.0	3,056.3	3,008.0	+1.60
Avg. 52-55	2,000	105.0	323.0	1.68	323.0	753.4	2,249.2	3,002.6	3,008.7	—0.20
Avg. 64-75	2,200	144.0	316.0	5.9	445.0	911.4	2,069.0	2,980.2	3,008.5	—0.96
Iron Ore and Disco										
60	1,600	272.0	139.0	8.1	41.6	460.7	2,595.5	3,056.2	3,007.0	—0.27
68	1,700	223.0	157.0	8.2	76.5	464.7	2,538.0	3,002.7	3,008.0	—0.17
Avg. 48-57	1,800	236.0	222.0	9.8	184.0	652.0	2,351.0	3,003.1	3,004.0	—0.03
65	1,900	224.0	288.0	9.7	406.0	927.7	2,048.5	2,976.2	3,009.0	—1.09
Avg. 51-53	2,000	240.0	299.0	11.13	434.0	984.1	1,995.2	2,979.4	3,008.2	—0.95
Avg. 69-72-73	2,200	237.0	296.0	11.67	47.0	1,015.7	1,972.3	2,988.0	3,007.9	—0.99
Ore-Disco Agglomerate										
70	1,500	167.0	133.0	9.4	44.5	353.9	2,646.0	2,999.9	3,003.0	—0.10
59	1,600	159.5	155.0	11.1	80.0	405.6	2,595.0	3,000.6	3,001.5	—0.03
67	1,700	162.0	272.0	11.9	255.5	701.4	2,286.0	2,987.4	3,001.0	—0.45
Avg. 47-56	1,800	194.5	328.0	13.3	381.0	916.05	2,109.0	3,025.5	2,502.0	+0.77
Avg. 50-54	2,000	196.0	339.0	13.2	403.0	952.5	2,023.2	3,025.7	3,003.5	+0.75
Avg. 62-71	2,200	200.5	358.0	13.8	443.0	1,018.05	2,038.5	3,054.5	3,003.0	+1.72
74-76										



tained when the tests were terminated at a predetermined maximum temperature and the produced gases swept out of the train with nitrogen. Attempts to collect and weigh first one and then another gas at short intervals in order to ascertain the current rates of gas evolution were inconclusive. The data in table II and fig. 5 showing the quantities of gases collected were obtained by terminating each test at a predetermined temperature and continuing the flow of nitrogen until the crucible had cooled.

With coke as the reducing agent, the gases evolved below 1600° were mainly water vapor and CO<sub>2</sub> with traces of hydrogen. Between 1800° and 1900°, both CO and CO<sub>2</sub> were given off at their maximum rates and above 2000°, the rates decreased rapidly. The ratios of CO to CO<sub>2</sub> in gases collected did not pass unity until the temperature was above 1900°. A ratio of two parts CO to one part CO<sub>2</sub> represents the minimum ratio possible for reduction to Fe at 1652°F (900°C).<sup>7</sup> With Disco as the reducing agent in mechanical mixture, there was measurable production of CO up to 1600°, about twice as much CO<sub>2</sub> and nearly three times as much water vapor as with coke. From 1700° to a peak at 1900°, CO and CO<sub>2</sub> were evolved at rapid rates; falling off as rapidly in the next 100° to 2000°. The ratio of CO to CO<sub>2</sub> increased rapidly from 0.47 at 1600° to 2.22 at 1900° and 2.28 at 2000°. No water vapor was found in the gases above 1500° and only traces of H<sub>2</sub>.

With the agglomerate the reactions were more rapid and at lower temperatures than with the mechanical mixtures. Peaks were reached at 1700° with rates of gas evolution declining to low points at 1900°. Ratios of CO to CO<sub>2</sub> were between 1.8 and 2.0 at 1800° and above.

**Material Balances:** For the series of tests numbered 47 to 76, in which the gases evolved at successively higher temperatures were weighed, material balances are given in table II. The weight of solid residue at the end of the test is added to the weight of gases collected and this total compared with the weight of original charge as 100. Although several of the individual tests were from 2 to 2.5

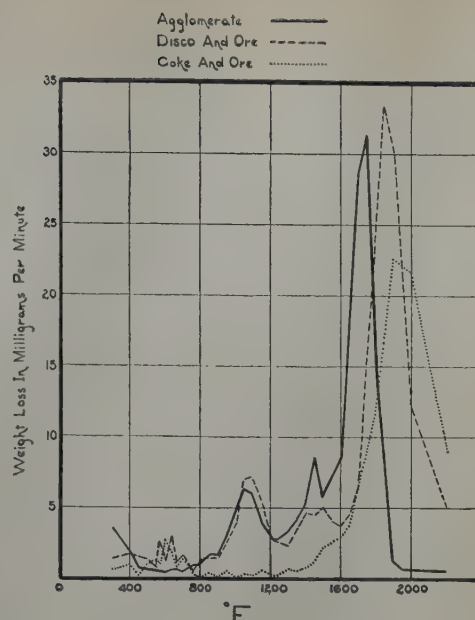
**Table III. Percentage Reduction of Iron Oxides to Metallic Iron**

First Series: Percentage of total oxide iron reduced to metallic iron at maximum temperature momentarily reached.

°F	Reduction Carbon		
	Coke	Disco	Ore-Disco Agglomerate
1,500			0.0
1,600		0.0	6.9
1,700		10.2	55.0
1,800	0.0	37.85	84.2
1,900	34.9	88.1	
2,000	71.0	93.0	88.4
2,200	99.15	100.0	97.7

Second Series: Percentage of total oxide iron reduced to metallic iron at maximum temperature at which held until there was little or no loss of weight.

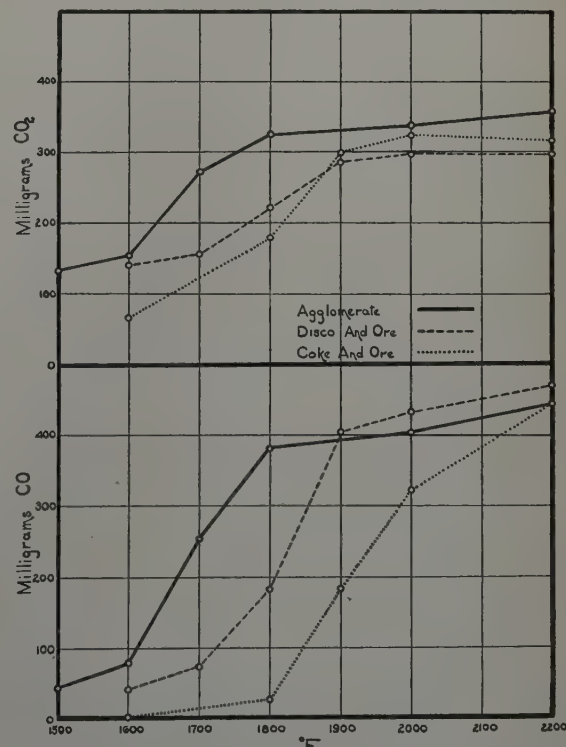
Reduction Carbon			
°F	Coke	Disco	Ore-Disco Agglomerate
1,400			0.3
1,500		4.35	28.3
1,600	1.98	44.90	75.7
1,700	64.5		
1,800	85.3	86.0	84.3
2,000	95.7	92.3	96.5



**Fig. 4—Weight losses in milligrams per minute on 3-g samples of coke and iron ore, Disco and iron ore, and ore-Disco agglomerate.**

pct over, none of the averages, as shown, were as much as 2 pct over or under.

**Reduction of Iron Oxides:** Results of reducing the charge of iron oxide with the three forms of carbon show that the agglomerate in which the oxide is intimately associated with the carbon is the most reactive, that Disco in mechanical mixture with oxide is less reactive, and that coke is the least reactive of the three carbons tested. Table III records the percentages of total iron that were found to have been reduced to metallic form by the several forms



**Fig. 5—Total weights of CO<sub>2</sub> and CO evolved to predetermined temperatures momentarily reached. 3-g samples.**

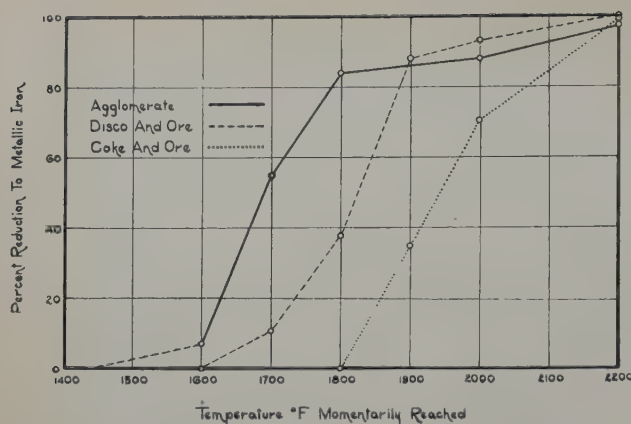


Fig. 6—Percentages of reduction of iron oxide to metallic iron at maximum temperatures momentarily reached.

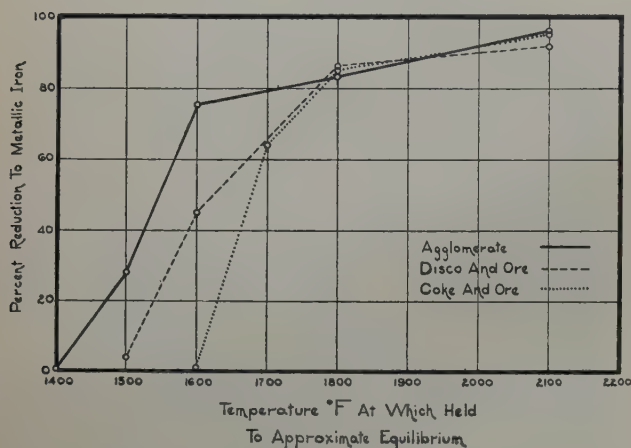


Fig. 7—Percentages of reduction of iron oxide to metallic iron at maximum temperatures at which held until approximate equilibrium.

of carbon at progressively higher retort temperatures. The figures are the averages from 45 tests at the stated temperatures. Figs. 6 and 7 present the data in graphic form. Measurable reduction to metallic iron was had with the agglomerate in the first series at 1600°, and at 1400° in the second series in which the heating was continued at uniform temperature. Reduction of 10 pct of the oxide to metallic iron was recorded with the mechanical mixture of Disco and ore at 1700° in the first series, 45 pct at 1600° in the second. No metallic iron was found at 1800° with coke and ore in the first series, and only 2 pct at 1600° in the second, or prolonged heating test.

Practically complete reduction to metallic iron was found in all tests that reached and were stopped at 2200°. The prolonged heating tests were carried out at maximum temperature of 2000° and gave consistent results for coke and agglomerate. With coke as the reducing carbon, the results, when the tests were stopped at 2000°, were 71 pct reduction as compared with 95.7 pct when the heating was continued. With the agglomerate, comparable figures were 88.4 and 96.5 pct. However, the mechanical mixture of Disco and ore showed approximately the same reduction—93.0 and 92.3 pct for both series.

What is more important than these total figures of percentages of reduction to metallic iron at the

top figures are the relative rates of reduction and the more rapid action of the Disco as compared with coke, particularly when in the agglomerate.

## Acknowledgments

To Lawrence A. Winter, formerly Research Engineer, Pittsburgh Coal Carbonization Co., predecessor of Disco Co., the author gives full and unstinted credit for the laboratory work and results on which the paper is based. Painstaking, careful laboratory procedures were developed by Mr. Winter over a period of more than two years (1943-1945). The description of methods and the tabulation of results have been prepared from his notes. Mr. Winter was also very helpful in a search of the literature preceding the laboratory investigations.

## References

- <sup>1</sup> Woolsey McA. Johnson: Physical Factors in the Metallurgical Reduction of Zinc Oxide. *Trans. AIME* (1907) **38**, 656-663.
- <sup>2</sup> Max Bodenstein: The Mechanism of the Metallurgical Production of Zinc. *Trans. Amer. Electrochem. Soc.* (1927) **51**, 365-372; discussion, 372-376.
- <sup>3</sup> C. F. Brenthel and I. A. Leroux: Einfluss von Koksart, Temperatur und Zeit auf die Reduktion von Zinkoxyd. *Metallu. Erz.* (1938) **35**, n.s. 26, 173-174.
- <sup>4</sup> G. Tammann and A. Sworykin: Zur Dynamik der Reduktion der Oxyde durch Kohle. *Ztsch. Anorg. Allgem. Chem.* (1928) **170**, 62-70.
- <sup>5</sup> Hans Heinz Meyer: Zur Reduktion der Eisenoxyde durch Kohlenstoff. *Mitt. Kaiser-Wilhelm-Inst. Eisenforsch. Dusseldorf.* (1930) **12**, 1-5.
- <sup>6</sup> A. M. Saitzew: Einfluss des Kohlenstoffes von Holzkohle und von Koks auf den Verlauf der Direkten Reduktion von Eisenoxyden. *Chem. Zentr. Part II*, p. 1474. 1938. From *Uralskaja Metallurgija*, No. 7, 9-17. 1937.
- <sup>7</sup> C. E. Williams, E. P. Barrett, and B. M. Larsen: Production of Sponge Iron. *U. S. Bur. Mines. Bull.* 270. (1927).
- <sup>8</sup> C. G. Maier: Zinc Smelting from a Chemical and Thermodynamic Viewpoint. *U. S. Bur. Mines. Bull.* 324. (1930).
- <sup>9</sup> D. A. Reynolds and J. D. Davis: Reactivity of Coke; Improved Method for Determining the Reactivity of Coke in Carbon Dioxide. *Ind. and Eng. Chem. Anal. Ed.* (1936) **8**, 33-36.
- <sup>10</sup> J. J. S. Sebastian and M. A. Mayers: Coke Reactivity; Determination by a Modified Ignition Point Method. *Ind. and Eng. Chem.* (1937) **29**, 1118-1124.
- <sup>11</sup> Jesse O. Betterton and Melville F. Perkins: Zinc Metallurgy. *U. S. Patent No. 2,219,914* (Oct. 29, 1940) 4 pp.
- <sup>12</sup> Woolsey McA. Johnson: The Reducibility of Metallic Oxides as Affected by Heat Treatment. *Trans. AIME* (1913) **47**, 219-227; discussion, 228-231.
- <sup>13</sup> Bohm Ivar: Om den indirekta och direkta reduktionen i belastermasugnen. (On the Direct and Indirect Reduction in the Blast Furnace.) *Jernkontorets Annaler*, (1934) **118**, 277-339. Also, *Chem. Abs.* (1934) **28**, 6668-6669.
- <sup>14</sup> A. Krupkovski and C. Taklinski: Le processus de la réaction ZnO-C aux températures élevées. *Chimie & Ind.* (1937) **37**, 275.
- <sup>15</sup> W. Leitgeb: Zur Entwicklung der Thermischen Zinkgewinnung u. Zinkreinigung. *Metallwirtschaft* (1936) **15**, 955-960; 1088-1091, 1148-1150.
- <sup>16</sup> Chas. G. Maier and Oliver C. Ralston: The Gaseous Reduction of Zinc. *Trans. Amer. Electrochem. Soc.* (1927) **51**, 339-356. Discussion, 356-363.
- <sup>17</sup> P. H. Emmett: A Résumé of the Factors Influencing the Rate of the Gaseous Reduction of Metallic Oxides. *Trans. Amer. Electrochem. Soc.* **51**, 207-212. Discussion, 212-215.
- <sup>18</sup> U. S. Pat. 2,429,416.



# A New Theory of Comminution

by Fred C. Bond and Jen-Tung Wang

**Comminution energy is principally energy of deformation before breakage, which appears as heat. An empirical equation is presented which covers the entire comminution range. The new strain-energy theory considers comminution from the known principles of mechanics and the reduction ratio. Energy requirements according to the different theories are compared.**

**T**HE present status of the theory of comminution is extremely unsatisfactory. The amount of various ores and rock products crushed and ground annually approaches the staggering sum of perhaps one half billion tons. Yet the basic concepts underlying an operation of such magnitude are unknown, and actual knowledge of what takes place in comminution is almost entirely empirical.

The responsibility for this situation cannot be charged to apathy or lack of appreciation of the importance of the problem. Many attempts have been made to evolve a workable hypothesis, but none has been completely successful.

## General Considerations

In order to break rock, it must be subjected to a stress which strains the rock beyond its critical breaking point. The stress imparts energy to the rock, most of which is released in the form of heat when the stress and resulting strain are removed. If the strain induced exceeds the critical strain, the energy is released by breaking; if the induced strain does not reach the critical point, the energy is released with removal of the stress as energy of resilience. Rock is a brittle material and it is assumed that it breaks approximately at its yield point, so that no permanent deformation results from strains below the critical.

When rock is broken the total energy input is accounted for by the heat liberated, and by the surface energy of the new surface produced. A small and probably negligible amount is released as noise.

Rock is commonly broken under compression and the applied stress is ordinarily compressive. However, for breakage to occur, it is only necessary that the induced strain exceed the critical value, and this strain may represent the resultant of compressive, shearing, and tensile forces. The stress-strain diagram of rocks under compression, and the modulus of elasticity as determined therefrom, should be fundamental considerations in the development of any theories of rock breakage.

Crushing in jaw or gyratory crushers results primarily from a squeezing action, and reduction in a hammer mill is the result of impact. The action in rod and ball mills is a combination of impacting,

squeezing, and wearing away by attrition, or rubbing.

In ordinary crushing and grinding, the forces are applied at protruding points and are not distributed evenly throughout the rock. This is one of the features of comminution which theoretical considerations have not covered adequately. Another feature is the effect of impact velocities, which may greatly influence the total energy required for breakage. Impact velocities may vary from 1 or 2 fps in crushers to 10 or 20 fps for ball mills, and as high as 100 or more for hammer mills. However, according to the findings of geophysicists, the velocity of sound and of compression waves in stone is many times greater; the primary longitudinal compression waves travel at perhaps 15,000 fps, and the secondary transverse waves, which may reach the surface and cause cracks to form thereon, travel at perhaps 5000 fps.

When a crack tip forms, the total surrounding stresses are concentrated in this tip, which rapidly extends throughout the rock particle.<sup>1</sup> It would seem that the energy required to deform the rock beyond its critical strain, resulting in the formation of a crack tip, represents practically all of the energy required. Most of the primary crack tips presumably form on the surface.

A simple method of designating feed and product size is necessary for the evaluation of crushing and grinding results. Taggart's suggestion<sup>2</sup> that the 80 pct passing size be used has been found very practical. In this paper, the square opening screen size which 80 pct of the feed passes is designated as the feed size  $F$ , and the size which 80 pct of the product passes, or the product size, is designated as  $P$ . The reduction ratio at the 80 pct passing size, or  $F/P$  is designated as  $n$ . The feed and prod-

*FRED C. BOND, Member AIME, is Technical Director, Basic Industries Research, Allis-Chalmers Manufacturing Co., Milwaukee, Wis., and JEN-TUNG WANG is Professor of Machine Design, Chekiang University, China; formerly on leave of absence at Allis-Chalmers Manufacturing Co., Milwaukee, Wis.*

*AIME Salt Lake City Meeting, September 1950.*

*TP 2902 B. Discussion (2 copies) may be sent to Transactions AIME before Sept. 29, 1950. Manuscript received Nov. 22, 1948; revision received April 12, 1950.*

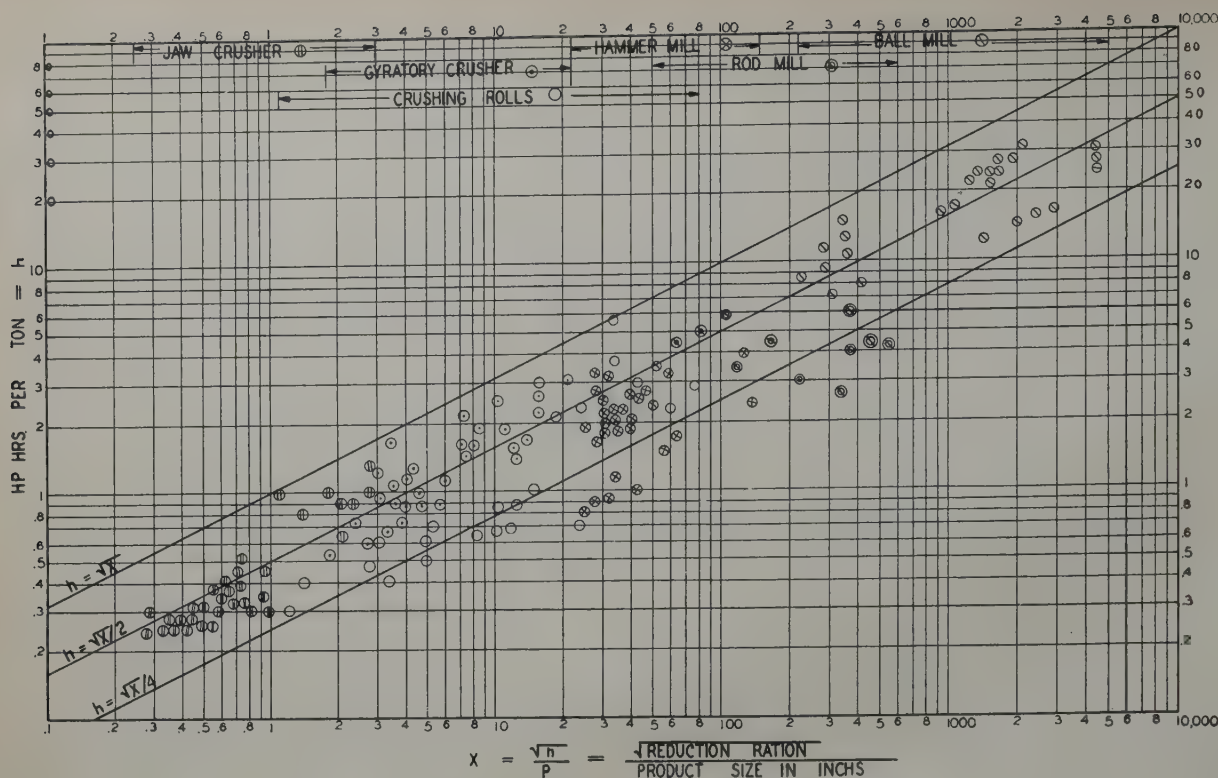


Fig. 1—Empirical energy chart.

$h$  - hp hr per short ton (total energy input).  
 $X = \sqrt{h}/P$ .  
 $F$  - Feed size, size in inches 80 pct of feed passes.  
 $P$  - product size, size in inches 80 pct of product passes.  
 $n$  - reduction ratio at Product Size =  $F/P$ .

uct sizes are determined from the intercepts of the 80 pct passing line with the plotted size distribution and may be expressed either in inches or microns.

### An Empirical Energy Chart

The energy required to crush and grind many different materials in different machines was tabulated and plotted in various manners. It was found by J. T. Wang, one of the authors of this paper, that these data could be shown consistently on a log-log plot of the energy input required in hp-hr per short ton vs. the square root of the quantity, feed size  $F$  divided by the cube of the product size  $P$ , which is equal to the square root of the reduction ratio  $n$  over  $P$ , or  $\sqrt{n/P}$ .

This chart is shown in fig. 1, and includes jaw crushers, gyratory crushers, crushing rolls, hammer mills, rod mills, and ball mills. It is seen from fig. 1 that the hp-hr per ton required for the average ore equals approximately one half of the term  $F^{3/4}$  divided by  $P^{3/4}$ , which equals  $0.5\sqrt{n^{3/4}/P}$ . This energy is doubled for very hard stone and halved for soft material. The product size  $P$  is in inches.

The chart indicates that

$$\text{hp-hr per ton} = K_1 \sqrt{n^{3/4}/P} \quad [1]$$

where  $K_1$  is 0.25 for soft, 0.50 for medium, and 1.00 for hard material.

This chart has been of considerable practical value in predicting approximate energy requirements. It can be used whenever the feed and product sizes

are known, and an assumption can be made regarding the relative resistance of the material to comminution. The energy required for hammer mills and rolls is slightly less than the average, indicating that these machines may be relatively more efficient. The chart is entirely empirical in origin, since no comminution theory was considered in tabulating and plotting the data.

### The Rittinger Theory

Over 80 years ago, Rittinger<sup>8</sup> stated his conclusion that the *useful work accomplished* in crushing and grinding is directly proportional to the new surface area produced, and to the reciprocals of the product particle diameters. This statement is merely a more or less arbitrary definition of useful work. However, he was evidently guided by the idea that all of the energy of crushing went into new surface,<sup>4</sup> and this implication is the basis of the Rittinger theory, which has been a center of controversy for many years. A. M. Gaudin<sup>5</sup> supports the Rittinger theory and restates it as follows: "Hence the efficiency of a comminuting operation is the ratio of the surface energy produced to the kinetic energy expended."

Energy input must be the product of force times distance, and the Rittinger theory completely ignores large variations in the distance (strain dimension or deformation) throughout which a force must act to produce breakage of different materials. The theory would perhaps have been in disrepute long ago, except for the disconcerting fact that



under certain assumptions it agrees fairly well with actual crushing and grinding results.<sup>6,7</sup>

Rittinger's theory indicates that a large portion of the work done in crushing or grinding is expended on the surface areas of the very fine particles produced. The particle size distribution of the —200-mesh fractions must be known accurately in order to evaluate the theory properly. Accurate size distribution measurements below 200 mesh, or 74 microns, were rarely accomplished until recently, and at the present time measurements below 10 microns are rare and many are of doubtful validity. In order to validate Rittinger's theory, an assumption must be made below the finest sieve size, or below the limit of the finest size measured by elutriation methods. This extrapolation was made possible by Gaudin's distribution equation<sup>8</sup> which as later modified<sup>9</sup> showed that

$$Y = 100 (x/K)^m \quad [2]$$

where  $Y$  is the percent weight passing any diameter  $x$ ,  $K$  is the diameter which 100 pct passes, and  $m$  is the slope of the log-log plot of the distribution line. This equation has been found to be approximately correct for the size distribution of any homogeneous product of crushing or grinding. It originated from study of screen analyses plots, and it has been shown later that it holds when the measured size distribution is extended down to about 13 microns by Infrasiser measurement. The probabilities are that it extends downward consistently to at least the colloidal size range. It follows a straight line log-log plot of the percent weight passing vs. the particle diameter in microns. It can be modified by replacing  $K$  in eq 2 with feed or product sizes  $F$  or  $P$  and 100 with 80.

An additional assumption is necessary before Rittinger's theory can be applied successfully, when using Gaudin's distribution equation. That is the existence of a grind limit, or a size limit above which the production of new surface area represents energy input; unless such a grind limit is assumed the total Gaudin surface area is indeterminate. It has been determined experimentally<sup>10</sup> that the Rittinger theory could be made to agree fairly well with the known energy inputs required in grinding if the Gaudin distribution line is extended to a grind limit of 0.7 micron, which happens to be approximately the beginning of the colloidal size range of rock. Weinig<sup>11</sup> had previously shown that a grind limit of one micron is probable. The surface area  $SA$  of cubical or spherical particles in square meters per 100 cc of solid can be calculated from the equation<sup>10</sup>

$$SA = \frac{600m}{K(1-m)} \left[ \left( \frac{K}{L} \right)^{1-m} - 1 \right] \quad [3]$$

where  $L$  is the grind limit of 0.7 micron.

When a cube of height  $a$  is reduced to cubes of height  $a/n$ , the number of breaks formed in each of the three dimensions is  $n-1$ . The total number of breaks is  $3(n-1)$ , and the total new surface area is  $6(n-1)a^2$ .

If  $a$  is in inches and  $w$  equals the energy in foot-pounds required to form 1 sq in. of new surface, the total energy required to reduce the cube is  $6(n-1)a^2w$ . The energy  $h$  in hp-hr required to reduce one ton (2000 lb) of cubes of size  $a$  to size  $a/n$ , with specific gravity  $Sp$ , is found from

$$h = 0.1680 (n-1) w / a Sp \quad [4]$$

The constant 0.1680 is designated as  $K_s$ .

When the log-log plot of the percent weight passing vs. the particle size yields parallel straight lines for the feed and the product, the reduction ratio  $n$  remains constant for all sizes.

Most tests and calculations using the Rittinger theory are based upon the reduction of feed of a given particle size  $F$  to products of various sizes. However, if the product size  $P$  remains constant and the feed size varies, the theory indicates that the surface production and the energy consumption will vary as  $2(n-1)/n$ . If a given weight of cubes of material, each of  $n$  units size, is reduced to cubes of unit size, the new surface formed is proportional to  $2(n-1)/n$ . Actual measurements indicate that as the feed size increases the energy consumption increases more rapidly than the theory would indicate. This appears to be true for both crushing and grinding.

Direct measurements of the surface energy of solids have never been made. But certain indirect and possibly questionable methods indicate that the surface energy is of the order of 300 to 900 ergs per sq cm.<sup>4</sup> Since the actual input required is perhaps 1000 times this amount it has been frequently stated<sup>12,13</sup> that the efficiency of grinding is of the order of 0.1 pct. Gross and Zimmerley<sup>14</sup> calculated that the net energy required to reduce quartz was 55,785 ergs per sq cm of total surface area produced, or about 56 joules per sq m. This is about 10 pct of the gross energy input required in commercial grinding mills to produce surface above the grind limit of 0.7 micron. A general treatment of this subject by John Gross has been published by the U. S. Bureau of Mines.<sup>4</sup>

If some method of comminution could be developed which would not require the expenditure of energy for deformation, which energy is liberated as heat, it might be possible to increase the mechanical efficiency as much as one thousand fold. However, as long as deformation is required for crushing and grinding, it is somewhat misleading to speak of efficiencies of 0.1 pct.

### The Kick Theory

More than 60 years ago, Friedrich Kick calculated<sup>15</sup> from a consideration of the stress-strain diagram that the energy required in comminution is independent of the feed or product size, and depends only upon the reduction ratio. The theory has been stated variously as: (1) "the energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of the bodies"; and (2) "For any unit weight of ore particles the energy required to produce any desired reduction in volume of all the particles in the mass is constant no matter what may be the original size of the particle."

An original mathematical treatment of Kick's theory is as follows:

Let  $K_s$  represent the energy in hp-hr per ton required to reduce one ton of stones of size  $a$  to size  $a/2$ . This represents one reduction step, and  $r$  steps are required for reduction to size  $a/n$ . It follows that the reduction ratio  $n$  equals  $2^r$ , and  $r$  equals  $\log n$  divided by  $\log 2$ .

The total energy required is the sum of that required for each step. In the first step, size  $a$  is reduced to  $a/2$ , and in the second step size  $a/2$  is reduced to  $a/4$ . In the final step,  $a/2^{r-1}$  is reduced to  $a/2^r$ . In each ton there are  $2^r$  times the number

of particles that were present in the preceding reduction step. Therefore, the total energy  $h$  required for reduction is

$$h = K_s + 2^3 K_s (1/2)^3 + K_s 4^3 (1/4)^3 + \dots$$

$$K_s (2^{r-1})^3 (1/2^{r-1})^3 = r K_s = K_s (\log n / \log 2) \quad [5]$$

Eq 5 shows that according to Kick's theory the energy required for each reduction step varies directly as  $\log n$  divided by  $\log 2$ , and is independent of the feed size.

H. Stadler<sup>16, 17</sup> vigorously supported Kick's theory and condemned that of Rittinger. From the stress-strain diagram, he derived a scale of ordinal energy units proportional to the energy input required according to Kick's theory. These Stadler ordinal numbers equal  $-10$  times the log of mesh aperture in inches, and are proportional to  $\log n / \log 2$ . However, his illustrations of the agreement of the theory with the actual energy input required in crushing and grinding are not conclusive.

A complete mathematical exposition of Kick's theory was published by Taggart.<sup>7</sup>

### The Strain Energy Theory

This theory was evolved by Jen-Tung Wang and is presented here for the first time. It was derived from a mathematical consideration of the forces and energy involved in the deformation of elastic materials.

It is assumed that the stone particles before and after crushing take the shape of cubes, just as was done in analyzing the energy per ton of stone crushed using Rittinger's and Kick's theories.

Let  $a$  equal initial size of stone in inches,

$n$  equal reduction ratio, or  $F/P$ ,

$Sp$  equal specific gravity of the material,

$S$  equal ultimate compressive strength of stone in pounds per square inch,

$E$  equal Young's Modulus in pounds per square inch.

Then the force required to crush equals  $Sa^2$ .

For the purpose of this analysis, it is considered that the reduction of the stone particles from size  $a$  to size  $a/n$ , under compression parallel to the  $X$  axis, takes the following steps:

1. First consider a break perpendicular to the  $X$  axis dividing the stone into two pieces with thicknesses in the  $X$  direction equal to  $(a)(n-1)/n$  and  $a/n$  respectively.

In this action:

$$\text{Force} = Sa^2$$

$$\text{Unit strain} = S/E$$

$$\text{Total strain} = Sa/E$$

The total energy of resilience equals one half force times strain, or  $1/2$  times  $Sa^2$  times  $Sa/E$ , or  $S^2 a^3 / 2E$ .

2. The next step is dividing the piece with thickness  $(a)(n-1)/n$  into two pieces of thickness  $(a)(n-2)/n$  and  $a/n$ . The slabs that have been already reduced to a thickness  $a/n$  in the  $X$  direction need no more reduction in this particular direction.

In this step:

$$\text{Force} = Sa^2$$

$$\text{Unit strain} = S/E$$

$$\text{Total strain} = (S/E)[(n-1)/n](a)$$

$$\text{Total energy} = (S^2 a^3 / 2E)[(n-1)/n]$$

3. The action is continued until all the slabs have thickness  $a/n$  in the  $X$  direction. It will take

altogether  $(n-1)$  steps. The total energy ( $U_x$ ) required in this process of reduction in the  $X$  direction is:

$$U_x = \frac{S^2 a^3}{2E} + \frac{S^2 a^3}{2E} \left( \frac{n-1}{n} \right) + \frac{S^2 a^3}{2E} \left( \frac{n-2}{n} \right) + \dots + \frac{S^2 a^3}{2E} \left( \frac{2}{n} \right)$$

$$= \frac{S^2 a^3}{2En} [n + (n-1) + (n-2) + \dots + 2]$$

$$= \frac{S^2 a^3}{2En} [(n-0) + (n-1) + (n-2) + \dots + \{n-(n-2)\}]$$

$$= \frac{S^2 a^3}{2En} [n(n-1) - \{0 + 1 + 2 + \dots + (n-2)\}]$$

$$= \frac{S^2 a^3}{2E} \left[ (n-1) - \frac{1/2(0 + n-2)(n-1)}{n} \right]$$

$$= \frac{S^2 a^3}{2E} \left[ (n-1) - \frac{(n-1)(n-2)}{2n} \right]$$

$$= \frac{S^2 a^3}{2E} \left( \frac{n-1}{n} \right) \left[ n - \frac{n-2}{2} \right]$$

$$= \frac{S^2 a^3}{4E} \frac{(n-1)(n+2)}{n}$$

A similar amount of energy is required to break into seams perpendicular to each of the  $Y$  and  $Z$  directions.

Therefore, the total energy  $U$ , equals  $U_x$  equals  $U_y$ , and

$$\Sigma U \text{ equals } \frac{3S^2 a^3}{4E} \cdot \frac{(n+2)(n-1)}{n} \text{ for each cube.}$$

It follows that the energy  $h$  in hp-hr per short ton required to crush is

$$h = \left[ \frac{0.001748 S^2}{SpE} \right] \left[ \frac{(n+2)(n-1)}{n} \right] \quad [6]$$

The constant 0.001748 is designated as  $K_s$ .

According to both Kick's theory, eq 5, and the strain energy theory, eq 6, the energy required for comminution is independent of the particle size concerned, and is a function of the reduction ratio  $n$ . According to Kick's theory it is proportional to  $\log n / \log 2$ , while according to the strain energy theory it is proportional to  $(n+2)(n-1)/n$ , or almost directly proportional to  $n$ . As the reduction ratio increases, the energy required for comminu-

Table I. Energy Units Required

$n$ Reduction Ratio of Unit Cube ( $a = 1$ ) ( $P = 1/n$ )	No. of Cubes Formed	Ritt- inger New Surface Theoret- ical ( $n-1$ )	Kick  $\log n$ $\log 2$	Strain Energy  $\frac{(n+2)(n-1)}{n}$	Empir- ical Eq 1  $\frac{n^{1/4}}{\sqrt{P}}$  $= n^{3/4}$
1	1	0	0	0	0
2	8	1	1	1	1
3	27	2	1.584	1.67	1.355
4	64	3	2.000	2.25	1.682
5	125	4	2.320	2.80	1.986
6	216	5	2.584	3.33	2.280
7	343	6	2.808	3.85	2.558
8	512	7	3.000	4.37	2.828
1	1	0	0	0	0
2	8	0.143	0.333	0.229	0.354
3	27	0.286	0.528	0.382	0.479
4	64	0.428	0.666	0.515	0.595
5	125	0.572	0.773	0.641	0.702
6	216	0.714	0.861	0.763	0.807
7	343	0.857	0.936	0.882	0.905
8	512	1	1	1	1



tion increases much more rapidly according to the strain energy theory than it does according to Kick's theory. The derivation of Kick's theory is based upon a stage by stage reduction, while the strain energy theory is based upon a generalized reduction ratio of any value. The strain energy theory assigns a greater proportion of the total energy input to the fine size reductions than Kick's theory, and thus appears to fit the facts more closely.

The strain energy theory indicates that the energy required for crushing varies inversely as the modulus of elasticity and specific gravity, and directly as the square of the compressive strength and as the approximate reduction ratio.

The accuracy of extrapolation below the finest measured particle size affects the Rittinger results much more than it does the Kick or strain energy results.

### A Modified Rittinger Method

It has been shown that if the feed size  $F$  remains constant, the energy required according to Rittinger's theory varies inversely as the product size  $P$ . It also can be shown, according to Rittinger's theory, that if the product size remains constant and the feed size varies, the energy for a constant volume of feed will vary as  $2(n-1)/n$ . Combining these expressions shows that when both feed and product size vary the energy  $h$  required in hp-hr per ton should be proportional to  $2(n-1)/Pn$ .

If  $K_r$  represents the hp-hr required to reduce one ton of 1-in. cubes to  $\frac{1}{2}$ -in. cubes the energy required is

$$h = K_r (n-1) / Pn \quad [7]$$

The constant  $K_r$  should be numerically equal to the  $K_s$  used in Kick's theory if the comminution efficiencies are equal.

The above expression assumes that the slopes of the feed and product distribution lines are equal and does not give the same results as Rittinger's theory, when the surface areas of the feed and product are calculated directly with a slope of 0.7 and a grind limit of 0.7 micron; it is designated here as the modified Rittinger method.

### Separate Size Fraction Calculations

If the size distribution data are sufficiently complete, the energy required to reduce each size frac-

tion separately can be calculated, using the plotted distribution line only for extrapolation below the finest size determination. It is assumed in this calculation that the largest size fraction of the feed goes into the largest size or sizes in the product. This is not necessarily true in actual crushing and grinding operations, but with a homogeneous material the statistical effect should be the same, and the reduction ratio of each size fraction is determined on this basis. When the standard  $\sqrt{2}$  screen scale is used, the reduction ratio  $n$  for any size equals  $\sqrt{2}$  to the exponent of the number of screen intervals between the feed and product sizes. The relative energy for each size fraction can be calculated for any comminution theory, using the appropriate function of the reduction ratio and feed or product size, and the summation of these values should be somewhat more accurate than the calculations in which a straight line size distribution is assumed. The particle size used for each fraction is the average between the retaining sieve opening and the next larger sieve through which it has passed.

That portion of the product which passes the finest testing sieve is considered to have the same reduction ratio as the lower half of the sized material.

This method is particularly applicable where the feed is scalped, or has undersize removed by screening, since in this case the log-log plot of the distribution line is not straight.

### Comparison of Theories

Table I has been computed to show the relationship of the different theories for different reduction ratios of one cube of unit size ( $a = 1$  in.). In the first half of the table, the relative energy requirements are calculated for reduction ratios of 1 to 8, inclusive, on the assumption that one unit of energy is required in all theories for a reduction ratio of 2. In the second half of the table, it is assumed that one energy unit is required in all theories for a reduction ratio of 8.

The theoretical Rittinger new surface and the modified Rittinger method have the same values in table I. However, the total energy required for reduction of a unit weight with a constant reduction ratio increases as the feed or product size decreases, according to the Rittinger theories and the empiri-

Table II. Critical Strain Necessary to Break Nine Varieties of Rock

Rock	Moduli/1,000,000			$\sigma$ Poisson's Ratio	S Compressive Strength (Crit. Stress)	Q S/E (Critical Strain)	C Impact Crushing Strength
	E Stress Strain	G Rigidity	K Bulk				
Quartzite	9.520	4.275	4.110	0.112	28,730	3.018	
Syenite	10.420	4.280	6.170	0.218	27,890	2.676	
Diabase and gabbro	13.030	5.390	7.490	0.207	25,610	1.966	20.11
Gneiss	10.520	4.765	4.440	0.104	22,190	2.109	
Granite	6.460	2.764	3.235	0.168	21,050	3.260	14.67
Slate	11.520	5.190	4.925	0.110	21,050	1.827	
Marble	6.380	2.657	3.540	0.200	14,500	2.272	
Limestone	9.030	3.680	5.490	0.227	13,660	1.513	11.45
Sandstone	7.540	3.215	3.980	0.172	10,530	1.396	
Average	9.380	4.010	4.715	0.169	20,579	2.227	15.41

All three moduli values (E, G, and K), and the compressive strength (S) are given in pounds per square inch. The critical strain (Q) is given in thousandths of an inch per inch of length.

The impact crushing strength (C) is the breaking strength in foot-pounds per inch of thickness, measured on the rock types tested in the Allis-Chalmers Laboratory.

$$G = \frac{E}{2(\sigma + 1)}$$

$$K = \frac{1}{\frac{9}{E} + \frac{3}{G}}$$

Table III. Grinding and Crushing Data

Pct Passing	Grinding		Crushing	
	Feed	Product	Feed	Product
1 + 1/2 in.			100.0	
1	100.0		70.0	
3/4	96.0		22.0	
1/2	93.2		7.5	
3/8	90.2	100.0	4.0	
3 mesh	83.0	99.0		100
4	72.3	98.0		91
6	59.6	97.0		72
8	43.0	96.5		57
10	31.5	95.9		48
14	23.0	94.0		38
20	16.6	91.7		30
28	12.5	86.9		23
35	9.6	76.0		18
48	8.2	67.3		15
65	6.6	58.5		12
100	5.1	50.0		10
150	3.3	40.0		8
200	2.2	31.8		6.5

cal equation, while it is independent of the size according to the Kick and strain energy theories.

Fig. 2 shows the relationship between the empirical eq 1 of fig. 1 for hard, medium, and soft stone; and the various theories for material of medium hardness. The material was assumed to be the average material shown in table II, with a modulus of elasticity  $E$  of 9,380,000 psi, an ultimate compressive strength of 20,579 psi, a critical strain of 0.002227 in. per in., a specific gravity of 2.65, a distribution line slope of 0.7, and a grind limit of 0.7 micron. Two feed sizes  $a$  were considered: (1) 50 cm or 19.69 in., and (2) 2 cm or 0.788 in., corresponding roughly to primary crushing and grinding respectively. Intermediate feed sizes should give lines lying between those shown, resulting in a family of curves for each theory. A sufficient number or reduction ratios were calculated to determine the position and curvature of both lines according to each theory.

It was assumed in all cases that the minimum breakage which would completely relieve the compressive stress on a cube is one break across the center of the cube parallel to each of the XYZ planes, which corresponds to a reduction ratio  $n$  of 2. An inch cube of the above material broken in this manner requires

(3)  $(20,579) (0.002227) / (2) (12)$ , or 5.724 ft-lb. One ton contains 20,900 cubes, and requires 119,700 ft-lb, or 0.06045 hp-hr.

A cube of 1 in. forms 6 sq in. of new surface, so that the surface production energy  $w$  is 0.954 ft-lb per sq in., 2004 joules per sq m, or 2,004,000 ergs per sq cm.

Substituting in eq 1 gives

$$0.06045 = K_1 \sqrt{2^{1/2} / 0.5}$$

from which  $K_1$  equals 0.036 hp-hr per ton. Since the actual energy required for average material shows  $K_1$  equal to 0.5, the indicated efficiency of comminution with the above assumptions is 7.2 pct, according to the empirical chart.

The Rittinger new surface areas were computed from eq 3, and the surface production energy  $w$  was converted to hp-hr per ton and plotted. It is seen that the Rittinger curves indicate a higher energy consumption than actually is required. The Rittinger primary crushing curve for average material is close to the actual energy required for hard material, and the Rittinger grinding curve is the highest curve on the chart. If the Rittinger theory is correct the mechanical efficiency of crushing is less than that of grinding. As the reduction ratio decreases below 3 the indicated energy required drops off more rapidly than the actual energy.

It should be emphasized that these curves are entirely dependent upon the grind limit of 0.7 micron, which was derived years ago from a completely different approach.<sup>10</sup>

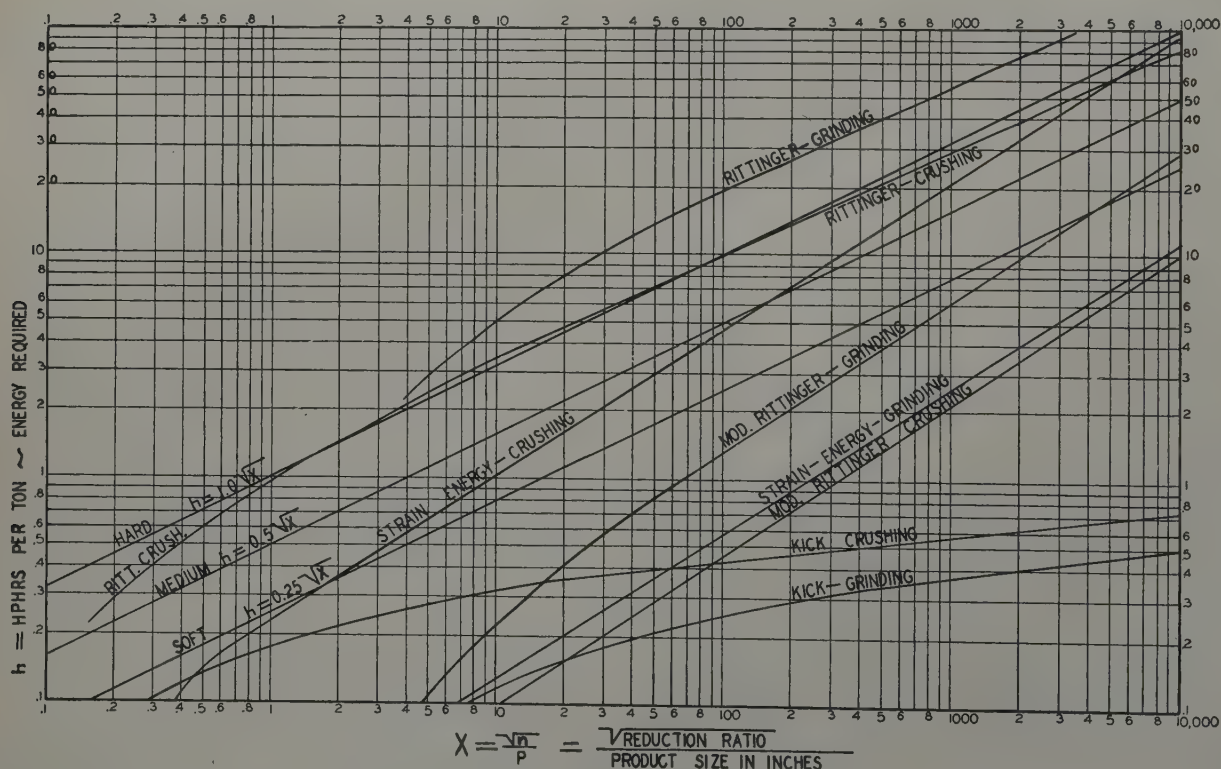


Fig. 2—Theoretical and empirical energy.



The Kick curves were computed from eq 5, with  $K_s$  equal to 0.06045 hp-hr per ton. The Kick energy is very much less than the actual energy, and is relatively low for grinding and high for crushing, the slope of the curves being much less. The 50-cm feed requires more energy than the 2-cm feed. The Kick curves correspond very poorly with actual energy requirements.

The strain energy curves were computed from eq 6. The term  $0.001748 S^2/(E) (Sp)$  has the value of 0.0297. The curves are somewhat steeper than the actual energy requirements, and the primary crushing requires much more energy than the grinding. The primary crushing curve corresponds with the actual energy requirements more closely than any other theoretical curve; however, as the feed size decreases the indicated mechanical efficiency decreases rapidly. The plot shows that if the strain energy theory is correct, crushing is a more efficient operation than grinding in tumbling mills; this appears reasonable because of the mechanical action in the machines. The true relative efficiencies of crushing and grinding cannot be determined until one comminution theory is accepted as correct.

The modified Rittinger values were computed from eq 7, in which the constant  $K_r$  has the value 0.06045. The product size  $P$  is in inches. The curves are roughly parallel to the strain energy curves. However, the primary crushing requires less energy than the grinding, and crushing is less efficient than grinding. Mechanical efficiency is lower than in the strain energy theory.

Table II has been prepared from the averages of published values.<sup>18</sup> It lists the critical strain, in thousandths of an inch per inch of thickness, required to break nine different classes of stone and the compressive strength or critical stresses of these stones. It is seen that the modulus of elasticity  $E$ , the modulus of rigidity or shear modulus  $G$ , and the bulk modulus or reciprocal of compressibility  $K$  all tend to decrease with the compressive strength although the correspondence is not close. Poisson's ratio ( $\sigma$ ) appears to be independent of the compressive strength. More accurate determinations of the modulus of elasticity are necessary for the proper application of the strain energy theory. The large variations in the critical strain values show the large possible errors resulting from the Rittinger theory in which the strain is not considered.

### Examples of Calculations

Operating data from a typical grinding and a typical crushing installation have been analyzed according to the different theories.

**Grinding:** Portland cement clinker was ground dry at the rate of 75 short tons per hour in open circuit in a preliminary ball mill 10 ft in diam by 11 ft 1 in. long inside, running at 17.9 rpm, with a ball charge of 110,000 lb.

The average temperature of the feed was 90°F, and the product was 145°F. Assuming a 10 pct radiation loss, and a specific heat of 0.221, all of the 10.6 hp-hr per ton energy input required to grind is accounted for as heat produced.

**Crushing:** Limestone scalped at 1/4 in. was crushed in open circuit in a gyratory crusher.

Table V exemplifies the separate size fraction calculations used in the grinding section of tables III and IV.

Table IV. Grinding and Crushing Data

	Grinding	Crushing
Material	Clinker	Limestone
Total hp-hr per ton, $h$	10.6	2.27
Specific gravity, $S_p$	3.15	2.60
Comp. strength, $S$	12,000	11,000
Modulus, $E$	$5 \times 10^6$	$7 \times 10^6$
Crit. strain, $Q$	0.0024 in.	0.00158 in.
Feed size, $F$	0.228 in.	1.14 in.
Product size, $P$	0.0185 in.	0.15 in.
Reduction ratio, $n$	12.3	7.6
Constant, $K_1$	0.77	0.53
Rittinger calculations		
Eq 3		
Feed, $m$	0.840	3.1
$K$	6,400	30,000
$SA$	1.6	0.03
Prod., $m$	0.640	0.64
$K$	440	5,400
$SA$	22.2	4.80
New surface	20.6	4.77
Joules per gram	31.4	6.72
Joules per sq m	480	366
Ft. lb per sq in., $w$	0.228	0.1745
Eq 4, $h$	0.603	0.0653
Ft. lb to break in. cube		
In mill (6 sq in.)	1.368	1.046
By compression	1.20	0.724
Pct efficiency	87.8	69.4
Kick calculations		
Eq 5, $K_s$	2.93	0.776
Ft. lb to break in. cube		
In mill	330	72
By compression	1.20	0.724
Pct efficiency	0.36	1.00
Strain energy calculations		
Eq 6, $h$	0.210	0.097
Pct eff. ( $h$ per actual hp-hr per ton)	1.98	4.27
Separate size calculations <sup>a</sup>		
Avg. $(n+2) (n-1) / h$	16.193	27.32
Eq 6, $h$	0.259	0.318
Pct eff. ( $h$ per actual hp-hr pr ton)	2.44	14.0
Modified Rittinger calculations		
Eq 7, $K_r$	0.94	0.392
Ft. lb to break in. cube		
In mill	106	36.4
By compression	1.20	0.724
Pct efficiency	1.13	1.99

<sup>a</sup> These calculations were made from the screen analyses as described in the text under the heading "Separate Size Fraction Calculations". They result in more accurate values of  $h$ , particularly when the feed is scalped.

Table V. Separate Size Fraction Calculations

Feed	Product	Screen Intervals	Percent Reduced	Reduction Ratio ( $n$ )	Strain Energy $(n+2) (n-1) / n$	Col. 4 Times Col. 6
1 in.	3/8 in.	3	0	2.83	3.12	0
3/4 in.	3 mesh	3	1.0	2.83	3.12	3.12
		4	1.0	4.00	4.50	4.50
		6	1.0	5.66	6.30	6.30
		8	0.5	8.00	8.75	4.37
		10	7	0.5	11.31	6.07
1/2 in.		10	6	0.1	8.00	0.87
		14	7	1.9	11.31	23.08
		20	8	0.8	16.00	13.50
3/8 in.		20	7	1.5	11.31	18.22
		28	8	1.5	16.00	25.31
3 mesh		28	7	3.3	11.31	40.60
		35	8	3.9	16.00	65.80
4		35	7	7.0	11.31	85.00
		48	8	3.7	16.00	62.40
6		48	7	5.0	11.31	60.70
		65	8	7.7	16.00	130.00
8		65	7	1.1	11.31	13.36
		100	8	8.5	16.00	143.50
		150	9	7.0	22.62	164.60
10		150	8	3.0	16.00	50.60
		200	9	8.2	23.52	192.90
Sum	—200	7.82 (avg.)	68.2	(15.00)	915.87	1,114.80
			31.8		Total .....	504.50
						1,619.30

### Summary and Conclusions

As empirical equation for the total energy input required in crushing and grinding has been derived. The energy in hp-hr per short ton of average material equals one half the square root of the term —reduction ratio to the one half power divided by the product size in inches. This energy is doubled

for very hard materials and halved for very soft materials. The product size is the size which 80 pct of the product passes. The approximate energy input required for any crushing or grinding installation can be calculated from this equation.

Practically all of the energy required is energy of resilience, which deforms the rock beyond its elastic limit, and is released as heat after breaking or release; the amount absorbed as increased surface energy is negligible. There is no known way of breaking rock without first deforming it beyond its elastic limit, excepting fatigue failures. The deformation or critical strain at breakage varies widely for different materials.

The mechanical efficiency of comminution might be defined as the percentage of the total energy input that results in deformation beyond the critical strain and consequent breakage. Contacts which cause deformation below the critical strain are wasted except as they result in fatigue breaks.

A new theory of comminution has been developed called the strain energy theory. According to this theory, the energy required varies directly as  $(n+2)(n-1)/n$ , where  $n$  is the reduction ratio and is independent of the feed or product size. It also varies directly as the square of the compressive strength, and inversely as the modulus of elasticity. In primary crushing, it apparently agrees with actual energy requirements better than any other theory, and indicates that crushing installations are more efficient than grinding, which appears reasonable.

There are two mutually contradictory methods of measuring the work done in comminution. In one method, the work is dependent upon the reduction ratio as calculated by the strain energy and Kick theories. These theories have a mathematical background, and the strain energy theory as applied to cubes can be adapted from the known principles of mechanics and the strength of materials. The energy required to break cubes of cement clinker and limestone, according to the strain energy theory, ranged from 2 to 14 pct of the actual energy required to grind or crush. The reduction ratio at the sizes which 80 pct of the feed and product pass is usually sufficiently accurate to serve as a measure of the work done if the log-log plots of the feed and product distribution lines are approximately parallel. However, if the feed has had the fines removed by screening, the weighted reduction ratio should be calculated over the entire size distribution range. The Kick theory was derived from a consideration of stage reductions and indicates lower efficiencies than the strain energy theory.

In the other method, the work done is measured by the new surface area produced, as first proposed by Rittinger, and later defined by Gaudin as, "... the efficiency of a comminuting operation is the ratio of the surface energy produced to the kinetic energy expended."<sup>10</sup> This theory cannot be justified mathematically, since work is the product of force times distance, and the distance factor is ignored. In two examples listed, the energy required for the initial break of cubes showed the comminution efficiency to be 88 pct and 68 pct. The energy input required per unit of new surface area produced is called surface production energy, and averages perhaps one thousand times the true surface energy. It is principally the energy required for deformation to beyond the critical breaking strain, which is liberated as heat.

The correct theory will not necessarily correspond with the empirical eq 1, since the latter may include wide variations in the present operating efficiencies of reducing large and small particles.

The strain energy theory was evolved to provide a working formula, which would apply over the entire crushing and grinding range and would have a rigorous mathematical background based upon mechanics. Although the Kick theory has such a background, it obviously assigns too large a portion of the total energy input to reduction of the largest particles. The Rittinger theory is not tenable from the viewpoint of mechanics and has been criticized as assigning too much energy to reduction of the very fine particles, while requiring the use of a more or less arbitrary grind limit. Between the two extremes, the strain energy theory appears to form a rationalization which may lead ultimately to a more satisfactory concept of the theory of crushing and grinding.

### Acknowledgments

The authors wish to thank the following members of the Allis-Chalmers organization for encouragement and ideas in the exploration of this highly controversial subject: F. Nagler, F. T. Agthe, B. H. Irwin, F. R. Gruner, and W. Mitchell, Jr.

Data were supplied for the preparation of the empirical energy chart by J. Furlong, A. J. Jorgensen, E. C. Greisen, and G. W. Shores.

### References

- <sup>1</sup> Eugene F. Poncelet: Fracture and Comminution of Brittle Solids. *Trans. AIME* (1946) **169**, 37. *Min. Tech.*, May 1944, TP 1684.
- <sup>2</sup> Arthur F. Taggart: New Units of Crusher Capacity and Crusher Efficiency. *Trans. AIME* (1943) **153**, 319.
- <sup>3</sup> P. Ritter von Rittinger: *Lehrbuch der Aufbereitungskunde*. 1867. Berlin.
- <sup>4</sup> John Gross: Crushing and Grinding. U.S. Bur. Mines. *Bull.* 402.
- <sup>5</sup> A. M. Gaudin: Principles of Mineral Dressing. Chapter VI, (1939). McGraw-Hill Pub. Co.
- <sup>6</sup> John Gross and S. R. Zimmerley: Crushing and Grinding, I—Surface Measurement of Quartz Particles. *Trans. AIME* (1928) **30**, 7.
- <sup>7</sup> Arthur F. Taggart: The Work of Crushing. *Trans. AIME* (1914) **48**, 153.
- <sup>8</sup> A. M. Gaudin: An Investigation of Crushing Phenomena. *Trans. AIME* (1926) **73**, 253.
- <sup>9</sup> R. Schuhmann, Jr.: Principles of Comminution—Size Distribution and Surface Calculations. *AIME Min. Tech.* (July 1940) TP 1189.
- <sup>10</sup> Fred C. Bond: Measuring Surface Area in Grinding. *AIME Min. Tech.* (March 1941) TP 1296.
- <sup>11</sup> A. J. Weinig: A Functional Size Analysis of Ore Grinds. Colo. School of Mines. *Quarterly*, **28**. (July 1933).
- <sup>12</sup> Geoffrey Martin: Tube Mill Grinding. *Inst. of Chem. Eng.* (Jan. 13, 1926).
- <sup>13</sup> S. G. Lipsett et al.: The Surface Energy of Solid Sodium Chloride. *Jnl. Amer. Chem. Soc.* (1928) **50**, 2701.
- <sup>14</sup> John Gross and S. R. Zimmerley: Crushing and Grinding Studies of Quartz. U. S. Bur. Mines. *R. I.* **2880** (1928).
- <sup>15</sup> Friedrich Kick: *Das Gesetz der proportionalen Widerstande und seine Anwendung*. 1885. Leipzig.
- <sup>16</sup> H. Stadler: The Law of Crushing I. *Eng. and Min. Jnl.* (Nov. 21, 1914).
- <sup>17</sup> H. Stadler: The Law of Crushing II. *Eng. and Min. Jnl.* (Nov. 28, 1914).
- <sup>18</sup> Handbook of Physical Constants: Geol. Soc. of Amer. Special Papers No. 36. (1942).
- <sup>19</sup> Page 130 of ref. 5.



# An Improved Method of Gravity Concentration in the Fine-Size Range

by Arvid Thunaes and H. Rush Spedden

Pilot plant test work in 1942 and 1943 showed that by a combination of desliming, fine-size classification, and Sullivan deck concentration it is possible to recover heavy minerals such as cassiterite at least as fine as 10 microns in size. This appreciable improvement in gravity concentration practice has been substantiated by several full-sized plants.

IN the past, mills treating ores of tin and tungsten by gravity concentration have recovered very little mineral finer than 325 mesh, although some form of slime concentration has been generally attempted by the use of buddles or round tables. This paper describes a series of pilot plant tests made in 1942 and 1943 in which the use of the Sullivan deck was investigated for the recovery of an appreciable amount of the fine values formerly lost.

The investigation was initiated by the U. S. Government in an effort to increase the wartime production of tin. Bolivian milling practice at that time included jigging of the coarse sizes, tabling of the intermediate sizes on conventional shaking tables, and the use of buddles or round tables on the

finest sizes. Flotation of the pyrite was employed at different stages of the treatment, depending upon the quantity present and the preference of the operator. Usually good recoveries and the bulk of the production were made in the jigging and tabling operations provided that these sections were not overloaded. Buddles and round tables were used for a small additional recovery if it could be shown that they could pay for their high cost of operation.

Extensive test work established that by a combination of desliming, classification, and Sullivan deck treatment, cassiterite as fine as 10 microns (1500 mesh) could be recovered when proper conditions were maintained. This test work has been verified by several full-scale installations. The method is generally applicable to the preconcentration of large tonnages of low-grade ores or tailings for the recovery of a small amount of a valuable heavy mineral.

The Sullivan deck (or now the Denver Buckman tilting concentrator), which is the essential part of the method, was devised originally for the recovery of tin from tailings of the Sullivan concentrator, Kimberly, B. C. Since the material treated contained only 0.05 pct Sn, it was impractical to attempt to recover cassiterite finer than 500 mesh.\*

ARVID THUNAES, formerly Special Engineer, Office of Economic Warfare, and Chief Metallurgist, Patiño Mines and Resources, Bolivia, is now Chief, Radioactivity Division, Canadian Dept. of Mines and Resources, Ottawa, Ont. and H. RUSH SPEDDEN, Member AIME, formerly Metallurgist, Office of Economic Warfare, is now Assistant Professor of Mineral Engineering, Massachusetts Institute of Technology, Cambridge, Mass.

AIME Salt Lake City Meeting, August 1950.

TP 2904 B. Discussion (2 copies) may be sent to Transactions AIME before Sept. 29, 1950. Manuscript received Oct. 17, 1949.

\* Trans., Can. Inst. of Min. and Met. (1941) 44, 611-622.

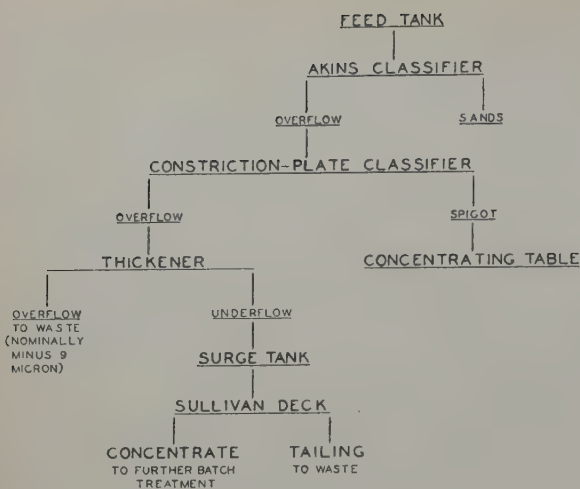


Fig. 1—Flowsheet for pilot plant of 5-ton per day capacity.

The work reported herein is thus an extension of the original application.

### Test Work, Catavi Mill

The largest and most important producer in Bolivia is the Catavi mill of Patiño Mines and Enterprises Consolidated, Inc. Even though milling practice at Catavi was excellent, the large daily tonnage of ore treated and the several million tons of impounded tailings determined that the first work should be done there. The "slime tailing" from this plant represented one half of the total tailing and, since it contained the major portion of the tin loss, appeared to be a promising product for investigation. A sizing analysis and distribution of tin content through the various sizes of this product are given in table I.

Of the material shown in table I, the sizes coarser

Table I. Slime Tailing—Catavi Mill

Size	Pct Wt	Pct Sn	Pct Dist., Sn
+ 48 mesh	1.9	0.25	0.71
65	1.5	0.20	0.45
100	4.2	0.16	1.00
150	7.3	0.18	1.95
200	10.4	0.21	3.28
325	11.6	0.31	5.36
+ 26 micron <sup>a</sup>	2.7	2.46	10.12
18	7.9	1.05	12.44
13	7.0	1.18	12.39
9	7.2	1.13	12.18
— 9	38.3	0.70	40.12
	100.0	0.67	100.0

<sup>a</sup> All subsieve sizing was made by sedimentation in beakers according to the settling rate of cassiterite (sp gr 7.0) as calculated from Stokes' law.

Table II. Size-Assay Analysis of Sullivan Deck Feed

Size	Pct Wt	Pct Sn	Pct Dist., Sn
+ 270 mesh	14.4	0.15	3.4
400 mesh	24.2	0.20	7.6
26 micron	3.9	2.30	13.9
18 micron	14.7	0.90	20.7
13 micron	20.3	0.92	29.2
9 micron	9.8	0.77	11.8
— 9 micron	12.7	0.67	13.4
	100.0	0.64	100.0

than 325 mesh have consistently low tin contents. In lieu of a complete mineralogical study on these products, but, as determined by spot microscopic examinations, we considered that the tin in this range was composed of locked grains.

The four grades from —325 mesh to +9 microns contained 47 pct of the tin content of this product. It was therefore apparent that if gravity concentration could be extended into the fine-size range, a substantial improvement in total tin recovery might be realized.

**Pilot Plant:** A pilot plant of 5-ton per day capacity was constructed for test work on a continuous cut from the slime tailing launder. The flowsheet is given in fig. 1.

The Akins classifier made a separation at about 65 mesh. This coarse sand contained very little free tin but was tabled in a batch operation for checking purposes. The constriction-plate classifier was a laboratory adaptation of one cell of a Fahrenwald classifier and was used to make a separation at 200 to 270 mesh. The concentrating table in the continuous tests was used primarily to provide a check on the control of the classifier.

The thickener was operated as a hydroseparator making a size separation at 9 microns. A portion of the underflow could be returned to the thickener to obtain the required density of feed in the surge tank. The experimental Sullivan deck used in these tests consisted of a hand-tilted deck, 18 in. wide by 6 ft long, with a surface of rubberized riffle cloth (figs. 2 and 3). In operation, the feed was allowed to flow over the deck for a predetermined length of time, usually 5 min. Following this, the feed was shut off, the deck tilted to a steep angle and the rougher concentrate washed into a catch basin. This rougher concentrate was cleaned and recleaned on the Sullivan deck in subsequent batch operations by using only the surge tank and the deck of the circuit. The recleaned concentrate was brought up to final grade by treatment on the laboratory concentrating table, which was a diagonal-deck table with all dimensions one half of those of a standard Deister slime table. Final grade, as used here, means a pyrite-cassiterite gravity concentrate of from 20 to 25 pct Sn. (Flotation of the pyrite would produce a shipping concentrate of better than 51 pct Sn.)

A typical analysis of the feed to the deck is presented in table II.

When treating a feed approximately that as given in table II, the following rates and conditions were used for Sullivan deck operation: pulp flow, U. S. gal per min per ft width, 1.1; pct solids in pulp, 15.5; dry short tons per 24 hr, 1.36; dry short tons per 24

Table III. Size-Assay Analysis of a 51 Pct Sn Concentrate from Sullivan Deck Concentration Followed by Pyrite Flotation

Size	Pct Wt	Pct Sn	Pct Dist., Sn
+ 270 mesh	1.0	5.9	0.1
400 mesh	0.9	31.2	0.6
26 micron	10.8	58.9	12.6
18 micron	38.4	57.4	43.8
13 micron	39.4	47.6	37.2
9 micron	8.8	31.0	5.4
— 9 micron	0.7	19.6	0.3
	100.0	51.0	100.0



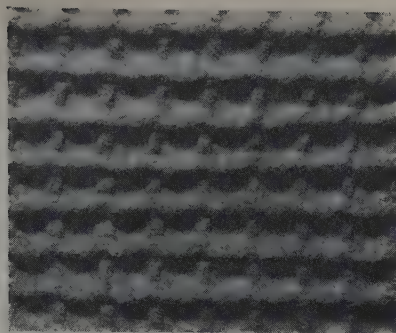


Fig. 2—Rubberized riffle fabric.

hr per ft width, 0.91; tilting cycle, min., 5; and slope, in. per ft, 1.85. Pilot plant tests indicated that 27.2 pct of the tin in the Catavi slime tailing could be recovered. This corresponds to an overall increase in mill recovery of 3 pct.

Table III indicates the size range of cassiterite recovered.

**Full-sized Plant:** A Sullivan deck plant of several hundred tons per day capacity commenced operations at Catavi in 1944. It was designed and constructed on the basis of the results of the above-described pilot plant. The operation proved profitable from the beginning and the construction cost of the plant was repaid in less than one year. Tin was recovered at a cost of 8c to 10c per lb up to at least 1947. Reports indicate that a profitable operation has continued.

The actual recoveries were lower than test work indicated, mainly because less recoverable tin was present in the tailing than in 1943. Recovery from the finest sizes ( $-18+9$  microns) was equal to test results, but recovery from the coarse sizes was less. Plant practice in the coarser range had improved in the meantime.

#### Test Work, Colquiri Mill

The Colquiri mill, operated by Mauricio Hochschild, had a capacity of 600 to 800 tons per day. Mill records showed that from 30 to 40 pct of the total tailing losses occurred in the slime tailing. Size analysis, assays, and distribution of tin content are shown in table IV.

It will be noted that only 10.5 pct of this product was in the  $-13$  micron size range. This is due to the fact that the true slime from the mill (that is, finer than may be recovered by known methods of gravity concentration) was removed without re-treatment as the overflow of two thickeners.

The quantity of sulphur present indicated a much higher content of sulphides than is desirable in the feed to the Sullivan deck. The excess iron beyond that which is accounted for by the sulphur was in the form of siderite, a troublesome problem at this particular plant. To eliminate the sulphides, the sample was first treated by flotation. The flotation non-float was then classified into three sizes which were approximately  $+100$  mesh,  $-100+270$  mesh and  $-270$  mesh. Size analyses of the coarse and intermediate sizes appear in table V.

The intermediate size was treated on an 18 in. by 6 ft laboratory Sullivan deck under the following conditions: slope 2.3 in. per ft, rate of flow 12 liters per min and, pulp density 16 to 20 pct solids. The pulp was fed continuously for 5 min and then a 30-

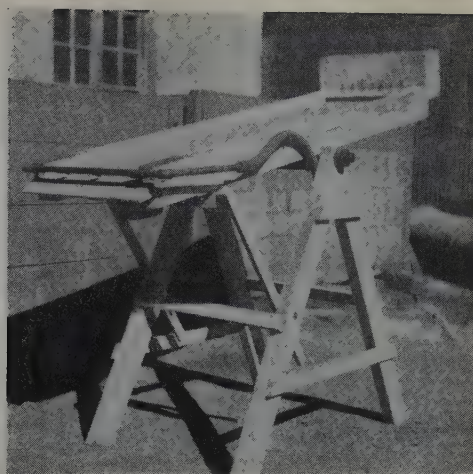


Fig. 3—Experimental Sullivan deck.

sec wash of clear water was used before the table was tilted to wash the concentrate into the catch basin. This Sullivan deck concentrate was further treated on a standard Deister slime table to produce a final concentrate.

The finest size was treated on the Sullivan deck also. The procedure was slightly different, however, in that two stages of Sullivan deck concentration were used and no period of clear water washing was employed. Operating conditions were as follows: slope 1.1 in. per ft; rate of flow, 8 liters per min, and pulp density 15 to 18 pct solids. The cleaner concentrate was taken to final grade on the same Deister table used for the intermediate size. In this concentrate, 95.6 pct of the tin was in the size range

Table IV. Slime Tailing Colquiri Mill

Size	Pet Wt	Pet Fe	Pet S	Pet Sn	Pet Dist., Sn
+ 48 mesh	2.14	27.2	21.4	0.75	1.31
65	4.51	27.0	22.0	0.80	2.96
100	6.96	24.0	19.1	0.50	2.85
150	11.67	21.9	16.1	0.85	8.13
200	18.69	24.2	15.9	0.75	11.48
270	10.59	24.8	15.4	0.95	8.24
400	8.77	25.0	15.2	1.15	8.26
26 micron	7.40	39.0	14.5	3.20	19.40
18	9.93	25.5	13.3	2.20	17.90
13	8.79	19.2	8.7	1.45	10.44
9	2.38	17.4	7.5	1.20	2.34
— 9	8.17	14.3	5.4	1.00	6.69
Composite	100.00	24.1	14.4	1.22	100.00

Table V. Laboratory Classification of Slime Tailing, Colquiri Mill

Size	Coarse Size		Intermediate Size	
	Pet Wt	Pet Sn	Pet Wt	Pet Sn
+ 28 mesh	3.38	0.80		
35	5.31	0.85		
48	9.26	0.75		
65	18.44	0.70		
100	26.17	0.60	4.49	0.50
150	23.72	0.70	18.40	0.40
200	9.06	2.35	33.03	0.45
270	1.49	13.10	15.54	1.20
400			15.48	2.65
—400	3.23	5.75	13.06	6.20
Composite	100.01	1.19	100.00	1.65

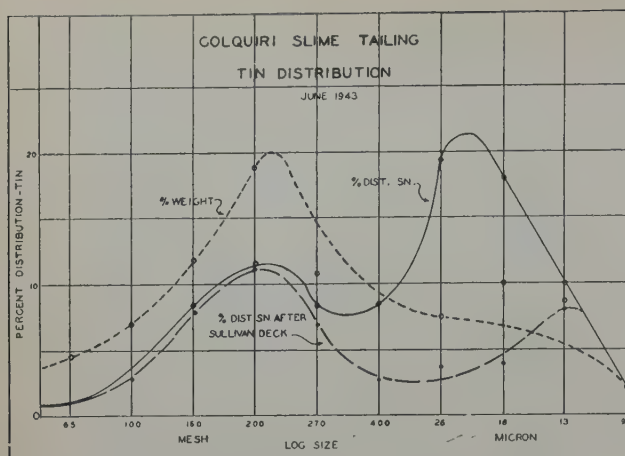


Fig. 4—Colquiri slime tailing tin distribution.

finer than 400 mesh and coarser than 13 microns. Fig. 4 shows the tin distribution in the slime tailing of the Colquiri mill and also what the calculated tin distribution would be after treatment of this product on Sullivan decks. The latter curve was obtained by subtracting the weighted percentage of tin in the pilot plant concentrates from the figures used for the original curve.

The results of the test work on the Colquiri slime tailing are presented in table VI. The total recoveries are shown to be 34.2 pct as a high-grade concentrate and 7.7 pct as a low-grade concentrate. By the use of classification in conjunction with the Sullivan deck it was proved therefore that tin at least as fine as 13 microns could be recovered and also that a high-grade concentrate could be made with a very satisfactory rejection of the trouble-

Table VI. Test Results Colquiri Slime Tailing

	Pct Wt	Pct Sn	Pct. Dist., Sn
Feed to pilot plant	100.0	1.22	100.0
Pyrite float	32.6	0.72	19.4
Classification			
Coarse size	11.4	1.14	10.7
Intermediate size	26.2	1.50	32.2
Fine size	29.8	1.54	37.7
			100.0
Concentration			
Coarse size			
Table concentrate	0.16	26.3	3.2
Intermediate size			
Deck concentrate	4.0	7.4	24.2
Final concentrate	0.32	62.9	16.4
Middling	0.10	32.8	2.6
Fine size			
Deck concentrate	3.0	10.1	24.8
Final concentrate	0.35	60.9	17.8
Middling	0.08	28.6	1.9
Total concentrates	0.67	61.9	34.2
Total middlings	0.34	28.7	7.7
Total tailings	98.99	0.72 (calc.)	58.1

Table VII. Sizing Analysis

Mesh or Microns	Pct Wt	Pct Sn	Pct Dist., Sn
270 mesh	8.42	10.96	2.44
37 microns	16.72	30.14	13.10
18 microns	35.24	38.48	35.95
10 microns	34.77	49.70	45.83
—10 microns	5.35	18.93	2.68
	100.00	37.7	100.00

some mineral, siderite. These results gave promise of an additional total mill recovery of at least 6 pct.

Following the recommendations made as a result of this test work, the company installed a full scale Sullivan deck plant. The improvement in recovery is reported to have been as satisfactory as the pilot plant results predicted.

### Additional Plant Results

Installations were made in a total of 6 tin mills during 1944 to 1946. Test work in 1943 had also shown very good results for two tungsten ores, one containing wolframite, the other scheelite. These latter two plants did not go into production because of the collapse of the tungsten market late in 1943.

Most plants gave recoveries close to those estimated from the test work. Two plants gave better results than predicted. One plant, where good records were kept, treated an extremely difficult ore. The tin occurred in aggregates of very fine cassiterite of near colloidal grain size. The Sullivan Deck section in this plant accounted for 40 pct of the total production. A sizing analysis of the concentrate is given in table VII.

This appears to be the only record of a gravity concentration mill in which nearly half (44 pct) of the total concentrate is finer than 37 microns. The tin ore from this mine was largely in the form of a mud and the most careful grinding could not prevent the production of slimes which were naturally present in the ore body.

### Operating Suggestions

The importance of dispersing the pulp and of desliming prior to fine-size classification and Sullivan deck concentration has been proved in practice. Sodium silicate (up to 1 lb per ton) has been used as a pulp dispersant. Other dispersants may be equally effective and economical.

In the course of the pilot plant test work, it was noticed that the normal pulp waves on the Sullivan deck caused sufficient agitation to lift many of the fine particles that had already settled to the surface of the deck. In an effort to reduce the size of these waves, a surface tension effect, small quantities (0.3 lb per ton) of pine oil were added ahead of the deck. The waves were reduced in magnitude, the grade of the tailing decreased and the grade of the concentrate increased. Such a use of reagents in gravity concentration may be worth investigating further.

During the progress of the pilot plant test work, the Sullivan deck concentrate was graded up to shipping quality by the use of shaking tables. This is admittedly not the best method for minerals in the size range finer than about 18 microns. Other tests have shown that Vanners are more efficient for this final cleaning operation on the very fine sizes.

### Conclusions

Pilot plant tests on Bolivian tin mill tailings proved that gravity concentration could be conducted economically on mineral particles much finer than were treated formerly. The novel features of the method are the combination of desliming, fine-size classification, and the treatment of the classified products on the Sullivan deck. The method has been successfully applied to the treatment of the "slime tailing" from many mills using conventional jig and table circuits.



# Building Stone of the Crab Orchard District, Tennessee

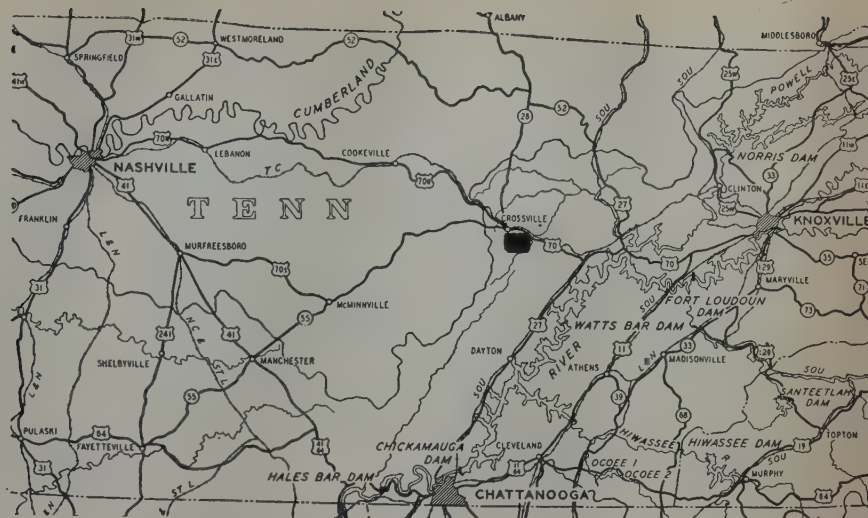


Fig. 1—Index map showing location of principal quarries in the Crab Orchard District.

by Benjamin Gildersleeve

Uniquely colored, thin-bedded quartzite is quarried between Crossville and Crab Orchard in Cumberland County, Tenn. It is produced in all sizes up to the limits of transportation from beds usually ranging in thickness from less than  $\frac{1}{2}$  in. to 6 in. The stone is very resistant to wear and weathering and is marketed throughout the country for many different purposes in exterior and interior construction. The annual production exceeds 36,000 tons, valued at nearly \$1,000,000.

**T**HE building stone of the Crab Orchard District is a fine-grained, thin-bedded sandstone which is so strongly cemented by silica that it may be classed as quartzite. As used in this paper the term building stone includes dimension stone, rubble, ashlar, flagstone, and rough broken stone.

The Crab Orchard District is in the central portion of Cumberland County, Tenn. It is named from the town of Crab Orchard near which the stone was quarried first for industrial use. The heart of the district is an area of some 1200 acres from 1 to 3

miles east of Crossville which is 70 miles west of Knoxville and 125 miles east of Nashville (fig. 1). At the present time, there are about 15 quarries worked by companies and individuals.

**Geology:** The Crab Orchard District is on the Cumberland Plateau, or Cumberland Mountain as it is called frequently. Most of the area is characterized by a comparatively even surface from 1800 to 2000 ft in elevation and is drained by streams flowing into the Tennessee River. Included in the district are two quarries located 7 miles west of Crossville and this area is drained by tributaries of the Cumberland River. Most of the quarries are located near the 1800-ft contour which is just below the general upland surface.

The surface rock over most of the district is the Rockcastle sandstone which has a thickness of about 260 ft.<sup>1</sup> It is the topmost of the six formations of

*BENJAMIN GILDERSLEEVE is Geologist, Tennessee Valley Authority, Knoxville, Tenn.*

*AIME Tampa Meeting, November 1949.*

*TP 2903 H. Discussion (2 copies) may be sent to Transactions AIME before Sept. 29, 1950. Manuscript received Nov. 18, 1949.*



Fig. 2—Slab of stone 15 ft long, 10½ ft wide, 3½ in. thick. Scale at right is 6 ft.

alternating sandstones and shales comprising the Lee group of Pennsylvanian age.<sup>2</sup> The principal quarries are in the Rockcastle, however, the westernmost operations included in the district have been opened in the Bonair sandstone the top of which is between 140 and 200 ft stratigraphically below the Rockcastle.<sup>1</sup> Generally through the region, the Rockcastle is notably cross-bedded, however, this condition is less pronounced in the main quarries. In general, the strata are gently warped into broad synclines and anticlines which trend north-eastward across the district. Locally there are more strongly developed folds and well-defined faults.

**Character and Uses:** The stone is predominantly a multicolored, fine-grained, thin-bedded, quartzitic sandstone. It occurs in beds of remarkably uniform thickness that usually vary from less than ½ in. to 6 in., with several ledges from 6 to 22 in. The quartzite is easily separated along its bedding planes and may be quarried in very large pieces of uniform thickness with smooth surfaces up to handling and transportation limits (fig. 2). The maximum size used is about 16 ft long, 11 ft wide, and 3 in. thick. To date the largest single sheet quarried in one piece was 111 ft long, 8½ ft wide, and 3 in. thick. Of necessity this sheet, weighing approximately 19 tons, was reduced to pieces which could be handled and shipped. Quarried stone can be broken readily without splitting and with equal ease in all planes perpendicular to the bedding. The colors are monotonous of tan, buff, gray, blue gray with various blending shades of yellow, brown, mauve, red, and pink forming stripes, swirls, and an infinite variety of designs, some of which resemble fern leaves and geometrical figures.

The following generalizations are based on laboratory data furnished by the producers: (1) Composition, from 93.54 to 94.87 pct silica; from 3.60 to 4.48 pct alumina and iron; the remainder made up principally of titanium and small amounts of calcium, magnesium, and alkalis. (2) Weight per cu ft, 160.7 to 162.7 lb. (3) Crushing strength, 18,760 to 28,050 psi. (4) Absorption, 0.92 to 0.98 pct.

As a building stone this quartzite is accorded high rank because of its pleasing colors, strength, durability, imperviousness, inertness to acid-laden gases present in the atmosphere of smoky cities, and availability in many sizes, either irregular and rough broken, or shaped to specified dimensions. It is used as dimension stone, rubble, ashlar, flagstone, and rough broken stone for many purposes. In ex-

terior work it is used for roofing; as even-course stone or as broken ashlar for entire walls; for trim on buildings made chiefly of brick, limestone or other materials; for copings, lintels, window sills, treads, terrace floors, paving, garden furniture, retaining walls, memorials, etc. For interior use it is employed as floor tile, wainscoting, trim, decorative panels, fireplace facing, and hearths.

Stone from the Crab Orchard District has been used throughout the country in the erection of structures combining attractiveness and permanence. Many examples could be cited of its architectural use in residences, churches, hospitals, post offices, court houses, offices, and commercial buildings.

**Quarry Methods:** The overburden, consisting of soil and partially disintegrated sandstone, varies from a few inches up to a maximum of about 12 ft, and averages from 4 to 7 ft. It is stripped well ahead of quarrying and is removed by pans, scrapers, and bulldozers. In places, it is necessary to plow, or scarify, the weathered sandstone immediately overlying the sound rock before it is removed.

The quarries are developed laterally by a series of benches and the average overall face is from 7 to



Fig. 3—General view in a large quarry.

9 ft. The largest quarry floor covers approximately 15 acres and the smallest is less than 50 ft square (fig. 3). Between these extremes the production is mainly from quarries averaging 5 to 6 acres in extent.

Although cross-bedding is present in varying degrees of intensity in all the quarries, in general it does not affect quarrying operations adversely. Throughout the district the thinner beds predominate and vary in thickness from less than ½ in. to 6 in., the greater proportion being from 2 to 5 in. Usually they are most prominent in the upper portions of the quarry face, but are not restricted to this position. The next most common thicknesses are from 5 to 10 in., and a few ledges are found up to as much as 22 in. in thickness.

Joints are nearly perpendicular to the bedding, usually trend SE-NW, and are spaced up to 50 ft apart. Many of them are curved and vary as much as 30° (N 50 to 80° W) along their strike. Frequently the joints are off-set and, as seen in the quarry face, may be likened to the vertical divisions in a brick wall. The presence of clay parting seams along the bedding and the partially weathered character of the stone along many of the joints greatly facilitate quarrying.

Quarry methods now employed are quite simple and most of the work is done by hand. Wedges are



driven at short intervals along open seams parallel with the bedding. The stone is then loosened and lifted from the quarry by using crowbars and mat-tocks. To assist in making straight cross breaks in the thinner beds, a gad is struck successive blows with a sledge hammer along the lines of desired parting. In beds 6 in. or more thick, holes are drilled with jackhammers to a depth of 3 or 4 in. at inter-vals of about 10 in. along the desired line of break. Plug-and-feather wedges are placed in the holes and sledged in succession beginning at one end of the line (fig. 4). Although the steel bits are dulled rather quickly by the very abrasive stone, their use has been found less expensive than those tipped with tungsten carbide. One operator reports that the cost of drilling with tungsten carbide bits is approximately one and one half times that of steel bits.

Quarry waste is handled with Brooks load luggers, and other trucks are used to transport the quarried rock to stockpiles and to shipping points. The larger pieces that are too heavy to be loaded by hand labor are moved by trucks equipped with cranes. Hand trucks are also used to handle material in the quar-ries.



Fig. 4—Plug-and-feather wedges are used in quarrying the thicker beds.

The quarries have almost automatic drainage be-cause they are shallow and located near the general upland surface. Little pumping is necessary except to remove surface water in times of heavy rains.

**Fabrication:** The most revolutionary development in the district was the introduction in 1947 of guillo-tine machines for cutting the stone. These machines are operated by a hydraulic pressure system and have capacities up to 12x48 in. between the cutting blades (fig. 5). At the present time, there are four guillotines of this type in the district. In addition, one producer has built a similar hydraulic machine with two sets of 26 teeth each instead of opposing blades, and having a cutting surface of 60 in. Stone is moved to and from the machines over roller conveyors.

Although the guillotines have made possible in-creased production and profits, hand cutting is still a major operation in the district. Hammers, points, chisels, pitching tools, and tracers are the hand tools most commonly used. The stone is supported on strong work benches at a height convenient for working. As previously mentioned, the stone breaks easily in all planes perpendicular to the bedding, consequently, pieces can be trimmed either in rec-tangular or in circular shapes. The pieces are marked and broken in a manner similar to that used in cut-



Fig. 5—Guillotine machines for cutting the stone.

ting plate glass. After a tracer tool has been sledged along the desired trim line, a sharp blow will cause the stone to break without fracturing. The thicker ledges, when cut in this fashion, may have somewhat irregular faces which are roughened down to com-paratively uniform surfaces with pitching tools and hammers. These hand tools are also used to rough down blocks of rubble stone cut with guillotines.

The stone does not take a very high polish and it is so abrasive that machine work is both difficult and expensive. Consequently, only a limited amount of sawing, rubbing, and sand blasting is done. Both diamond and carborundum saws are used.

**Production and Marketing:** It is estimated that approximately 450 men are employed in the indus-try, and that the annual production is over 36,000 tons, valued at nearly \$1,000,000. Over 90 pct of the output in the district is produced by three operators, namely: Crab Orchard Stone Co., Inc., Tennessee Stone Co., and Turner Brothers Stone Co., Inc.

Markets for the building stone of the Crab Orchard District are country-wide. It is sold under the com-mercial names of: Crab Orchard Stone, Tennessee Quartzite, Tennessee Variegated Stone, Cumberland Mountain Stone, Chromastone, and Tennessee Ledge Rock. The stone is sold according to thickness and general quality. Prices for rubble and other rough broken stone range from \$5.00 to \$35.00 a ton. Flags, trim, sills, coping, and other pieces with straight broken edges, or cut to specifications, bring special prices according to thickness and vary from \$0.25 to \$3.50 a sq ft.

#### Acknowledgments

The cordial interest and cooperation of the quarry operators and property owners is greatly appre-ciated. Thanks are expressly given to E. Rohde and H. A. Schubert, Tennessee Stone Co., for assistance in making a reconnaissance survey of the principal quarries and for furnishing data on quarry methods and marketing; to R. N. Pelot, Crab Orchard Stone Co., Inc., for information on quarrying, fabrication, and other aspects of the industry; to the Turner Brothers Stone Co., Inc., for data on their quarry and the district; and to residents in the district for information about old quarries and the growth of the industry.

#### References

- <sup>1</sup>C. Butts and W. A. Nelson: *Geology and Mineral Resources of the Crossville Quadrangle, Tennessee*. Tenn. Dept. Educ., Div. Geol. *Bull.* 33-D (1925).
- <sup>2</sup>M. G. Wilmarth: *Lexicon of Geologic Names of the United States (including Alaska)*. U. S. Geol. Survey. *Bull.* 896 (1938).

# Fluoride in Ground Water of Alabama

by Philip E. LaMoreaux

**Fluoride, generally less than 0.5 ppm, is present in ground water from rocks of Paleozoic age and older, in northern and eastern Alabama. Some of the water-bearing formations in the Coastal Plain area of the State yield water with as much as 6.8 ppm fluoride.**

IN June 1940, the U. S. Geological Survey, in co-operation with the Geological Survey of Alabama, began a study of the ground-water resources of the part of Alabama where water is obtained from Cretaceous rocks. The purpose of the study was to determine the quality, quantity, occurrence, and availability of ground water in that area.

These studies have been expanded to include ground-water investigations in the area of Tertiary rocks, or southern quarter of the State, and certain areas in northern Alabama.

The first report issued on these studies was by C. W. Carlston,<sup>1</sup> of the Alabama Geological Survey. In the present paper all references to ground-water data for the Cretaceous area are taken from this earlier publication.

In 1945, the Dentistry Division of the Alabama Department of Public Health became interested in the correlation of tooth decay and mottled enamel with the chemical quality of ground water used for public supplies in the State. Through a cooperative arrangement between the State Department of Public Health and the Ground Water Division (now Branch) of the U. S. Geological Survey, a report by the author<sup>2</sup> was published in 1948. These two reports give accurate information on the occurrence of fluoride in the Coastal Plain of Alabama.

At present, only scattered information is available on fluoride in ground water of the crystalline-rock or Piedmont area and for the Paleozoic area of

Alabama, but it is hoped that in the future more complete information on the presence of fluoride in ground water in these areas can be collected.

**Outline of Geology and Ground Water:** As described by Adams,<sup>3</sup> the State of Alabama includes parts of two major geologic divisions, the Appalachian region and the Coastal Plain. The boundary between these divisions is irregular and is known as the Fall Line of the Atlantic and Gulf Coast States. The Fall Line enters Alabama near Phoenix City, extends westward to Wetumpka, Clanton, and Tuscaloosa, and then swings northwestward to the northwest corner of the State.

The Appalachian region in Alabama includes three major provinces, the Piedmont province, the Appalachian Ridge and Valley province, and the Appalachian Plateau province (fig. 1).

The rock formations in the Piedmont province in east-central Alabama are mainly of pre-Cambrian age, chiefly crystalline schists and gneisses injected by younger igneous rocks (fig. 1, I). They are faulted and folded and have a complicated structure. These rocks are the oldest and among the most complex rocks in the State. Generally, only small yields of ground water are obtained from rocks in this area. Even though ground water is of great importance for the development of domestic and farm supplies in rural areas and a few small industrial and municipal wells, yields in the area from individual wells generally range from 5 to 25 gpm and rarely exceed 50 gpm.

The geologic formations of the Appalachian Ridge and Valley province and the Appalachian Plateau province (see fig. 1, II, and III) are separated on the basis of their structure. These rocks are of Paleozoic age, ranging from Cambrian to Carboniferous, and comprise a succession of formations consisting chiefly of shale, sandstone, limestone, and dolomite, aggregating many thousands of feet in

PHILIP E. LAMOREAUX is Geologist in charge of Ground Water Investigations in Alabama, Ground Water Branch, U. S. Geological Survey, University, Ala.

Paper published by permission of the Director, U. S. Geological Survey.

AIME Tampa Meeting, November 1949.

TP 2883 H. Discussion (2 copies) may be sent to Transactions AIME before Sept. 29, 1950. Manuscript received Oct. 24, 1949.



thickness. The limestones and dolomites and some of the sandstones in this area yield large quantities of ground water for industrial and municipal supplies, and smaller ground-water installations supply the entire rural area.

The rocks of the Alabama Coastal Plain area in the southern and western parts of the State are divided into two major age groups, the Cretaceous and Tertiary (fig. 1, IV A and B). These Cretaceous and Tertiary formations dip south toward the Gulf of Mexico in eastern and central Alabama and southwest toward the Mississippi Embayment in western Alabama at low angles, from 20 to 50 ft per mile. These formations consist primarily of unconsolidated silt, clay, sand, gravel, and limestone, and exceed 5000 ft in total thickness in southwest Alabama. Ground water developed from wells and springs is of great importance to the Coastal Plain area of the State. Nearly every city and water-consuming industry depends on ground water. In the rural areas nearly every farm has its own well or spring.

**Summary of Results of Dental Investigations:** Knowledge of the influence of fluoride on tooth enamel and tooth decay is by no means new, for Crichton-Browne<sup>4</sup> in England in 1892 speculated on the correlation of dental caries or tooth decay with fluorine in the diet. Much additional work, especially during the last 20 years in the United States, has been carried forward on the studies of the relation of the amount of tooth decay and mottling of tooth enamel to the amount of fluoride in drinking water. A general summary of this work in the United States shows fluoride to have two effects on the development of teeth, (1) mottled enamel (dental fluorosis) from too much fluoride in drinking water, and (2) reduction of tooth decay (dental caries) from the presence of smaller quantities of fluoride.

H. Trendley Dean<sup>5</sup>, dental surgeon of the U. S. Public Health Service, points out that the "harmful factor" or mottled enamel is on the permanent teeth and results from drinking water that contains toxic quantities\* of fluorine during the time of development of the permanent teeth. In other words, only

\* More than 1 ppm according to the 1942 paper; over 1.5 ppm according to present standards of the Public Health Service.

children under 8 yrs of age are susceptible to this dental defect. Fortunately, there are beneficial effects of water containing small amounts of fluoride. Dean<sup>5</sup> has summarized this aspect as follows:

An inspection of the range of dental caries experience associated with the use of domestic water of different fluoride concentration discloses an inverse relation in general between the amount of dental caries and the fluoride concentration of the common water supply. Relatively low dental caries experience rates are found associated with the use of domestic waters whose fluoride (F) concentrations have a range of 1 or more parts per million. Immediately, e.g., at concentrations of 0.9 to 0.5 part per million, the influence is less marked than at the higher concentrations; nevertheless, the dental caries experience rates are distinctly lower than those associated with the use of relatively fluoride-free waters.

**Fluoride in Ground Water of Alabama:** Few chemical analyses have been made on samples of water collected from the crystalline-rock area of the State. The following information, therefore, necessarily is highly generalized. For the most part, water from wells in the crystalline-rock area is fairly low

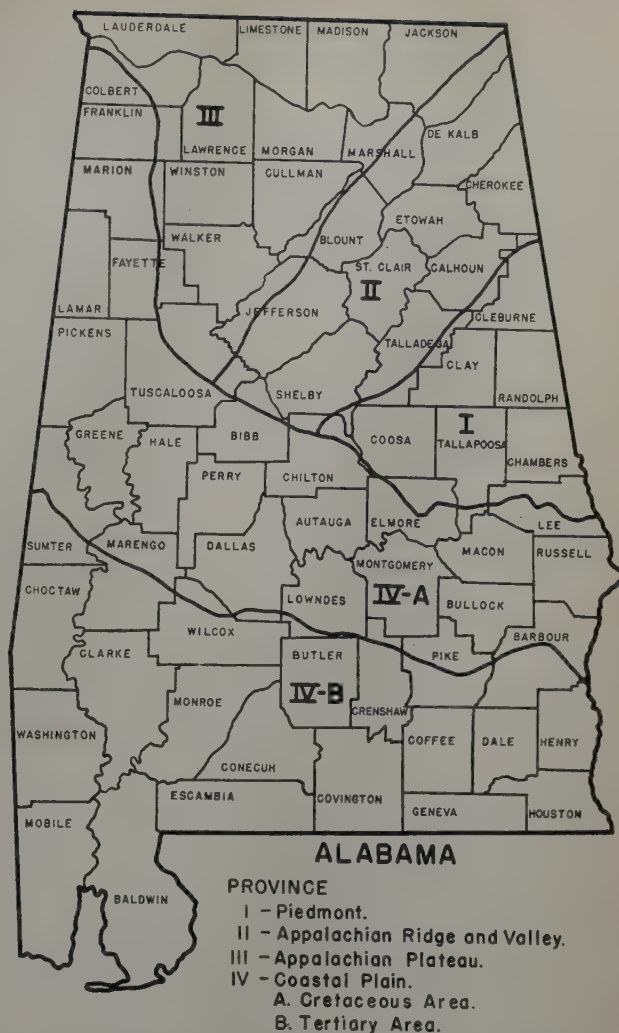


Fig. 1—Outline of Alabama showing geologic provinces.

in dissolved solids and no samples of water from this area showed toxic quantities of fluoride. In general, the fluoride content of samples of water from wells in the crystalline rocks ranged from a trace to 0.5 ppm. Some samples contained no fluoride.

Though the information for the Paleozoic area of northern Alabama is somewhat better than that for the Piedmont area, the number of chemical analyses is not sufficient to give an accurate report on fluoride in ground water in the area. In general, water from sandstone and limestone in the Paleozoic rocks of northern Alabama contains from a trace to 0.2 ppm of fluoride. One exception to this was a well drilled in the Warsaw limestone near Huntsville, Madison County, Ala. This well yielded water containing 3.6 ppm of fluoride from a solution cavity in limestone.

Many of the analyses of water from the Coastal Plain deposits of Alabama show the presence of fluoride. It is interesting to note, from a bar graph prepared by the Dentistry Division of the State Department of Health during a preliminary dental survey of the State, that the counties with the largest percentage of children having nearly perfect teeth are counties in which analyses of well water show appreciable amounts of fluoride.

The deposits of the Coastal Plain have been divided into two groups, the Cretaceous and Tertiary, according to geologic age. The principal water-



bearing beds of the Upper Cretaceous deposits in Alabama include the sands of the Tuscaloosa group and the Eutaw, Blufftown, and Ripley formations and the Cusseta and Providence sands. Of these, the glauconitic sand beds of the Eutaw, Blufftown, and Ripley formations yield water containing up to 6.8 ppm of fluoride. The heaviest concentrations of fluoride (6.8 ppm) in Alabama were found in water from wells penetrating the Eutaw formation at Letohatchie in Lowndes County.

Nearly every Tertiary formation has a sandy or limestone facies that is a potential source of ground water. Many municipal and industrial wells recorded in the ground-water study of the Tertiary area were developed in the sands of the Nanafalia formation, Tuscaloosa and Gosport sands, and Lisbon formation, from solution channels in the Ocala limestone, and from sands in the Miocene beds and the Citronelle formation.

Of the representative samples of water from wells tapping these formations only two had a fluoride content of 1 ppm or more. One well yields water from the Nanafalia formation at Thomasville in Clarke County, the other well yields water from the Lisbon formation at Gilbertown, Choctaw County. In general, the fluoride content of the samples of water from the Tertiary formations was 0.5 ppm or less.

**Source of Fluoride in Ground Water:** Relatively little information is available on the amount and distribution of fluoride in rocks, particularly in sedimentary rocks, because until recent years accurate methods of determining small amounts of fluoride in the analysis of rocks have not been available. As the amount of fluoride in ground water differs from place to place, it appears that a direct relation exists between the composition of sedimentary material and the amount of fluoride present in water from that sediment. Several theories have been advanced as to the source of fluoride. In some regions fluoride may be traced directly to magmatic sources, to vapors in the atmosphere from a volcanic vent, or to volcanic ash and tuffaceous materials in sediments.<sup>6</sup> Another possibility may be that fluoride present in sea water during the original period of deposition of a marine bed has not been flushed out completely. Fluoride may also be furnished by weathering or solution of some of the more common fluorine-bearing minerals, such as apatite, tourmaline, topaz, vesuvianite, lepidolite, or glauconite, in the rocks through which ground water percolates. If these fluorine-bearing minerals are the source, they would have to be affected by some erosional process that would release the fluoride in soluble form.

Gwynne<sup>7</sup> suggests that the breaking down by weathering processes of the more insoluble fluorine-bearing minerals might be accomplished by the action of sulphuric acid formed by the decomposition of pyrite contained in a sedimentary rock. Van Burkalow<sup>8</sup> theorizes that fluoride in soluble form should be expected where two conditions prevail: (1) an abundance of fluorine-bearing minerals, and (2) an abundance of pyrite in association with concentrated organic material that would facilitate the decomposition of the fluorine-bearing minerals. To check these theories, lithologic data were compiled on the water-bearing formations of the Coastal Plain of Alabama. The presence of phosphates, bentonite, pyrite, lignite, glauconite, and other fluoride-bearing minerals was checked against the occurrence of fluoride in the water. It was found

that high fluoride content in ground water from formations in the Alabama Coastal Plain area is associated with the presence of abundant phosphatic material, pyrite, lignite, and glauconite. In addition to this information, it was discovered that, in general, the deeper wells penetrating water-bearing sands containing these minerals and yielding water high in fluoride had greater concentrations of fluoride than the water from shallower wells penetrating the same sands up the dip. This indicates the possibility that the amount of fluoride in the water may be related not only to the presence of certain minerals in deposits but also to the depth to the formation and the distance to the outcrop, and therefore, to the extent of ground-water circulation and the length of time the water has been present in the deposits. It is also possible that greater pressure and temperature at greater depths promote the reaction.

**Summary:** The water from the glauconitic sands in the Eutaw, Blufftown, and Ripley formations of the Coastal Plain deposits contains larger amounts of fluoride than does that of any other formation in the State, as far as known. Samples of water from wells in the Eutaw and Ripley formations used for public supplies in the central Coastal Plain area, including Greene, Dallas, Montgomery, Marengo, Wilcox, Lowndes, Barbour, and Butler counties, contain from 1 to 6.8 ppm of fluoride. The fluoride content of samples of water from wells in the Tertiary area or southern quarter of the State ranged from a trace to 1.2 ppm.

To date, only a limited number of samples have been collected in the Piedmont crystalline-rock area of Alabama and Paleozoic area of northern Alabama. The fluoride content of 10 of these samples, with one exception, was zero. One sample contained 3.6 ppm of fluoride. It is possible that, with more detailed studies, higher concentrations of fluoride will be found, although it is believed that in general the fluoride in ground water will not exceed 1 ppm in this area.

In areas where high fluoride concentrations occur, it is usually possible to obtain water free from fluoride by drilling to overlying or underlying formations. Therefore, after more complete information is available on the effects of fluoride, it may be possible to combine fluoride-free water with fluoride-bearing water to obtain the optimum amount of fluoride necessary for health.

## References

- <sup>1</sup> C. W. Carlston: Fluoride in the Ground Water of the Cretaceous Area of Alabama. *Ala. Geol. Survey. Bull.* 52 (1942).
- <sup>2</sup> P. E. LaMoreaux: Fluoride in the Ground Water of the Tertiary Area of Alabama. *Ala. Geol. Survey. Bull.* 59 (1948).
- <sup>3</sup> Charles Butts, L. W. Stephenson, C. W. Cooke, and G. I. Adams: *Geology of Alabama*. Ala. Geol. Survey. Special Rept. 14, 25, (1926).
- <sup>4</sup> Crichton-Browne: *Jnl. Lancet.* (1892) 2, 6.
- <sup>5</sup> H. T. Dean, F. A. Arnold, Elias Elvove, D. C. Johnston, and E. N. Short: Domestic Water and Dental Caries. *Public Health Repts.* (Aug. 1942) 57, No. 32, 1176-1177.
- <sup>6</sup> G. R. Mansfield: The Role of Fluorine in Phosphate Deposition. *Amer. Jnl. Sci.*, (1940) 238, 863-879.
- <sup>7</sup> C. S. Gwynne: Geological Significance of Fluorine in Iowa Well-waters. *Pan-Amer. Geol.* (1934) 62, 139-140.
- <sup>8</sup> Anastasia Van Burkalow: Fluorine in United States Water Supplies. *Geog. Rev.* (1946) 36, No. 2, 187-188.



# An Oxidation Method

## For Investigating the Petrographic Composition of Some Coals

by Reynold Q. Shotts

Data are presented which show that fractions of varying densities from the same coals are oxidized at different rates by nitric acid. From oxidation data, the approximate quantity of "bright" and "dull" components may be calculated. Definite relationships between oxidation rate and rank are shown for density fractions from at least two bituminous coals.

IN 1945, the Alabama State Mine Experiment Station initiated a program on the determination of fusain in certain coals of the state.<sup>1</sup> Because of the lack of equipment and personnel necessary for microscopic examination, the chemical method<sup>2</sup> was used exclusively. During studies of the distribution of fusain in fractions of various sizes and densities on a medium volatile coal, the author noted a higher rate of oxidation of the nonfusain part of the coal in the lower density fractions and the difference in rate became more marked the shorter the oxidation period used. Since durain and other "dull" components of coal have been reported<sup>3, 4, 5</sup> to have higher densities than vitrain and the bright components, it appeared probable that the observed differences in rates of oxidation in the different density fractions were due to variations in petrographic composition. Such effects have been noticed by other workers.<sup>2, 3</sup>

In order to obtain further evidence on this point, several Alabama coals of different rank have been separated into three or more fractions on the basis of density and their rates of oxidation determined, using the procedure specified for the determination of fusain, except that measurements were made at short time intervals as well as for the longer periods specified in the regular procedure. Significant differences in the rates of oxidation of the nonfusain part of the various fractions have been observed, and it is believed that studies of this type may make

possible the identification of individual petrographic components or lead to methods for their semiquantitative estimation.

**Properties of the Coals:** Analyses and some properties of the whole coals studied are given in table I. The range in rank of the coals is not the widest possible within the state, but samples of the low volatile coals that occur in thin beds in the northeastern part of the state, were unavailable.

The Clements bed is not, at present, considered workable. The coal is of medium volatile rank, quite friable, and very strongly swelling. Analyses of different size fractions show that the larger sizes are of higher rank than the smaller ones. Material passing the No. 50 sieve (all sieve sizes refer to the U. S. Standard Series) is close to high volatile A in rank, except for the size passing a No. 200 sieve, in which fusain is partially concentrated. Because of the variation in rank of the sizes of Clements coal, oxidation studies were made of two widely different size ranges. The + 1-in. material was crushed on rolls to about  $\frac{1}{8}$  in. and separated into five fractions by float-and-sink treatment in a benzene-carbon tetrachloride mixture. The — No. 4 + No. 8 and the — No. 4 + No. 50 size ranges tested overlap but the latter size was studied last with the expectation that it would include more oxidizable material than the first size. This proved not to be the case. Yields and properties of the specific gravity fractions oxidized are shown in table II.

Milldale bed coal probably represents, as nearly as it is possible to do so, the "average" coal of the state. The material passing a No. 4 sieve was the only size range oxidized. It was divided into four specific gravity fractions. As in every case the heaviest fraction, which was essentially rock, was discarded.

The Black Creek is the most extensive in area of

---

REYNOLD Q. SHOTTS, Member AIME, is Associate Professor of Fuel Engineering, University of Alabama, University, Ala.

AIME St. Louis Meeting, February 1951.

TP 2867 F. Discussion (2 copies) may be sent to Transactions AIME before Sept. 29, 1950. Manuscript received Oct. 31, 1949; revision received March 6, 1950.

Table I. Properties of the Whole Coals

Bed	Analyses						Fusain <sup>c</sup>			
	M	A	VM	FC	S	Btu	Content	FSI	Rank	Condition
Clements	<sup>a</sup> 1.1 <sup>b</sup>	8.4	25.9 27.5	64.6 72.5	2.1	14,150 15,840	3.3 <sup>d</sup>	9+	Med. vol. Bit.	Raw-mine run
Milldale	<sup>a</sup> 1.8 <sup>b</sup>	8.9	32.5 35.6	56.8 64.4	1.2	13,830 15,660	7.7	7½	High vol. Bit. A	Raw-mine run
Black Creek <sup>e</sup>	<sup>a</sup> 2.6 <sup>b</sup>	3.9	36.4 38.4	57.1 61.6	1.5	13,910 14,980	<sup>f</sup>	1½	High vol. Bit. A	Washed slack <sup>g</sup>
Clarke County Lignite	<sup>a</sup> 12.6 <sup>b</sup>	5.8	41.1 49.5	40.5 50.5	3.7	8,950 11,080	1.3	NA <sup>h</sup>	Lignite <sup>i</sup>	Raw

<sup>a</sup> Air dry basis.<sup>b</sup> Dry, mineral-matter free (Parr) basis.<sup>c</sup> Moisture and ash-free basis.<sup>d</sup> Slack passing a 1¼ in. square opening.<sup>e</sup> Black Creek group. Bed designated Jefferson in late Bureau of Mines reports.<sup>f</sup> Not determined. Black Creek Bed known to contain considerable fusain.<sup>g</sup> Passing a 1¼ in. square opening. A light water spray over a No. 30 wedge wire screen, was observed to remove a little material.<sup>h</sup> Non-agglomerating.<sup>i</sup> "Bed" moisture unknown. See ref. 6.

the coal beds of the state. In the northwestern part of the Warrior field it is the lowest in rank of any of the coals mined in Alabama. Only the — No. 4 + No. 50 sieve size was subjected to oxidation studies. The material passing a No. 50 sieve was very high in fusain (samples No. 118, 119, table II) and was removed to facilitate the float-and-sink operation.

Thin beds of lignite occur in the upper Cretaceous and Tertiary formations in many places in Alabama, the most extensive occurrences being in the Wilcox Group of Tertiary Age.<sup>6</sup> A small sample was obtained from a 14-in. bed in a new highway cut. Nitric acid oxidations and analyses were not made until after more than a year of storage in the laboratory. No determinations of "bed" moisture could be

made but there is no doubt that the sample was lignite.

In addition to the lignite, samples of two sizes of raw Clements coal (samples No. 63, 124) and two high fusain, raw coal samples from the Black Creek bed (samples No. 118, 119) were oxidized.

**Principles of Procedure:** Briefly, the method proposed by Fuchs and associates for the determination of fusain<sup>2</sup> consists of the oxidation of 1-g samples of coal in boiling 8 N nitric acid in a flask fitted with a reflux condenser, for periods of 2, 3, and 4 hr. The unoxidized residue is carefully filtered from the liquid and washed. The residue is transferred to a beaker, treated with 40 ml of normal sodium hydroxide, diluted to 900 ml, and allowed to set overnight. The brown, supernatant liquid is

Table II. Properties of Coal Fractions

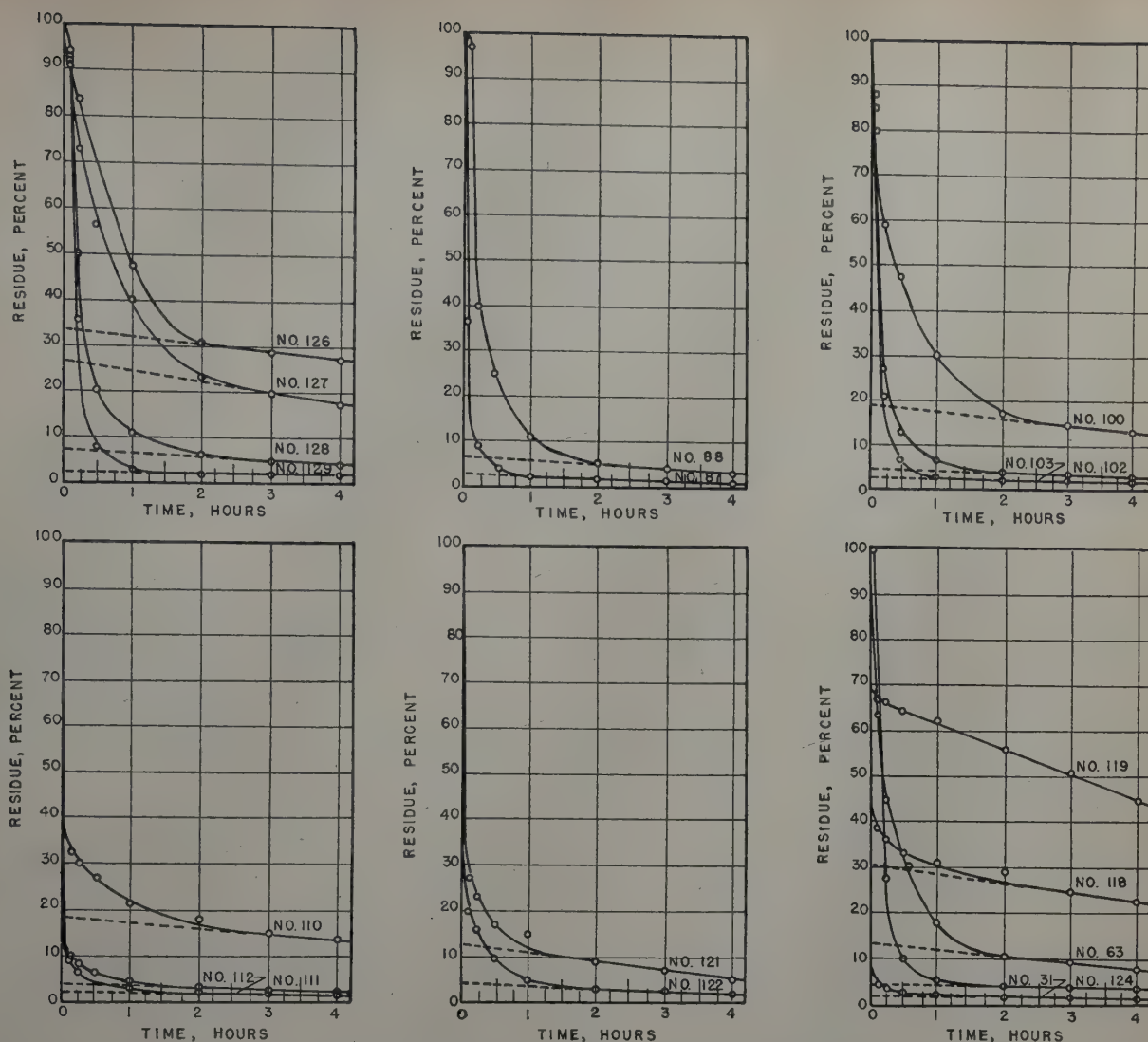
Sample No.	Bed and Size	Specific Gravity Limits	Yield, Pct	Proximate Analysis <sup>a</sup>			Fusain <sup>b</sup>	FSI <sup>b</sup>
				FC	VM	Btu		
125	Clements + 1 in.	Sink-1.800	5.5					NA <sup>d</sup>
126	Clements + 1 in.	1.397-1.800	22.6	77.7	22.3	15,120	33.0	1
127	Clements + 1 in.	1.310-1.397	22.1	76.7	23.3	15,750	26.0	4½
128	Clements + 1 in.	1.280-1.310	12.7	73.0	27.0	15,640	6.4	10
129	Clements + 1 in.	Float-1.280	37.1	71.9	28.1	15,520	1.4	10¼
88	Clements No. 4xNo. 8	1.28-1.60	31.7	73.7	26.3	15,670	5.2	9¾
87	Clements No. 4xNo. 8	Float-1.28	65.8	71.9	28.1	15,680	0.8	10
99	Clements No. 4xNo. 50	Sink-1.60	5.7					NA
100	Clements No. 4xNo. 50	1.35-1.60	4.5	75.2	24.8	15,360	18.0	2
102	Clements No. 4xNo. 50	1.27-1.35	21.8	71.7	28.3	15,370	3.1	9¾
104	Clements No. 4xNo. 50	1.26-1.27	27.1	72.2	27.8	15,650		10
103	Clements No. 4xNo. 50	1.25-1.26	38.7	72.3	27.7	15,620	0.9	10¼
101	Clements No. 4xNo. 50	Float-1.25	2.2	71.2	28.8	15,420		10
109	Milldale No. 4x0	Sink-1.60	17.8					NA
110	Milldale No. 4x0	1.32-1.60	18.5	68.1	31.9	15,020	17.5	3¾
111	Milldale No. 4x0	1.283-1.32	38.5	66.6	33.4	15,200	3.1	7
112	Milldale No. 4x0	Float-1.283	25.2	64.2	35.8	15,000	2.0	7½
120	Black Creek No. 4xNo. 50	Sink-1.510	4.8					NA
121	Black Creek No. 4xNo. 50	1.305-1.510	28.1	65.1	34.9	14,400	11.7	1
123	Black Creek No. 4xNo. 50	1.283-1.305	31.1	61.4	38.6	14,520		2
122	Black Creek No. 4xNo. 50	Float-1.283	36.0	55.9	44.1	14,530	3.3	2
63	Clements + 1 in.	Raw		72.9	27.1	15,660	13.3	8½
124	Clements No. 50x0	Raw		71.2	28.8	15,250	4.2	
119	Black Creek — No. 200	Raw		76.7	23.3	14,880	67.0	1 <sup>e</sup>
118	Black Creek No. 50xNo. 200	Raw		68.2	31.8	14,640	30.5	NA
31	Lignite	Raw		50.5	49.5	11,080	1.3	NA

<sup>a</sup> Dry, mineral matter-free (Parr) basis.<sup>b</sup> Dry, ash-free basis.<sup>c</sup> Free swelling indexes larger than 9 determined by area measurement.

(See U.S. Bur. Mines, R.I. No. 4238).

<sup>d</sup> NA—non-agglomerating.<sup>e</sup> Button coherent but could not be handled.





Figs. 1-6—Percentage of residue plotted against time.

Fig. 1. (Upper left)—Clements bed coal, +1-in. size. Fig. 2 (Upper center)—Clements bed coal, —No. 4+No. 8 size. Fig. 3 (Upper right)—Clements bed coal, —No. 4+No. 50 size. Fig. 4 (Lower left)—Milldale bed coal, —No. 4 size. Fig. 5 (Lower center)—Black Creek bed coal, —No. 4+No. 50 size. Fig. 6 (Lower right)—Various sizes of raw bituminous coal and one raw lignite.

removed, the residue filtered, dried, weighed, ignited, and weighed again. The ash-free residue is expressed as a percent of original dry, ash-free coal. The percent residue is plotted against time, using linear coordinates. As a rule, the three points fall along a straight line. The extrapolation of this line to zero time gives the percentage of dry, ash-free fusain present in the original sample. In some cases the 2 hr oxidation may prove insufficient for the solution of all nonfusain material. In such a case, the 3 and 4 hr oxidations are connected with a straight line and extrapolated to zero time.

The method and procedure proposed by Fuchs and associates was followed as closely as was possible under the circumstances. Much trouble was experienced in the case of some coals, by the tendency of the fine material to "crawl" up the sides of the boiling flask. For this reason, most of the determinations were run on — No. 100 material rather than — No. 200 coal as specified by Fuchs. The flattening of the curves after 2 hr of oxidation, figs. 1 to 6, shows little evidence of a "dragging out" of the oxidation of the nonfusain material due to large particle size.

Fuchs and associates explain the shape of oxidation curves like figs. 1 to 6 by assuming them to be compound, or to represent two distinctly different

types of reaction. They state that the first part of the curve represents an apparent first-order reaction in which the reaction rate changes with the

Table III. Values of  $K_0$  at Various Times

Sample No.	Time, Hr				
	0.125	0.25	0.50	1.0	2.0
126	1.05	1.11	0.94	1.36	2.28
127	0.87	1.81	1.73	1.51	1.86
128	0.66	3.01	3.86	2.93	2.62
129	0.52	4.27	5.60	4.50	3.10
88	0.42 <sup>a</sup>	4.13	3.18	2.81	2.74
87	12.58 <sup>a</sup>	10.56	7.36	5.52	3.11
100	2.28	2.80	2.06	1.80	1.38
102	1.07	5.75	4.79	3.51	2.75
103	1.34	6.57	6.19	5.29	3.45
110	13.76	7.40	4.31	2.89	1.86
111	21.74	12.28	7.16	4.39	2.89
112	21.85	13.01	8.00	4.70	2.75
121	14.24	8.26	5.07	3.00	3.39
122	14.48	8.37	5.70	4.17	
63	0.05	4.06	3.11	2.67	
124	3.89	5.68	5.75	4.38	3.43
119	20.0+				
118	17.1	9.87	5.98	3.33	1.70
31	28.8	16.02	9.18	5.10	3.45

<sup>a</sup> 0.083 hr oxidation.

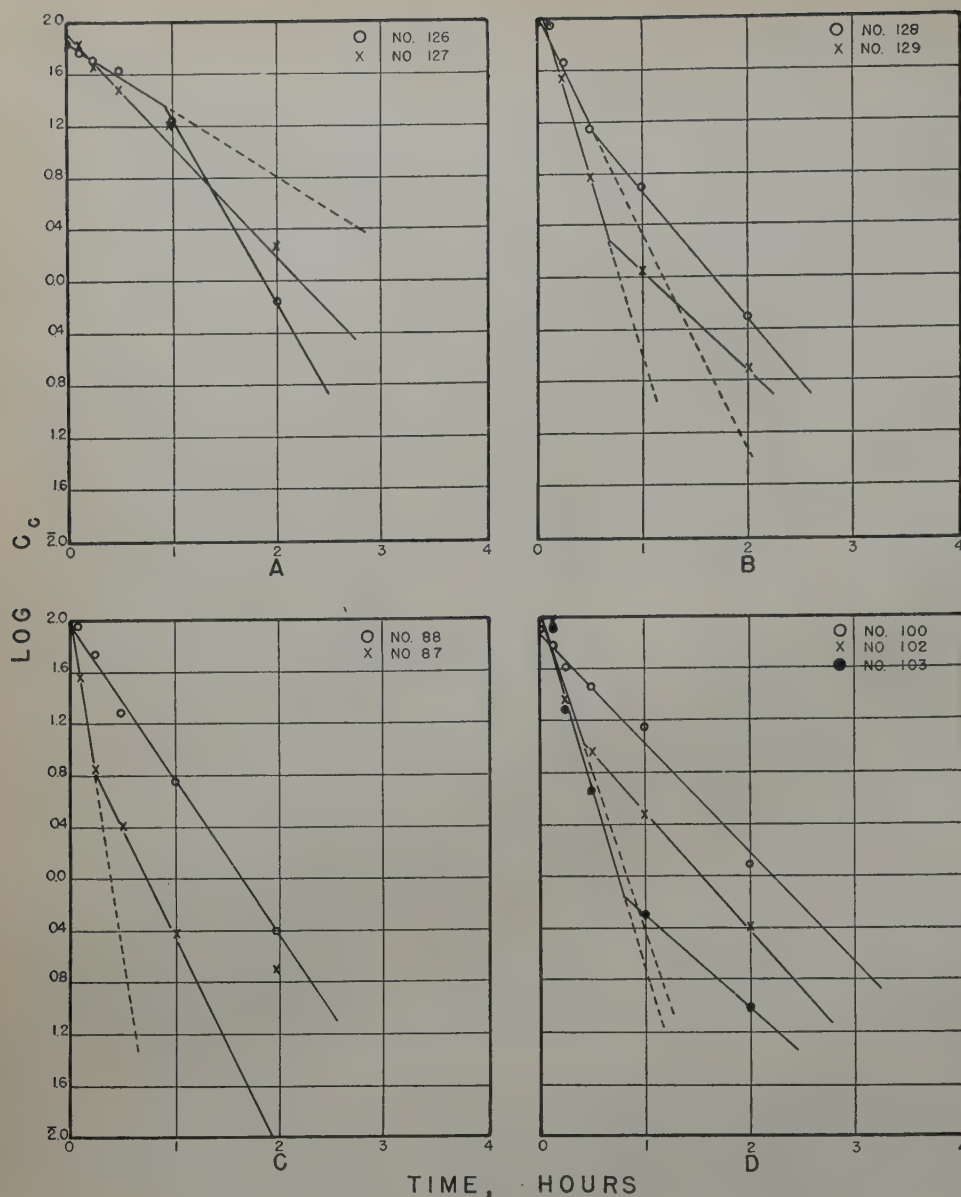


Fig. 7—Logarithm of concentrations of “coal” plotted against time of oxidation.

(A) Two light specific gravity fractions of +1-in. Clements bed coal. (B) Two heavy specific gravity fractions of +1-in. Clements bed coal. (C) Two specific gravity fractions of —No. 4, +No. 8 Clements bed coal. (D) Three specific gravity fractions of —No. 4 + No. 50 Clements bed coal.

concentration and the second, or straight portion, reflects a zero-order reaction in which the rate is independent of the concentration. Early in the process, the first-order oxidation of “coal” will predominate and the overall rate will be governed by it. When all of the “coal” is oxidized, the oxidation of the remaining material, which is fusain, will follow the zero order law.

The reaction rate for the coal fraction then will be:

$$\frac{-dC_c}{dt} = K_c C_c \quad [1]$$

in which  $C_c$  represents the concentration of coal at time  $t$  and  $K_c$  is the specific reaction rate. After integration, eq. 1 becomes

$$K_c = \frac{2.303}{t} \log \frac{C_{c_0}}{C_c} \quad [2]$$

in which  $C_{c_0}$  is the initial concentration of “coal” and  $t$  is in hours;  $K_c$  is in units of percentage points per hour.

**Results of the Oxidations:** The condition noted initially and which led to the closer study of oxidation rates of various specific gravity fractions, was that reaction rates increased with decreasing density

of the fraction as is clearly shown in table III. At the same time it was noted that the “brightness” of the fractions increased with decreasing density. No attempt was made to assess “brightness” qualitatively by determining relative numbers of “bright” and “dull” particles but the generalization was based on appearance only. In some cases, as for example samples No. 87 and 88, the ash content of both fractions was low but the differences in “brightness” were striking.

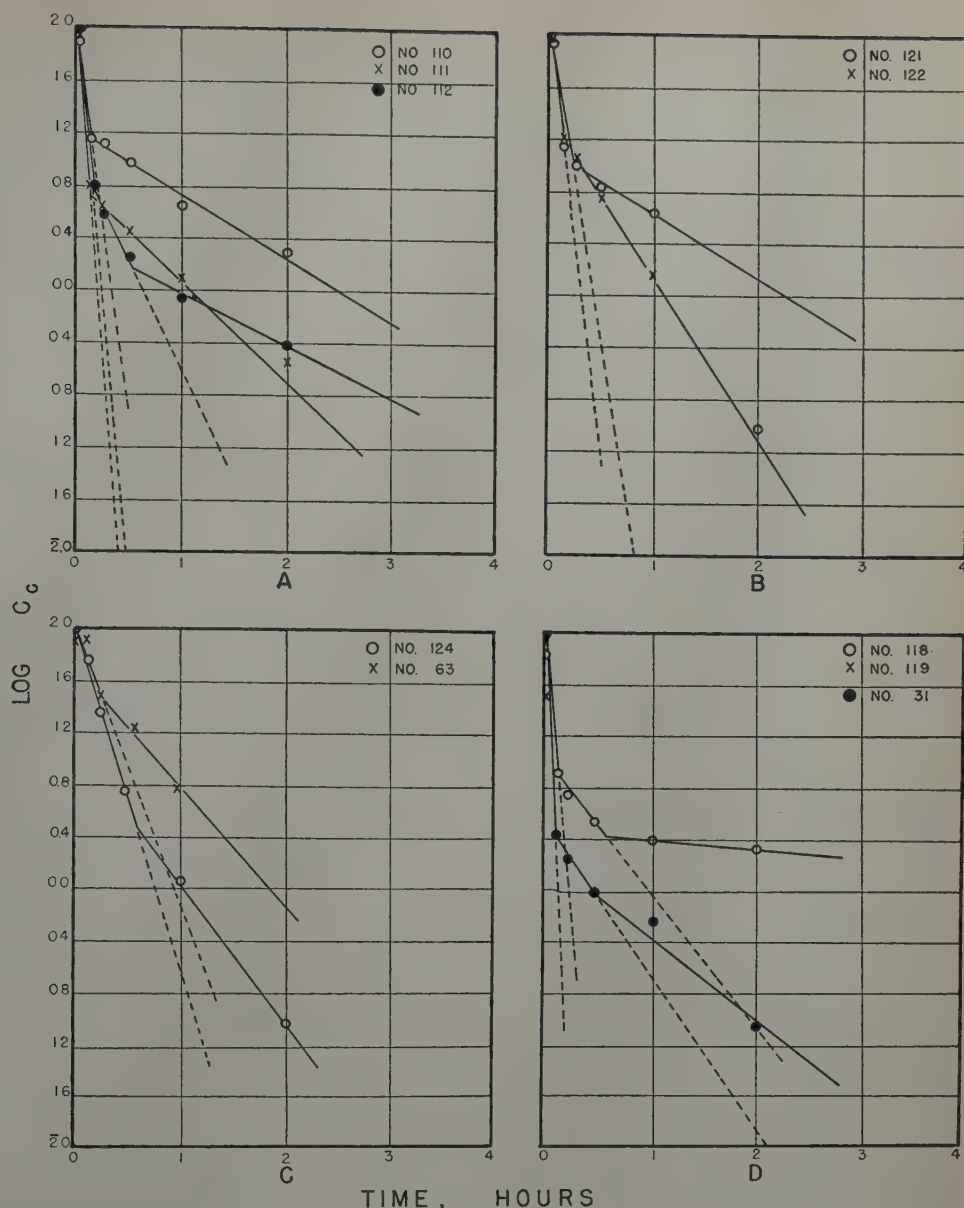
The value of  $K_c$  at 0.125 hr was generally lower than its value at 0.25-hr oxidation for Clements bed samples. Without exception, the maximum specific reaction rate occurred at 0.125 hr for all fractions of all the lower rank coals. These facts indicate that the higher rank Clements coal was comparatively slow to begin oxidation. Sample No. 87, although of comparatively high rank, oxidized much like the lower rank coals. All lower rank fractions exhibited high initial specific reaction rates.

The samples of natural, raw size fractions showed the same variations as were shown by the float-and-sink fractions but they generally had a wider range of values for  $K_c$  than any single specific gravity fraction. This undoubtedly was due to the greater heterogeneity of their petrographic composition.



**Fig. 8—Logarithm of concentrations of "coal" plotted against time of oxidation.**

(A) Three specific gravity fractions of —No. 4 + 0 Milldale bed coal. (B) Two specific gravity fractions of —No. 4 + No. 50 Black Creek bed coal. (C) —No. 50 + 0 + 1-in. Clements bed raw coal. (D) —No. 50 + No. 200 and —No. 200 raw Black Creek bed coal and raw Clarke County lignite.



The rather wide variations in the values of  $K_c$  for different oxidation intervals differ from the findings of Fuchs and associates.<sup>2</sup> They point to the fact that for the three coals on which they had complete data,  $K_c$  was fairly constant. Their fig. 4, in which  $\log C_c$  was plotted against time, exhibits a straight line relation. They say, "the straightness of the  $\log C_c$  versus time lines and the constancy of the values of  $K_c$  vary from coal to coal, in no way conflicts with the assumption of a first-order mechanism. It is fully realized that 'coal' is a mixture of petrographic components and that the relative proportion of these components will vary from sample to sample." Figs. 7A, B, C, and D and 8A, B, C, and D show that  $\log C_c$  versus time does not always yield a straight line but that at least two straight lines with differing slopes are often required to satisfy the points. In spite of these differences, this worker believes that the views of Fuchs and associates, quoted above, are essentially correct for coals containing very large proportions of one pure "coal" component. The very departures from the relationships that were suggested by them may serve as a basis for an approximation of petrographic composition.

For some coals it is possible that the nonfusain

"coal" material is uniform in composition, that is, it contains essentially but one component. In this case one should expect a constant value for  $K_c$  and a straight line for the time- $\log C_c$  relation. If two or more components are present and all have substantially the same reactivity toward nitric acid, the time- $\log C_c$  relation again should yield a straight line. If, however, two components having widely differing reactivities are present, two straight lines would be needed to satisfy all the points. If there are a great many components or if reactivities are similar, a curved line changing gradually in slope should be expected. In figs. 7 and 8, all three of these types of arrangement are suggested for one or more samples. The fact that, in most cases, two lines seem to satisfy the points best and the rather sharp changes in directions are observed, strongly suggests differing reactivities toward nitric acid. Fig. 8c illustrates a possible interpretation of the meaning of single lines or intersecting segments. Sample No. 88, representing the float 1.60-sink 1.28 fraction, apparently has only one component with regard to reactivity toward nitric acid. The second point does not fit the line drawn very well, indicating a slow start for the oxidation action. Sample No. 87,

representing the float 1.28 material, is evidently multicomponent. The first three points fit a line with a very steep slope. The last of these points and the next two fit another line with a smaller slope, only a little greater than the slope of the line drawn for sample No. 88. The last point fits neither line and is close to the last point of No. 88. Some of the points do not fit perfectly the lines drawn, but there are enough good fits to establish the presence of definite changes of direction. The oxidation rate of the more reactive component evidently governs the overall rate early in the period but after it is reduced to a small concentration, the average rate becomes very nearly that of the less reactive component which remains in considerable quantity.

**Table IV. The Percentage Present and the Specific Reaction Rate of Each "Coal" Component**

Sample No.	$C_{c_{0-1}}$	$C_{c_{0-2}}$	$C_{c_{0-3}}$	$C_f^a$	$K_{c_1}$	$K_{c_2}$	$K_{c_3}$
126		42	b	33		1.1	
127		74		26		1.9	
128	49	45		6	3.8	2.2	
129	93	6		1	5.9	1.7	
88		95		5		2.8	
87	82	17	c	1	11.2	3.6	
100		82		18		2.0	
102	72	25		3	5.5	2.1	
103	97	2		1	6.2	1.7	
110	66	16		18	13.8	1.1	
111	90	7		3	22.0	1.7	
112	88	8	2	2	20.8	3.7	0.9
121	75	13	c	12	10.5	1.2	
122	73	24	c	3	15.0	2.8	
63	36	51	c	13	4.7	2.1	
124	86	10		4	5.8	2.4	
119	33			67	20+		
118	59	7	3	31	17.4	7.5	0.1
31	95	2	2	1	32.1	5.3	1.9

<sup>a</sup> Concentration of fusain, pct.

<sup>b</sup> Quantity of heavier fractions of No. 126 not determinable by this method.

<sup>c</sup> A small quantity of less reactive material is represented by a point at 2 hr oxidation or with a concentration at 2 hr so small that its logarithm is less than 2.00.

It is quite probable that the composition of even a narrow specific gravity range of a coal is somewhat more complex than one, two, or three intersecting lines can represent accurately. It is of assistance, nevertheless, to be able to detect the presence of two or more distinct components with differing average reactivities even if each phase itself has a composition of considerable complexity.

The method of extrapolation, to zero time, of the flat portion of each complete oxidation curve to determine the percentage of fusain suggests the same procedure for the other coal components. Figs. 1 to 6, which are drawn with linear coordinates, are not suitable for this purpose as the oxidation rate of the "coal" components is not linear with respect to time. Figs. 7 and 8, however, are suitable since the log of the rate of change in concentration of any given component is linear with respect to time.

Assuming that  $C_o$  represents the concentration of all "coal" components in the original, dry, ash-free sample at zero time and not just the concentration of component  $C_{o-1}$ , and that the intersection of the zero time ordinate by the second straight line portion of the curve represents the concentration of the second most reactive component of the coal plus that of all less reactive components, the values for  $C_{o-1}$ ,  $C_{o-2}$ , and  $C_{o-3}$  in table IV were calculated. The values given are the concentration of each component, shown in figs. 7 and 8, at zero time, or the percentage of each in the original dry, ash-free coal.

The specific reaction rates shown for each component in table IV were calculated from eq. 2 with the values of  $\log C_{o-2}$ , and  $\log C_{o-3}$ , read from figs. 7 and 8.  $\log C_{o-1}$  at times other than zero was also read from figs. 7 and 8. To give a sum of 100 pct for all components of the dry, ash-free coal, values for the concentration of fusain at zero time,  $C_f$ , are given.

A study of table IV will show that successively lighter specific gravity fractions are made up of increasingly more reactive material, or of decreasing quantities of material of approximately the same reactivity as that composing the heavier fractions. Samples No. 87 and 88 illustrate a case of rather simple composition. No. 88 is made up of 5 pct fusain and 95 pct of a component having a specific reaction rate of 2.8 pct per hr. No. 87 is composed of 1 pct fusain, 82 pct of a component with a reaction rate of 11.5 pct per hr, and 17 pct of a component with a reaction rate of 3.6, or only a little greater than the rate of the entire "coal" component of No. 88. In addition to the two components shown, a small quantity of material (less than 1 pct), represented by the point at 2 hr oxidation, is included in the second component.

Samples No. 100, 102, and 103 illustrate a case of decreasing quantities of the "dull" component and increasing quantities of the "bright" component. No. 100 is "dull" coal with a specific reaction rate of 2.0 and is associated with 18 pct of fusain. No. 102 contains 72 pct of another component having a much higher specific reaction rate of 5.5, and 25 pct of practically the same component as the "coal" portion of No. 100. Finally No. 103 contains 97 pct of the "bright" component and only 2 pct of the "dull" one.

In samples No. 121 and 122, the content of "bright" coal is the same in both samples but the percentage of the "dull" is, quite unexpectedly, a little larger in the "bright" fraction. It will be observed, however, that both components are much more reactive in the lighter fraction than in the heavier one.

It will be noted that the "bright" and "dull" components delineated by oxidation are not precisely identified with the familiar components of coal petrography. If there is a correspondence between "chemical" and petrographic components of "coal" and "chemical" and petrographic fusain, it should be revealed by a careful quantitative study of several coals by both methods. Undoubtedly, the more reactive component of sample No. 87 was essentially vitrain and probably the least reactive one was durain or opaque attritus. Where intermediate components appear, they probably represent attritus of varying degrees of translucency. The relative proportions of "chemical" components shown in this study do not substantiate the existence of an entity fitting the description usually given to clarain but suggest that clarain is a mixture of "bright" and "dull" components.

An inspection of the specific reaction rates of the heavier fractions of several coals in table III reveals a peculiar trend right at the end of oxidation of the "coal" portion. Samples No. 126, 127, and 121 show a slightly lower specific reaction rate for 1 hr of oxidation than for 2 hr. The same thing probably would have been true for No. 63 had not all the "coal" been oxidized in less than 2 hr. The condition was so pronounced in the case of No. 126, that fig. 7A clearly shows an increased slope for the second component over that of the first. If the second component be extrapolated to zero time, it



appears to exceed 100 pct of the sample. This condition was not an accident of analysis because the entire curve was rechecked (data not shown) with the same result.

Such a condition clearly is impossible if the thesis supported thus far is true, namely, that the more slowly oxidizable components govern the apparent oxidation rate near the end of "coal" oxidation. The only explanation which appears to be at all satisfactory is that part of the material oxidized during the last hour, in the case of some heavier fractions, does not follow the first order reaction law of eq. 2. On the assumption that the material was similar to fusain, reaction rates were calculated using the zero order reaction law which governs the oxidation of fusain.<sup>2</sup> In all cases, the rates were much larger than those for fusain in the same sample, but they decreased steadily with a decrease in the dry, mineral-matter-free fixed carbon content (rank) of the sample. As extrapolation of a line connection 1 hr and 2 hr oxidations on fig. 1 would give much too large a percentage of this intermediate material, it appears likely that the actual oxidation rate law governing this portion of "coal" may be quite complex and follow neither zero nor first order. If this is true, fusain may not be a sharply distinct component of some coals but they may contain, in addition to true fusain, some materials which are so "dull" as to be semi-fusainized.

It was reasoned that if the quantities of the various components shown in table IV represented approximately the true composition of each sample, a weighted average reaction rate for all components should be very nearly the same as average rates calculated from table III, weighted as to time. Both average rates and the algebraic differences are shown in table V.

Differences were quite small for Clements coal but for all coals of lower rank (including sample No. 87 of Clements coal) they were large. Except for Clements bed coal, table V can not be considered a confirmation of the compositions calculated for table IV. A comparison of the values in the difference column, table V, with the corresponding fixed carbon percent, table II, shows that the difference generally increases with rank. Samples No. 121, and 122 show smaller differences than their ranks indicate they should, while No. 87 and 88 show larger ones.

**Specific Oxidation Rates and Rank:** It is apparent from a comparison of tables II and III that the rate of reaction of coals with nitric acids is influenced by the rank of the coals as well as by the nature of their physical components. A study of the oxidation data regarding the relation of rank and reactivity was made.

For all bituminous coals above high volatile bituminous A rank, the sole basis for rank determination in the Standard Specification for Classification of Coals by Rank, of the ASTM, is the dry, mineral-matter-free fixed carbon.<sup>7</sup> With the exception of the lignite sample, all coals included in this study were of high volatile A rank or above, so that the fixed carbon contents, calculated with the Parr formulas as required in the Standard Specification, have been used as the basis of rank. Fixed carbon content, of course, may not be the best criterion of rank. The presence of much fusain, which is almost always higher in fixed carbon than the associated "coal," may give some coals a higher rank than other prop-

erties indicate they actually have. Total carbon content probably would have no advantage over fixed carbon, for purposes of rank determination. If a clear and explicit relation between rank and nitric acid oxidation rate could be worked out, the latter might prove more suitable than fixed carbon content for rank classification purposes.

Values of  $K_c$  from table III and of the concentration of coal at various times of oxidation were plotted against dry, mineral matter-free fixed carbon. Heating values calculated to the same basis were also tried. The composition-weighted averages from table V, plotted as a function of fixed carbon, gave an S-shaped curve with the lower inflection of the S occurring near 73 pct fixed carbon and a similar plot of the time-weighted averages was fitted fairly well by two line segments of differing slopes intersecting at about 70 to 72 pct fixed carbon.

The best distribution of points was given by values of  $K_c$  at 0.25-hr oxidation and by values of the concentration of "coal" remaining after 0.25-hr oxidation, plotted against fixed carbon. These relations are shown in figs. 9 and 10. It is readily seen that there is a considerable scattering of points in both cases. That the scattering is much greater for the "bright" coal fractions, particularly of the lower rank coals, is apparent. If the solid lines drawn are accepted as best fitting the points, fig. 9 indicates a change of about 0.77 pct per hr in specific reaction rate per percent of fixed carbon for the lower (high rank) portion of the curve and about 1.9 pct per hr for each unit of fixed carbon for the low rank portion. In fig. 10, a decrease of about 3.9 pct in the concentration of "coal" after 0.25-hr oxidation for each percentage increase in fixed carbon, is indicated.

**Table V. Composition-weighted and Time-weighted Specific Reaction Rates Compared**

Sample No.	Avg. $K_c^a$ Composition	Avg. $K_c^b$ Time	Diff.
126	1.1 <sup>c</sup>	1.7	- 0.6
127	1.9	1.7	+ 0.2
128	3.0	2.8	+ 0.2
129	5.6	3.7	+ 1.9
88	2.8	2.8	0.0
87	< 9.9	5.3	<+ 4.6
100	2.0	1.7	+ 0.3
102	4.6	3.3	+ 1.3
103	6.1	4.3	+ 1.8
110	12.2	3.5	+ 8.7
111	20.5	5.6	+14.9
112	18.8	5.7	+13.1
121	< 9.1	5.5	<+ 3.6
122	<12.0	6.4	<+ 5.6
63	< 3.2	2.6	<+ 0.6
124	5.4	4.1	+ 1.3
119			
118	>15.6	4.1	>+11.5
31	30.9	6.9	+24.0

<sup>a</sup> Average weighted by percentage of each "coal" component, table IV.

<sup>b</sup> Average weighted for length of interval, table III.

<sup>c</sup> For first component only.

Black Creek bed coal shows little evidence of a dependence of reactivity upon rank. The independence of rank shown by the particular low rank coals studied may be partially accounted for by the very low proportion of "dull" material present, as shown in table IV. The "bright" fractions of Clements bed coals, samples No. 87, 103, and 129, exhibit widely differing reactivities although they are almost identical in rank. Of the three, sample

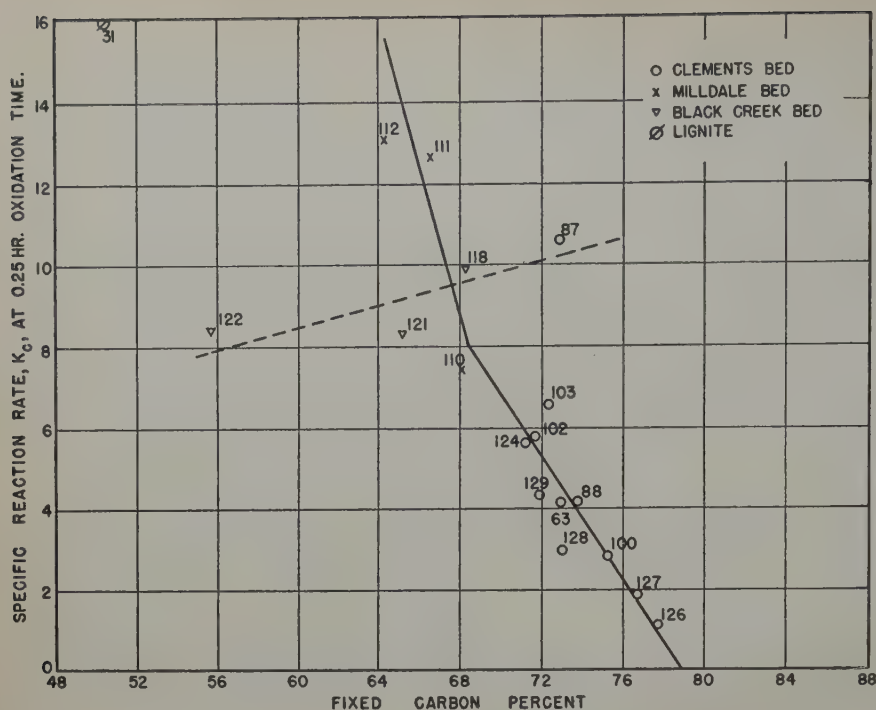


Fig. 9—Specific reaction rate at 0.25-hr oxidation time as a function of dry, mineral-matter-free fixed carbon (rank).

No. 129, the "bright" fraction from the higher rank, plus 1-in. size range, conforms most closely to the general trend indicated by the solid lines in figs. 9 and 10.

It is obvious, then, from the preceding discussion that the higher the rank the more definite the relation between rank and oxidation reactivity. This is confirmed by a private communication from C. C. Wright. Concerning unpublished work of a similar nature done at the Mineral Industries Experiment Station at the Pennsylvania State College, he says, "I believe that if the interest is limited to coals of a given rank something could be worked out. The higher the rank the more probability of definite correlations being developed judging by the rather limited results we have on record."

It must be concluded from this part of the study that no general criterion can yet be proposed for untangling the effects of rank from those of variations in petrographic composition, except in the case of coals or fractions having practically the same rank. Study of a larger number of coals and specific gravity fractions of coals of the lower ranks will be necessary if any general rank-oxidizability relation is to be found.

**Suggested Uses:** In addition to the possible use of oxidation data for determining approximate quantities of some petrographic components in coals, it should yield valuable information as to the suitability of coals for different purposes. Fuchs and associates<sup>2</sup> seemed to get the same relative orders of reactivity as are reported in this paper by the use of some slightly different oxidizing agents. Rees, Wagner, and Tilbury found that reactivity indexes obtained by the C.R.L. test generally increased with rank, indicating decreasing reactivity.<sup>8</sup> They found no definite trend in reactivity with respect to petrographic composition but suggested that the C.R.L. test probably measures more nearly the reactivity of the most active component in a mixture. This was confirmed when they found an increase in the reactivity indexes of fusain from which the more reactive components had been removed by boiling

in nitric acid. It should be stated that highest rank coal tested by Rees and associates was about the equivalent of the Black Creek coal of the present study, and that rank and petrographic composition effects are, as has been shown in this study, more difficult to untangle for the coals of lower rank.

Oxidation rate data should be useful in the evaluation of size and specific gravity fractions of a given coal as to relative suitability for direct hydrogenation. The Bureau of Mines<sup>9</sup> found essentially the same relation between rank and petrographic composition and amenability to hydrogenation as is reported in this paper for rank and petrographic composition with regard to reactivity toward nitric acid. Values of  $K_0$  calculated from oxidation rate data should be even more useful than rank for this purpose. If a coal contains considerable "splint" coal in bands, or disseminated, the specific gravity at which splint would be largely eliminated could be determined without much trouble. Lessing<sup>5</sup> has suggested that the "bright" portion of banded coal is more likely to be coking than the dull fraction. Oxidation data should indicate roughly the proportion of "bright" coal suitable for blending for coke, that could be expected from any specific gravity or size fraction of a coal of the proper rank, even if the whole coal were not quite suitable for that purpose.

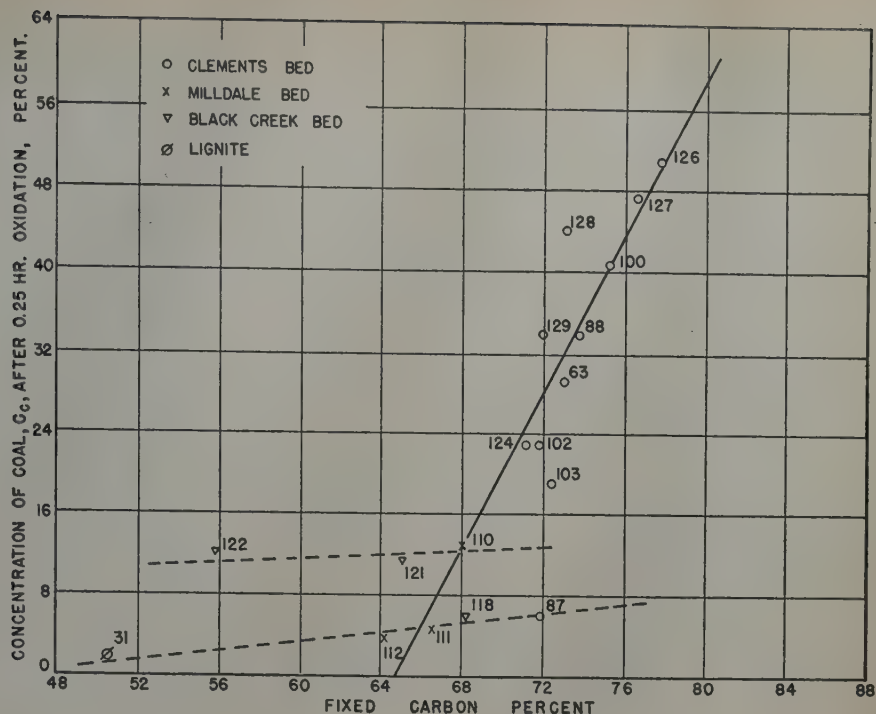
### Summary

One medium volatile bituminous and two high volatile A bituminous coals were separated into specific gravity fractions and samples of each fraction were oxidized in 8 N nitric acid for seven different periods of specified length. In addition to the determination of fusain content, studies were made of the first five oxidation periods during which nonfusain "coal" material was oxidized. Two raw samples each of the medium volatile bituminous, and the lowest rank high-volatile bituminous A coals, and one raw sample of lignite, were also studied.

The quantity of dry, ash-free residue at the end of any given oxidation period was found to decrease



Fig. 10—Concentration of coal after 0.25-hr oxidation as a function of dry, mineral-matter-free fixed carbon (rank).



and the specific reaction rate was found to increase with decreasing density of the fraction.

When the logarithms of "coal" concentration were plotted against time, for a few of the heavier fractions, all points fell on or near one straight line; in nearly all other cases, two straight lines with considerably different slopes were sufficient to satisfy all points.

Extrapolation of the line segments to zero time on log  $C_c$ -time plots, yielded estimates of the percentage of each coal component present. The specific reaction rate of each component was readily calculated.

A very close relationship between rank and both concentration of "coal" and specific reaction rate at 0.25-hr oxidation was found to exist for the medium volatile and the higher rank high-volatile A bituminous coals but the relationship was not clear for lower rank coals. The specific reaction rates for the very "bright" fractions seemed to be more nearly independent of rank than the rates for the other fractions.

In addition to the possibility that the method can be used to estimate the quantity and chemical reactivity of the different petrographic components in a given coal, it may yield valuable information as to the suitability of various coals or size and density fractions of the coals, for many chemical uses. Specific reaction rates might prove a more superior criterion of rank, in many cases, than the present chemical analysis.

Much more work remains to be done before the relative effects of petrographic composition and of rank upon specific reaction rates can be untangled. Extensive studies will be required also in order to make positive correlations of components determined chemically and the petrographic components commonly recognized optically.

#### Acknowledgment

The author wishes to acknowledge his indebtedness to Robert D. Brown of the Department of Chemistry, University of Alabama, who read the paper from the standpoint of the physical chemist.

C. C. Wright, along with C. R. Kinney, T. S. Polansky, and H. B. Charnbury, all of the Fuel Technology Department of the Pennsylvania State College, read the paper in a more primitive form. They made many invaluable suggestions, practically all of which are included in the paper in its present form.

Finally the author wishes to acknowledge the valuable assistance of E. L. Thomas, Jr., now of the Tennessee Coal, Iron and Railroad Co. All analyses were made by him with great skill and care. To him should go much credit for the consistent and reproducible data obtained from a tedious and difficult determination.

#### References

- <sup>1</sup> Reynold Q. Shotts: The Distribution of Fusain in Various Size Fractions of Three Alabama Coals. Paper read before the Ala. Acad. Sci. University, Ala. (April 16, 1948).
- <sup>2</sup> Walter Fuchs, A. W. Gauger, C. C. Hsiao, and C. C. Wright: The Chemistry of the Petrographic Constituents of Bituminous Coal. Part I. Studies on Fusain. Pa. State College, Min. Ind. Expt. Station. *Bull.* 23 (1938) 43 pp.
- <sup>3</sup> National Research Council: Chemistry of Coal Utilization. Chap. 3 and 9. 1945. New York. John Wiley and Sons, Inc.
- <sup>4</sup> G. C. Sprunk, W. H. Ode, W. A. Selvig, and H. J. O'Donnell: Splint Coals of the Appalachian Region: Their Occurrence, Petrography and Comparison of Chemical and Physical Properties with Associated Bright Coals. U. S. Bur. Mines. *T. P.* 615 (1940) 59 pp.
- <sup>5</sup> R. Lessing: The Rational Preparation of Coal, *Fuel*, (May-June 1947) 26, (3) 57-73.
- <sup>6</sup> Barksdale, Jelks: Lignite in Alabama. Ala. Geol. Survey. *Bull.* 33 (1929) 64 pp.
- <sup>7</sup> American Society for Testing Materials: Standard Specifications for Classification of Coals by Rank. A.S.T.M. Designation: D388-38. ASTM Standards on Coal and Coke (Aug. 1947) 159 pp.
- <sup>8</sup> O. W. Rees, W. F. Wagner, and W. G. Tilbury: Chemical Characteristics of Banded Ingredients of Coal. Ill. Geol. Survey, *R. I.* 132 (1948) 13 pp.
- <sup>9</sup> Arno C. Fieldner, Henry H. Storch, and Lester R. Hirst: Bureau of Mines Research on the Hydrogenation and Liquefaction of Coal and Lignite. U.S. Bur. Mines. *T. P.* 666 (1944) 69 pp.

# Wrapping Pillars

## With Old Hoist Rope

by B. T. Wykoff



Completed pillar before rope clamping on solid side of open eye bolts.

**This paper describes an important function of mining in Southeast Missouri. The practice is a necessary procedure to prevent disintegration of the pillars. Although the technique is peculiar to this district, it might have application in other mines using room and pillar method.**

CONSIDERING the subject alone, this paper would cover only a mechanical treatise of pillar roping. Because the St. Joseph Lead Co. originated the practice, it is proper to give a short introduction to the company. It is also proper to describe pillars and discuss their purpose to justify expending large sums of money to preserve them after the surrounding area has been thoroughly mined.

Because the St. Joseph Lead Co. originated the practice of back pinning with wedge bolts, patch plates and channel irons, and the pouring of substitute pillars of concrete, these too, are mentioned.

The Southeast Missouri Division of the St. Joseph Lead Co. is the largest lead producer in the United States. About 95 pct of the present producing mines are in St. Francois County, which begins 60 miles south of St. Louis. The entire lead ore production of this county, at the present time, is mined and milled by this company, treatment plant capacity being about 22,000 tons per day.

This area is also the oldest, lead having been mined at Mine La Motte, in Madison County, in 1723. Estimated production of all the various companies that have mined here is about 7,000,000 short tons of pig lead.

### Method of Mining

The lead occurs in the ore as galena. Galena is found either in the form of disseminated or solid streaks, of varying thickness. Ore stratum is the Bonne Terre dolomite, which is about 350 ft thick,

but almost all the ore is in the lower 100 ft. Ore is mined by room and pillar method in open stopes. These stopes vary in height from 7 ft to 200 ft, depending on the thickness of the pay ore in that spot. Back, or roof, is supported entirely by pillars. In a majority of stopes, after the loose scales are mined down, the back has remained solid for many years (fig. 1).

Under ideal conditions pillars are spaced to form an equilateral triangle and are as small and as far apart as the height of the stope and the character of the back will permit. They vary in diameter from 10 ft to 50 ft and are from 18 ft to 50 ft apart, measured from the outer edges. In general, the pillars are approximately 12 ft in diameter, and 25 ft apart. The spacing is very uncertain because of the vagaries of the ore trend, the desire to better support breaks in the back, and the attempt to spot out the thinner ore by not placing one in the core of the ore body.

### Use of Structural Steel Channels

In stopes where the back is bad because the overlying formations are thin with wet, weak and shaly

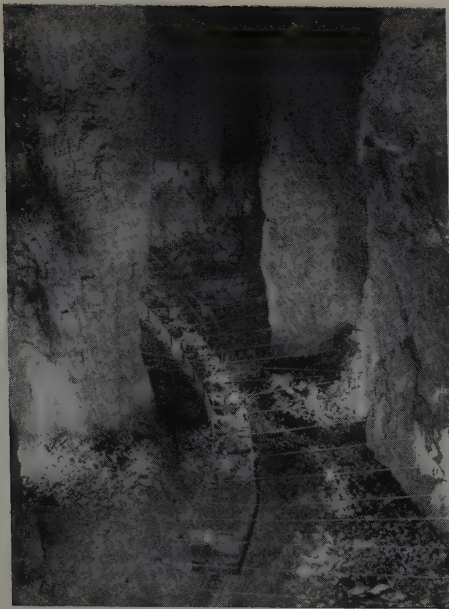
---

*B. T. WYKOFF, Member AIME, is Mine Superintendent, St. Joseph Lead Co., Bonne Terre, Mo. AIME Columbus Meeting, September 1949.*

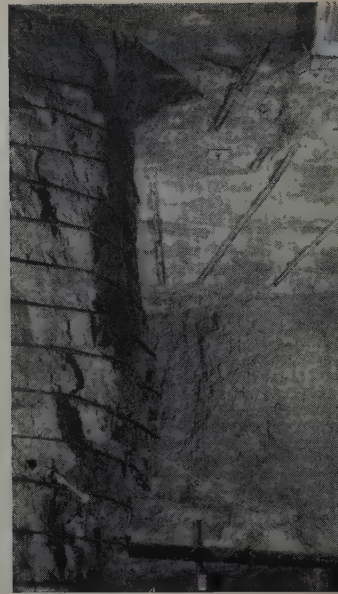
*TP 2906 A. Discussion (2 copies) may be sent to Transactions AIME before Sept. 29, 1950. Manuscript received Sept. 15, 1949.*

---





**Fig. 1 (left) — Room and pillar mining. In most stopes, the back has remained solid for many years.**



**Fig. 2 (right) — Channel iron in backing. Wedge bolts with patch plate washer stop peeling in loose areas.**



**Fig. 3 (left) — Concrete pillars. Concrete pillars substituted for normal pillars allow removal of old pillars in rich areas.**



**Fig. 4 (right) — Split pillar. Pillars weather and dry out, and shifts cause cracks and sluffing.**

**Fig. 5—Hour glass pillar, fenced and gunited.**







**Fig. 6—Irons on split pillar.**

seams, or are split vertically by slips and faults, additional support is needed, and this is adequately provided by the use of 4 in., 6½ lb structural steel channels. These channels are pinned flat against the back by 1 in. round mild steel threaded wedge bolts 10 to 12 ft long, in holes drilled on a 45°, about 3 ft apart, which reach up into a solid layer. End holes are drilled over the tops of pillars; the irons are of any length and are spaced any necessary distance apart. By the use of an impact wrench, nuts are drawn up tightly on an angle washer placed in the web of the iron. When properly driven, these rods have tested 20 tons each before slippage, elongation, or breakage. If the loose area is small, a single wedge bolt in a vertical hole, using a patch plate washer, has been sufficient to keep the back from peeling (fig. 2).

Pillars represent millions of tons of potential ore reserves. They contain about 10 pct of the original

**Fig. 7—Patch plates on pillar which later required roping.**



ore body. After the back and bottom have been thoroughly prospected for secondary ore and none found, slabbing is done for partial pillar recovery. This practice is unsatisfactory if the whole pillar can be recovered later. In one mined-out area of rich ore, some substitute pillars of concrete were poured between the old pillars. This allowed complete removal of the pillars but is economically satisfactory only in rich areas (fig. 3).

Some safe and economical method will be developed, and the pillar recovery program can become an important part of mining. Until that time, the old mined-out areas have to be inspected regularly and the old pillars guarded carefully, not only for the safety of men working in the secondary ore and the main lines running through these old stopes, but to see that no area is lost from cave-ins due to pillar failure, thereby losing forever the recoverable ore in the failed pillars.

Maintaining safe back is no problem provided the pillars remain intact. However, if the pillars start



**Fig. 8—Pillar roping in action.**

taking weight, splitting and cracking, spalling off slabs or if certain mineral bands disintegrate within, the problem is acute and immediate.

Splitting, cracking, and spalling off slabs are caused by minute vertical seams and slips which, at the time the pillar was formed, were filled with live mineral of high moisture content. The pillar at this time was more or less elastic and able to withstand high pressures and side thrusts without visible effect. However, after standing for several years in natural ventilation currents of varying temperatures and humidity, the pillars weather and dry out. The mineral seams no longer are adhesive, the least shift causes cracks and sluffing. The mineral bands, mainly galena and pyrite, oxidize and become soft, slimy material (fig. 4).

The glauconite and shale bands shrink and laminate; finely divided particles separate; and the pillar assumes definite hour glass shape. First stages





**Fig. 9—Pillar roping in action.**

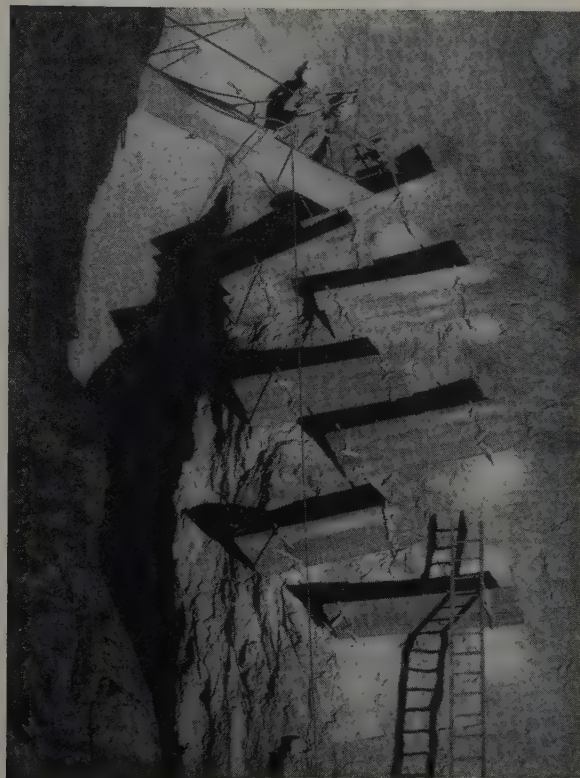
of disintegration have started, and mechanical assistance is needed to prevent complete loss (fig. 5).

Starting at the focus of trouble, where the pillars are the worst, the salvage campaign is put under way. Preliminary to rehabilitation, the pillars are prepared by removing all small spalls and pieces which are not necessary. If the loose is large and the pillars small, the pieces may be held together by tie-rods, threaded on each end, placed in holes drilled completely through the pillar and the nuts drawn up tightly on patch plates. If the pillars are large, these rods can be wedge bolts, long enough to reach the solid section of the pillar, and the patch plates can be channel irons placed vertically on the sides or circled around as in banding. If the ground in the area is not too heavy, and deterioration of the pillars not too far advanced, the use of these strengthening devices is usually sufficient to retard failing indefinitely. On the other hand, if the ground is heavy, and is an important area from

standpoint of plant and rich ore, and is required to be safe for the future mine, then it is best to resort to wrapping (figs. 6 and 7).

Hoist rope in the district varies from  $\frac{7}{8}$  in. to  $1\frac{5}{8}$  in., depending on whether the shafts are used only for man cages, exclusively for ore skips, or for a combination of cage and skip. Minimum factor of safety for men is 7.5, for ore 3.2, and both go as high as 12.0. Ropes are changed when they have been in use 3 years or when their strength has lowered to 80 pct of catalog value, other conditions such as lubrication and usage being normal. From this, it may be deduced that although a rope has been discarded for hoisting it may be in excellent condition for other uses, the most important of which has been underground on pillars.

The rope sizes most available are  $1\frac{1}{8}$  in. and  $1\frac{1}{4}$  in. These still have breaking strength of 30 to 40



**Fig. 11—High traps.**

tons. After winding on the old reels, they are sent underground to the area in need. Here they are placed on supports to facilitate unwinding. Several loose turns are made around the base of the pillar, using an air powered tugger hoist, if available, in order to eliminate heavy dragging. Pillar roping then begins with the first hole being drilled into the pillar close to the top. Into this hole is driven a welded eye wedge bolt. Fourteen inch is standard length. Through this eye the loose end of the rope is threaded and doubled back, being firmly secured with proper size rope clamps. New holes are drilled down the side of the pillar, in carefully selected, solid spots, one hole for each foot of vertical drop. These holes should not be in a vertical plane, but at an oblique angle. If the vertical plane were a weak section, all these holes might



**Fig. 10—Come-a-long, chain hoist, and auxiliary tugger.**





**Fig. 12—Properly roped pillar with back channels and patch plates.**

split the pillar under pressure. Into these holes are driven open eye wedge bolts. All these bolts are about 14 in. long and 1 in. round mild steel. Each turn of the rope, one at a time, is laid in its corresponding open eye, a come-a-long clamped on the loose end onto which is hooked a 10-ton Coffing chain hoist, the tail hook of which is securely fastened to a bolt in the floor or a neighboring pillar. On high pillars it is best to get the tail high for a straight-away pull (figs. 8, 9, and 10).

In pulling the rounds tight, care is exercised to place the rope around to the best advantage in order to cover cracks and to pull slabs in tightly. On sharp angles a sledge hammer is used to make the rope conform to the contour of the pillar. As each turn is completed, and while the pull is still on, a rope clamp is fastened on the pull side of the open eye. Turn after turn is secured on down the pillar. Ropes are made continuous easily by over-

lapping and clamping. About every third turn a rope clamp is locked on the solid side of the open eye. These clamps serve a dual purpose; they not only make it possible to tighten each wrap individually, but also, should any rope break, they would prevent its unraveling the full length of the pillar. At most, only three rounds would be lost, as the remainder still would be held securely by the clamps and eye bolts as shown in title photograph.

Work on pillars up to 35 ft high can be done by using sectional ladders. Trapeze or suspended platforms are required above 35 ft. These traps are run out to the pillar from a neighboring high breast. Work below is facilitated by a boatswain's chair or ladders dropped down from the traps (fig. 11).

In early practice, wooden wedges were driven between the rope and pillar. The use of these has been discontinued because of the instability of the wood over long periods. Chicken wire also has been placed over the completed pillars and the whole covered with a coating of gunite. This is an unnecessary expense because, although guniting is excellent to prevent oxidation and disintegration, it should be applied before the trouble starts.

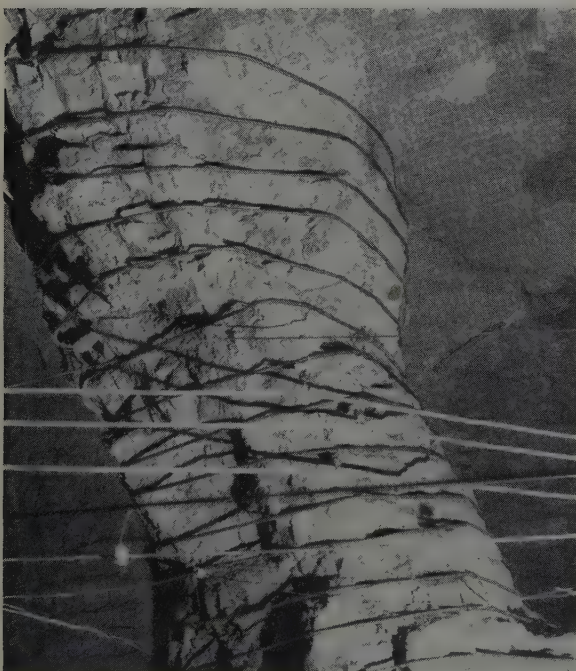
Costs were recently kept on a series of pillars 18 ft high and 14 ft in diameter requiring 15 rounds per pillar 45 ft in circumference. Three men wrapped 6 pillars in 6 weeks, 30 working days or 90 man shifts.

Time includes getting material to the area and that consumed in roofwork.

#### Per Pillar:

Rope, 675 ft @ \$0.05 equals \$33.75 (rope cost is surface labor wind on reels and send underground). Clamps, 23, 1½ in. @ 0.85, \$19.55. Eye bolts, 17, @ 0.30, \$5.10. Wedges, 17, @ 0.06, \$1.02. Total material, \$59.42. Total labor, \$186.20. Total cost per pillar, \$245.62.

Properly roped pillars need no extra precaution for a long life (see figs. 12, 13, and 14).



**Fig. 13—Properly roped pillars over ten years old.**



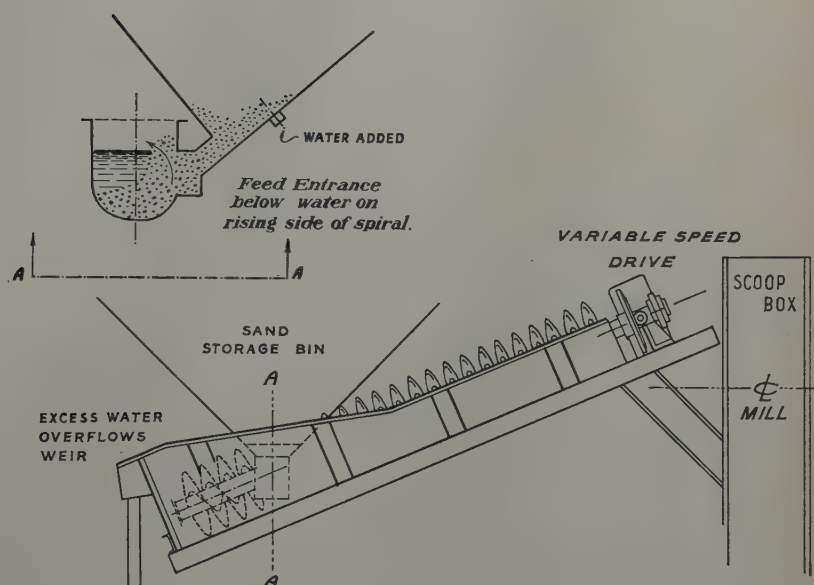
**Fig. 14—Roped and concrete pillars in same area.**



# The Use of Spiral Classifiers as Ball Mill Feeders

by T. C. King

Fig. 1—Diagram of 24-in. spiral classifier used to feed wet sand from storage.



AT the new Graham-Central Mill of Eagle-Picher, near Galena, Ill., material is simultaneously dewatered and introduced into the ball-mill scoop boxes by the use of variable-speed, 24-in. spiral classifiers. This arrangement has proved satisfactory in that the moisture content of the wet feed is reduced, thus allowing maintenance of density within the mills, and a constant feed at any desired rate to the mills is provided.

In this mill the primary feed is wet crushed to minus  $\frac{1}{2}$  in. and preconcentrated by jigs to produce a bulk lead-zinc-iron concentrate and a waste tailing. The jig concentrate is then dewatered by a conventional dewatering classifier and stored in bins ahead of the ball mills. Even though this material is well dewatered, the discharge at the bin bottom is sloppy because of the gradual drainage of the water from the surface of the particles. Conventional belt feeders with vertical cutoff gates are unsatisfactory because of the flushing characteristic of the wet sand.

The new installation employs 24 in. x 10 ft, 9-in. Simplex double-pitch spiral classifiers for feeders from the bin bottoms to the ball mill scoop boxes (fig. 1). These units are arranged with straight tanks

and are driven by 5 hp motors through a U. S. Vari-Drive to provide a 4 to 1 speed ratio.

Each bin discharges through a chute into the rising side of the spiral. The flushing characteristic of the material from the bottom of the bin is utilized to maintain a steady feed. A small amount of water tapped into the bottom of the bin insures uniform feed to the classifier. Excess clear water drains over the weirs.

Feed rate to the mills is controlled by the variable-speed drives. The feed delivered by the spiral at a given speed setting is relatively dry, uniform, and steady, and as such is easily controlled by the operator. The classifiers are set on a pitch of 5 in. per ft and have a sand-raking capacity of from 8 to 32 tons per hr.

It is believed that the solution of this problem offers possibilities for the storage of deslimed rod-mill sands and other similar products of the industry.

T. C. KING, Junior Member AIME, is Metallurgist, Eagle-Picher Mining and Smelting Co., Miami, Okla. AIME Columbus Meeting, September 1949.

TP 2873 B. Discussion (2 copies) may be sent to Transactions AIME before Oct. 31, 1950. Manuscript received September 9, 1949.

# The Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions

by C. G. Fink and G. L. Putnam

The dissolution of gold by cyanide solutions was studied by determining the time required for the solvents to dissolve gold leaf. Minute traces, even 0.5 ppm, of sulphide ion retard the dissolution of gold, and this behavior cannot be accounted for by the presently accepted hypotheses involving oxygen-depletion or thiocyanate formation. On the other hand, traces of the salts of lead, bismuth, mercury, and thallium considerably accelerate the dissolution. The beneficial effect of lead is dependent upon the pH and cyanide ion concentration of the solution.

ALTHOUGH cyanide solvents are used very extensively for the recovery of gold from ores and concentrates, the normal rate of solution of the precious metal as determined by Barsky, Swainson, and Hedley,<sup>1</sup> and others<sup>2</sup> is less than about 3 mg per sq cm of exposed gold surface per hour, corresponding to a corrosion depth of approximately 1.5 microns (0.00006 in.) per hr. Although the slow rate may be increased to some extent in certain cases, as, for example, when the gold is partially imbedded in iron pyrites,<sup>3</sup> the leaching operation is generally the most time-consuming step of the cyanide process.

Frequently it is necessary for the leaching solvents to remain in contact with the ores for 50 to 200 hr in order to dissolve all of the gold particles.<sup>4</sup> Not only does the slow solution rate cause some increase in the capital costs, but the cyanide consumption also may be increased because of vaporization of hydrocyanic acid and secondary reactions with minerals such as malachite ( $\text{CuCO}_3 \cdot \text{Cu(OH)}_2$ ), and pyrrhotite, which often are associated with gold. It is not surprising that numerous attempts have been made to accelerate the dissolution of gold in cyanide solutions.

## Experimental

**Gold Leaf Test:** The gold leaf test was apparently originated by M. Faraday, who, by means of it, discovered that dilute cyanide solutions would dissolve gold readily in the presence of oxygen. The usefulness and reliability of the test have been discussed elsewhere.<sup>5, 6</sup> Essentially, the method consists of determining the time required for 5-ml portions of cyanide solvents to dissolve squares of 23 carat gold leaf, 0.5 cm on edge, when shaken in 10-ml test tubes with the solvents. The weight of gold leaf per test was 0.051 mg as determined by the direct weighing of a leaf 8.6x8.6 cm (weight = 15 mg) and measuring the area of the test sample (0.25 sq cm). Gold leaf is of remarkably uniform thickness.<sup>6</sup> Since gold leaf, like native gold, contains small amounts of copper and silver, the results are comparable with those one might expect in cyaniding gold ores. As six or more tests can be made simul-

taneously, the relative efficiencies of the solvents can be evaluated readily.

All dilute solutions of lead, bismuth, thallium, and mercury salts were prepared by the ordinary dilution method used by chemists. Our C.P. sodium cyanide and buffering reagents had no detectable amounts of either sulphide ion or heavy metals.

**Soluble Sulphides in the Cyanide Process:** The action of soluble sulphides is of interest in the study of the dissolution of gold in cyanide solutions. That gold ores which contain compounds of arsenic and antimony often are not amenable to direct cyanide treatment is well known, and it is said that arsenic and antimony are "cyanicides."<sup>7</sup> It is commonly believed that part of the sulphur content of such ores is soluble in alkaline solutions to give sulphide ion, which reacts with the free cyanide content of the leaching solutions to form inert thiocyanates or reacts with the dissolved oxygen to form sulphites or sulphates.<sup>8</sup> A purpose of this paper is to demonstrate that sulphide ion may behave in a manner that cannot be explained by either the oxygen-depletion or thiocyanate formation theories and to prove directly that concentrations of sulphide ion which cannot be detected even by colorimetric methods may retard the dissolution of gold.

---

C. G. FINK, Member AIME, is Professor Emeritus, Department of Chemical Engineering, Columbia University, New York, N. Y., and G. L. PUTNAM is Research Associate, Department of Chemical Engineering, University of Washington, Seattle, Wash.

A part of this paper is part of a dissertation submitted in partial fulfillment of the requirements for the degree of Doctor of Philosophy, Columbia University.

This paper is one of a series of publications dealing with the applied chemistry of gold (cf. ref. 5, 6, 7, and U. S. Patents 2,283,196 and 2,283,198). Other papers are in preparation.

AIME New York Meeting, February 1950.

TP 2901 B. Discussion (2 copies) may be sent to Transactions AIME before Oct. 31, 1950. Manuscript received March 9, 1949; revision received April 10, 1950.



Courtney and Maguire<sup>9</sup> have studied the effect of sodium sulphide on the extraction of gold from its ores by solvents containing 3.1 g of sodium cyanide per liter. They gave no hypothesis to account for their observations, and the concentrations of sulphide ion in their solvents all exceeded 600 mg per liter.

Solvents were prepared which contained, per liter, 1.0 g of sodium cyanide, 10.6 g of sodium carbonate, 8.40 g of sodium bicarbonate, and variable amounts of sodium sulphide. The carbonate and bicarbonate are inert<sup>5, 9</sup> and were added to maintain a constant pH. When the efficiencies of the solvents were compared by the gold leaf method, the results shown in fig. 1 were obtained. It is noted that less than 0.50 mg of sulphide ion per liter is required to retard appreciably the dissolution of gold leaf. To oxidize this concentration of sulphide ion to sulphate, 1.00 mg of oxygen per liter would be required; yet cyanide solutions saturated with air at 25°C contain about 8 mg of oxygen per liter, and the oxygen solubility is not decreased by small percentages of dissolved salts.<sup>1</sup> Moreover, the oxygen contents of the solutions would have been brought rapidly to saturation by the agitation of the test tube. With regard to thiocyanate formation, this factor could have accounted for the loss of only 1.6 mg of sodium cyanide per liter from 1000 mg, and we have confirmed the fact that sodium thiocyanate<sup>9, 10</sup> does not retard the dissolution of gold.

It must be clear, therefore, that neither the oxygen-depletion nor the thiocyanate-formation theories can explain the behavior of sulphide ion. An important factor involved may be adsorption of sulphide ion by the gold at the metal-solution interface, thereby protecting the metal from corrosion by the solutions. Sulphides are harmful at concentrations below the ranges of the usual colorimetric methods.

**Metal Salts in the Cyanide Process:** The literature on the subject may be summarized as follows:

**Lead Salts:** According to Haden,<sup>10</sup> the use of lead compounds in large-scale operations at the Morro (Brazil) gold mine has permitted increased gold recoveries because of the buffering action of lead hydroxide and control thereby of the pH. But Haden overlooked the valuable accelerating effect of lead

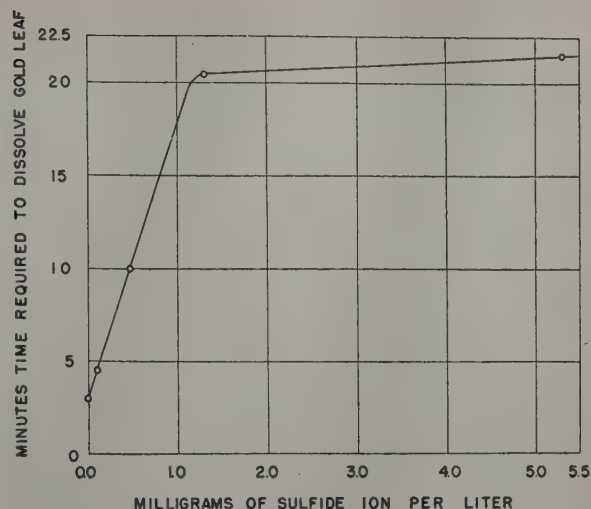


Fig. 1—Effect of sulphide ion on time required to dissolve gold leaf in a solution containing 1 g of sodium cyanide per liter.

compounds on the dissolution of gold. Beyers<sup>2</sup> found that lead compounds render gold passive when the potassium cyanide concentration is below 0.08 pct. And Leaver, Woolf, and Jackson<sup>11</sup> have reported that lead salts are very deleterious in lime-sodium cyanide solvents. The present study soon disclosed that these earlier investigators apparently overlooked the true function of the lead salts and the best operating conditions.

**Mercury Salts:** The use of 5 to 10 mg per liter of mercuric salts as "oxidizing agents" in the cyanide process was patented in 1934 by Dowsett and John.<sup>12</sup> Keith (cited by Beyers<sup>2</sup>) found that mercury salts to the extent of 50 mg of mercury per liter of solution increased the rate of dissolution of gold. Keith (1890) believed that the mercury acted as an oxygen carrier by being redissolved continuously by the cyanide solution and precipitated by the gold.

**Bismuth and Thallium Salts:** No one has shown that these salts will accelerate the dissolution of gold under any conditions.

Recorded below are observations on the behavior of various metal salts on the time required to dissolve gold leaf. The results indicate that there are several metal salts, in concentrations of 1 to 40 mg per liter, which decrease the time required for cyanide solutions to dissolve gold leaf down to 0.5 to 0.032 of the time required in the absence of these metal salts.

**Effect of Lead Nitrate Concentration:** A solution was prepared which contained, per liter, 1.0 g of sodium cyanide, 1.06 g of sodium carbonate, and 0.84 g of sodium bicarbonate. The pH of this strongly buffered\* solution is 9.9.<sup>13</sup> The results are

\* The Elsner equation<sup>14</sup> for the dissolution of gold in sodium cyanide is:  $4\text{Au} + 2\text{H}_2\text{O} + \text{O}_2 + 8\text{KCN} = 4\text{KAu}(\text{CN})_2 + 4\text{KOH}$ . Consequently, the pH of an unbuffered cyanide solution dissolving gold does not remain constant. The sodium carbonates have no appreciable effect on the rate of dissolution of gold.<sup>10</sup>

plotted in fig. 2.

Referring to fig. 2, 0.1 mg of lead nitrate was used per liter of cyanide solution, and the time required to dissolve the gold leaf was reduced more than 30 pct. In the 5 ml of cyanide solution used in the test, there was 0.0005 mg of lead nitrate. Accordingly, 0.0005 mg of lead nitrate will accelerate the dissolution of 0.051 mg, or over 100 times its

Table I. Effect of Lead Nitrate on the Dissolution Velocity of Gold Leaf in Saturated Calcium Hydroxide Solutions of Sodium Cyanide

Mg of lead nitrate per liter of solution .....	500	100	10	0
Seconds time required to dissolve gold leaf .....	1,800	1,260	800	800

Table II. Effect of Cyanide Concentration on the Dissolution of Gold Leaf in Sodium Carbonate-Lead Nitrate Solutions

Composition of Solution	Seconds Time Required to Dissolve Gold Leaf
0.2 g NaCN per liter .....	220
0.2 g NaCN + 10 mg of Pb (NO <sub>3</sub> ) <sub>2</sub> per liter .....	130
0.5 g NaCN per liter .....	210
0.5 g NaCN + 10 mg of Pb (NO <sub>3</sub> ) <sub>2</sub> per liter .....	100
1.0 g NaCN per liter .....	200
1.0 g NaCN + 10 mg of Pb (NO <sub>3</sub> ) <sub>2</sub> per liter .....	50

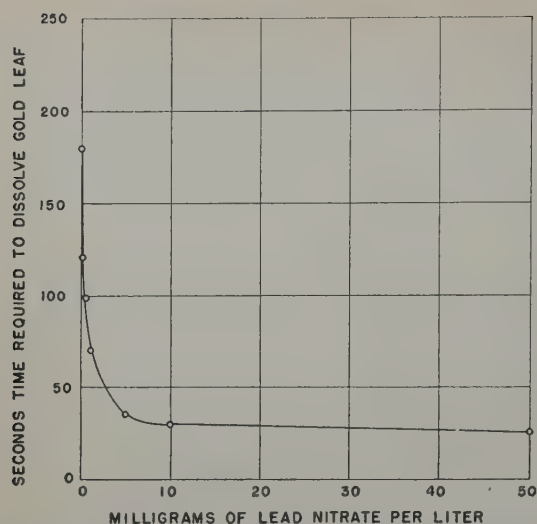


Fig. 2—Effect of lead nitrate on time required to dissolve gold leaf in sodium carbonate solutions containing 1 g of sodium cyanide per liter.

weight of gold. The observed acceleration cannot be attributed to the oxidizing power of the lead ion.

**Effects of Calcium Hydroxide and of Sodium Carbonate on the Accelerating Effect of Lead Nitrate:** A solution of 1.0 g of sodium cyanide in one liter of saturated limewater was prepared. The pH of this solution was about 12.7.<sup>13</sup> The results obtained when lead nitrate was added to this saturated solution of calcium hydroxide are shown in table I.

It is evident that in limewater solutions the addition of lead nitrate does not accelerate the dissolution of gold leaf. Quite the contrary, the lead compound increases the inhibiting effect<sup>1</sup> of calcium hydroxide. This is in accord with the reports of Leaver, Woolf, and Jackson<sup>11</sup> concerning the deleterious effect of soluble lead compounds in lime solutions.

The effect of increasing the sodium cyanide concentration in a limewater solution containing 10 mg of lead nitrate per liter was tested next. Referring to fig. 3, it is evident that the higher the cyanide concentration, the faster the gold leaf dissolves.

**Effect of Sodium Cyanide Concentration on Dissolution of Gold Leaf in Sodium Carbonate Solutions:** Tests were made next with a solution of somewhat lower pH (table II). All solutions were 0.02 mol (2.12 g per liter) with respect to sodium carbonate, and to alternate tests 10 mg of lead nitrate were added per liter of solution. The pH of 0.02 mol sodium carbonate is approximately 11.5.<sup>13</sup>

Table II shows that when lead nitrate is absent, change in the sodium cyanide concentration has little effect on the dissolution of gold leaf, and this is in accord with experiments of MacLaurin.<sup>15</sup> In the presence of lead nitrate, however, the sodium cyanide concentration has a very marked effect on the time required to dissolve gold leaf.

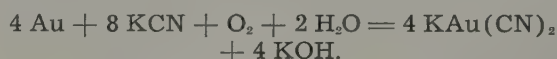
**Effect of the pH of the Cyanide Solution:** Although the accelerating effect of lead compounds had been definitely established in the case of sodium carbonate-sodium cyanide solutions, the results with the limewater cyanide solutions (fig. 3) required further study to account for the absence of the accelerating effect due to lead. It may be calculated that the solutions had a pH ranging between 12.3 and 12.6<sup>13</sup> and it was suspected that the high pH was to blame. Tests with sodium hydroxide in

place of limewater supported this hypothesis—lead nitrate had no accelerating effect in sodium hydroxide-sodium cyanide solutions. An upper limit in pH of about 12 was established beyond which there was no acceleration in the dissolution of gold leaf in solutions containing 1.0 g of sodium cyanide per liter. The more dilute (less than 2 g per liter) solutions of sodium cyanide are the only solutions which are of interest from an industrial viewpoint.

With this upper limit of a pH of 12 in mind, a series of experiments with strongly buffered cyanide solutions of lower pH were made, the values being estimated from tables compiled by Kolthoff and Rosenblum.<sup>13</sup> That the buffering agents used have no effect on the dissolution of gold has been shown by other workers<sup>1, 2, 9</sup> and has been confirmed by applying the gold leaf solution method. The use of buffering agents was chosen in preference to other methods for three reasons:

1. Measurements with the antimony electrode are vitiated by the presence of traces of metals which the antimony is capable of displacing. As little as 0.1 mg of copper per liter, for example, may be a source of great error.<sup>16</sup>

2. In the absence of buffering agents, the pH of a cyanide solution dissolving gold rises as the gold dissolves, in accordance with the Elsner equation:



3. Because of the high pH of our solutions and the presence of sodium salts, the glass electrode method was not believed applicable for our purpose.

Table III indicates that lead is effective over the pH range of 8.0 to more than 9.9. Without lead nitrate additions, as recorded in table II, the dissolution time was about 200 sec.

Table III indicates that the effective pH range for mercury is 7.6 to more than 14. In other tests, not shown, it was found that the effective concentration of mercury, as mercuric chloride, is from less than 0.1 mg to more than 50 mg per liter.

Salts of bismuth and thallium were likewise found to have a marked accelerating effect. The results of table IV were obtained with a solution containing 1.0 g of sodium cyanide per liter.

Table IV indicates that bismuth and thallium compounds accelerate the dissolution of gold.

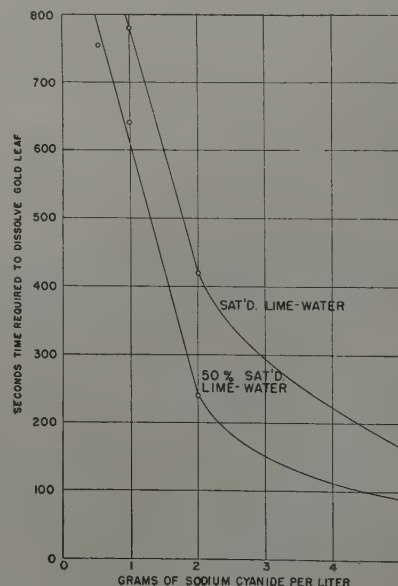


Fig. 3—Effect of sodium cyanide concentration on time required to dissolve gold leaf in lime water containing 10 mg of lead nitrate per liter.



**Table III. Dissolution of Gold Leaf in Buffered Cyanide Containing Lead Nitrate and Mercuric Chloride**

Concentrations in Grams per Liter, of Salts in Buffered Solutions Containing 1.0 G of Sodium Cyanide per Liter	Estimated pH	Seconds Time Required to Dissolve Gold Leaf in the Presence of	
		10 Mg of Pb(NO <sub>3</sub> ) <sub>2</sub> per Liter	10 Mg of HgCl <sub>2</sub> per Liter
40 g NaOH	14	200	80
10.6 g Na <sub>2</sub> CO <sub>3</sub> + 8.4 g NaHCO <sub>3</sub>	9.9	55	80
3.71 g H <sub>3</sub> BO <sub>3</sub> + 2.97 g Na <sub>2</sub> CO <sub>3</sub>	9.0	85	60
4.95 g H <sub>3</sub> BO <sub>3</sub> + 1.06 g Na <sub>2</sub> CO <sub>3</sub>	8.5	140	140
12.4 g H <sub>3</sub> BO <sub>3</sub> + 1.06 g Na <sub>2</sub> CO <sub>3</sub>	8.0	130	140
21.4 g Na <sub>2</sub> HPO <sub>4</sub> + 2.76 g KH <sub>2</sub> PO <sub>4</sub>	7.6	270	90

Many other metal salts, including those of zinc, cadmium, copper, tin, and magnesium, have been investigated. The salts of only four metals, lead, bismuth, mercury, and thallium, have been found effective.

### Discussion of Results

Calculations of electrode potentials in cyanide solutions<sup>9</sup> show that gold metal can actually displace the ions of lead, bismuth, mercury, and thallium under all conditions that might obtain in cyanide practice. Of the common metals, only these four metals are more noble than gold in cyanide solutions. Lead ion is most readily displaced by gold in the pH range of 8 to 12; at higher and lower pH, the electrode potential of lead approaches that of gold in cyanide solutions.<sup>9</sup> At pH of less than 8, the cyanide ion concentration becomes very small; at high pH, the lead ion concentration decreases. Perhaps gold dissolves more rapidly because its surface character is altered by alloying with the displaced metals. Putnam<sup>17</sup> showed that zinc would alloy with displaced copper at room temperature, and the diffusion of gold into lead at room temperature is well known. The obvious voltaic couple explanation (i.e., the action of copper in promoting the solution of zinc in acid) is not believed applicable in this case. If diffusion of oxygen to the gold-solution interface, not the kinetics of the reaction, were the controlling factor, a small deposit of metal would not alter the rate of solution, as is proved by the film theory of chemical engineering.<sup>18</sup> The kinetics of the surface reaction probably control, and the displaced metals probably cause, acceleration of the rate-controlling reaction at the gold-solution interface.

### Conclusions

1. Sulphide ion, even at concentrations below 0.5 mg per liter, may have a definite retarding effect on the dissolution of gold leaf, and this heretofore unknown deleterious action of minute traces (less than 1 mg per liter) of sulphide ion cannot be accounted for by either the oxygen-depletion or thiocyanate-formation theories. It is believed that the formation of an insoluble aurous sulphide film on the surface of the gold is responsible for the behavior of sulphide ion in protecting the gold from corrosion.

2. Salts of the four metals, lead, bismuth, mercury, and thallium, were found to decrease greatly the time required for sodium cyanide solutions to dissolve gold leaf. In some instances, less than 1 ppm (1 mg per liter) of metal salt had a definite effect.

**Table IV. Acceleration of Dissolution of Gold with Salts of Bismuth and Thallium**

Mg of Salt Added per Liter of Solution	Seconds Time Required to Dissolve Gold Leaf, Sec	
	Satd. Ca(OH) <sub>2</sub>	0.2 Mol Na <sub>2</sub> CO <sub>3</sub>
No salts of bismuth or thallium	780	210
TiCl <sub>3</sub> , 40 mg per liter	90	150
TiCl <sub>3</sub> , 300 mg per liter	60	45
TiCl <sub>3</sub> , 40 mg per liter	60	45
TiCl <sub>3</sub> , 300 mg per liter	25	45
Bi(NO <sub>3</sub> ) <sub>3</sub> , 30 mg per liter	50	90
Bi(NO <sub>3</sub> ) <sub>3</sub> , 300 mg per liter	50	90

3. The beneficial effect is apparently limited to these four metals.

4. In the case of lead nitrate, increase in the sodium cyanide concentration above 0.2 g per liter increases the rate of dissolution of gold leaf in calcium hydroxide and in sodium carbonate solutions. The optimum pH range is about 8 to 11.5.

5. A lower limiting concentration of contact metal ion is necessary to bring about the accelerating effect. Thus, if the pH of the cyanide solution is above about 11.5, the lead ion concentration is too low to be effective. Similarly, the effect disappears if the cyanide ion concentration is too low.

### References

- G. Barsky, S. J. Swainson, and N. Hedley: *Trans. AIME* (1934) **112**, 660-671.
- E. J. Beyers: *Chem. Met. Mining Soc. S. Africa*. (1936) **37**, 37-39.
- S. B. Christy: *Trans. AIME* (1900) **30**, 896.
- J. V. N. Dorr: *Cyanidation and Concentration of Gold and Silver Ores*. 99-139. 1936. New York. McGraw-Hill Book Co.
- G. L. Putnam: *Eng. and Min. Jnl.* (March 1944) **145**, 70-73.
- G. L. Putnam: *Electrochemical Aspects of the Hydrometallurgy of Gold*. Ph.D. Thesis, Columbia (1942) 35-53.
- N. Munro: *Can. Chem. Process Ind.* (Sept. 1940) **24**, 438.
- R. J. Lemmon: *Chem. Eng. Min. Rev.* (Oct. 1939) **32**, 31.
- R. G. Courtney and J. H. Maguire: Thesis, Univ. Idaho; cited by A. W. Fahrenwald and J. Newton; *Eng. and Min. Jnl.* (Jan. 1939) **140**, 44-47.
- T. Haden: *Bull. Inst. Min. Met.* (1938) **401**, 16.
- E. S. Leaver, J. A. Woolf, and T. A. Jackson: *AIME Contr. No. 5* (1933).
- C. W. Dowsett and W. E. John: U.S. Pat. 1,952,976 (March 27, 1934).
- I. M. Kolthoff and C. Rosenblum: *Acid-Base Indicators*, 1937. New York. The Macmillan Co.
- L. Elsner: *Jnl. Prakt. Chem.* (1846) **37**, (1) 441.
- R. C. MacLaurin: *Jnl. Chem. Soc.* (1895) **65**, 728.
- G. A. Perley: *Trans. Electrochem. Soc.* (1939) **76**, 85.
- G. L. Putnam: *Copper-Zinc Alloy Formation at Room Temperature*. *Trans. Electrochem. Soc.* (1938) **74**, 252.
- W. H. Walker, W. K. Lewis, W. H. McAdams, and E. R. Gilliland: *Principles of Chemical Engineering*. 1937. 34-168. New York. McGraw-Hill Book Co.
- M. Faraday: *Trans. Phil. Magazine, London, Edinburgh and Dublin*. (1857) Series **4**, 14, 403.
- W. C. Gardiner and H. C. Sanders: *Ind. and Eng. Chem. Anal. Ed.* (1937) **9**, 278.
- W. M. Latimer: *The Oxidation States of the Elements and Their Potentials in Aqueous Solutions*. 185. 1938. New York. Prentice-Hall, Inc.
- C. G. Fink and G. L. Putnam: *Ind. and Eng. Chem. Anal. Ed.* (1942) **14**, 468-470.

# The Cleaning of Fine Sizes Of Bituminous Coals By Concentrating Tables

by R. E. Zimmerman

Wide attention is being placed upon various methods for cleaning the fine sizes of bituminous coals. The author describes and analyzes the results achieved on wet concentrating tables of modern design on unclassified feeds of  $\frac{1}{2}$  in. particle sizes, including tests on high-sulphur coals.

THE increased economic necessity of cleaning the small sizes of coal has forced wide attention upon various methods or processes being used. Although numerous cleaning plants in the past have been cleaning coal smaller than  $\frac{1}{2}$  or  $\frac{3}{8}$  in., the large quantities of higher ash fines being produced by mechanical mining and the need for cleaner coal under competitive market conditions have brought about many new installations of fine-coal cleaning plants.

Of various methods used, one of the more popular types is the wet concentrating table. Approximately 525 tables have been installed in the United States in the past four years of which 320 have been installed in 20, large new plants. There are now probably 1600 tables of all types in existence in this country for cleaning coal.

The purpose of this paper is to describe the cleaning of fine coal by wet concentrating tables. Its scope is confined to a discussion of the theory of tabling and some of the factors entering into tabling action, a description of the Deister SuperDuty diagonal-deck coal table and a discussion of the results of tabling various fine sizes of coals. Al-

though tables have been used for cleaning coal as coarse as 2 in. and favorable results are reported, this paper is limited to reporting on the results of cleaning coal fines from  $\frac{1}{2}$  in. down. Also, although it is well known that the performance of tables is even superior under classified or closely sized feed conditions, this paper concerns itself with the results achieved by tabling unclassified feeds.

## History of Wet Concentrating Tables

The wet concentration of minerals by some form of tabling is quite old, starting originally from fixed, smooth surfaced, tilted boards taking advantage of the laminar flow of liquid films and the separation of minerals according to specific gravity. This was followed by the development of rough deck surfaces

---

R. E. ZIMMERMAN, Member AIME, is Chief, Mineral Preparation Division, School of Mineral Industries, The Pennsylvania State College, State College, Pa.

AIME New York Meeting, February 1950.

TP 2875 F. Discussion (2 copies) may be sent to Transactions AIME before Oct. 31, 1950. Manuscript received Feb. 3, 1950.

---



induced by grooves or riffles and by the final achievement of adding a reciprocating action or differential motion to the table deck.

The development of tables occurred first in the separation of minerals other than coal, particularly in the concentration of heavy metallic minerals. There is an endless variety and, like other processes, it has required lengthy periods of evolution before achieving its final efficient form. Ore dressing men are familiar with the application of the early vanners, circular stationary Linkenbach tables, multiple round tables such as the Anaconda-Evans, and bumping or shaking tables such as the Rittinger or the Gilpin County concentrator.<sup>1</sup>

The advantage of some form of shaking motion was visualized early with the imparting of a reciprocating or shaking action by means of eccentrics or cams and by supporting the table deck on stilts or from hangers. The advantage of a differential action was found. This was first induced by a bumping action against a "bumping block" and later by a differential head motion applied through various designs by means of pitman and toggle actuated through an eccentric or crank shaft.

Some of the later designs of concentrating tables used in ore dressing are those of Garfield, Butchart, Deister-Overstrom, and Wilfley.<sup>2</sup> The first record of tables being used for coal in the United States appears to be of the bumping type called the Campbell Bumping Table. It was described by Phillips in 1893 when he wrote of the installation at Earlington, Ky.<sup>3</sup>

Campbell Bumping Tables have been used for many years at the Rosedale Washery, Johnstown, Pa., and their use has been described by Richardson.<sup>4</sup>

Practically all tables now used for coal cleaning use a driving mechanism, producing a differential motion, a rectangular or rhombohedral shaped deck, and some form of riffling on the deck surface. Possibly 95 pct of all tables being used for wet washing of coal in the United States are either the Deister SuperDuty diagonal-deck type or the Plat-0 type. The former is the one most commonly used.<sup>5</sup> Until 1940 tables were more widely used on anthracite coals, but, with the current upswing of their use on bituminous coals, this condition has been reversed. For the past few years, ratio of placements have been approximately 3 to 1 in favor of bituminous coals.

### Theory of Tabling

A great deal has been written concerning the theory of tabling and the various physical laws which enter into its operation.<sup>2, 5-8</sup> Much more work has yet to be done in studying thoroughly the numerous variables that can be made in tabling and the interrelation of the physical laws that affect separation.

Gaudin's<sup>9</sup> mathematical analysis of flowing film concentration is of fundamental value and a worthy

contribution. His analysis pertains mostly to smooth deck surfaces and does not go into detail concerning the phenomena surrounding the use of riffles. Bird and Davis,<sup>6</sup> Yancey,<sup>7</sup> and Gandrud<sup>8</sup> have made valuable contributions to the study of the various factors affecting tabling of coal.

Separation of particles on a table deck is not merely the result of their differences in specific gravity, but rather there are numerous interrelated phenomena. A decided advantage of tabling over

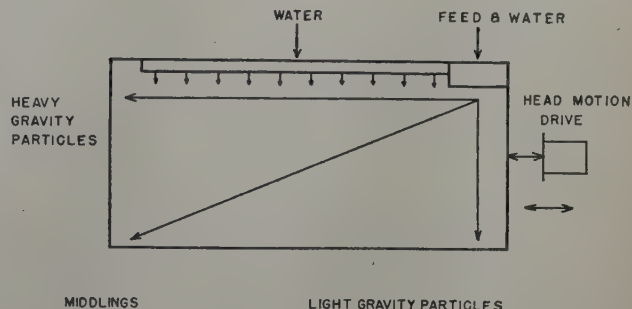


Fig. 1—Natural path of particles on a table.

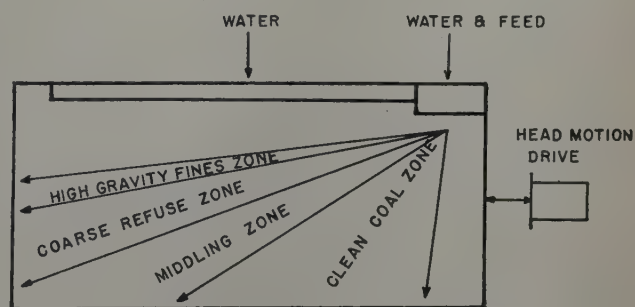


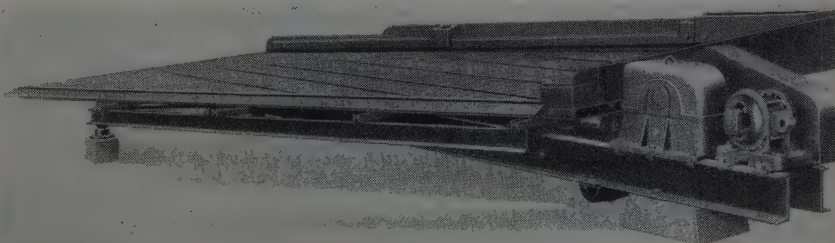
Fig. 2—Zones of separation on a table.

some other known processes is the fact that a great many variables can be made to aid and assist in effective separation.

Basically the separation of different minerals on a concentrating table are the result of the following factors: (1) Specific gravity of the particles, (2) size of particles, (3) shape of particles, (4) stratification (or consolidation trickling), and (5) hindered settling.

The effective use of these factors may be controlled and aided by the following table adjustments: (1) Differential action of the drive or head motion, (2) speed of motion, (3) length of stroke, (4) side tilt, (5) end elevation, (6) size of deck area, (7) shape of deck, (8) character of deck surface, (9) riffle design, (10) amount and distribution of

Fig. 3—Deister SuperDuty diagonal deck No. 7 coal washing table.



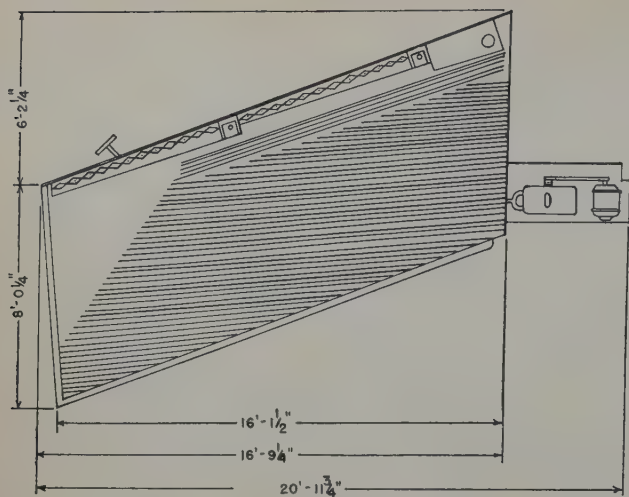


Fig. 4—Plan view of Deister table deck.

wash water, (11) feed water-solids ratio, (12) feed rate, and (13) feed uniformity.

Concentrating tables are essentially a rectangular or rhombohedral shaped deck supported by suitable frame and bearings, generally sloping in the direction of the short axis and usually tilted upwards at the refuse end of the long axis and differentially reciprocated in the direction of the long axis.

This action of head motion tends to move the particles longitudinally towards the end of the table, while at right angles to this movement wash water is distributed along the high side of the table which tends to move particles down the slope parallel to the short axis of the table. Hence, the natural path of a particle on a concentrating table would be from the feed-box corner to the diagonally opposite corner of the deck as shown in fig. 1.

There is a natural tendency for heavier particles to sink to the deck of the table and for lighter particles to stay nearer the surface. As a consequence, the heavier particles are carried farther along the longitudinal axis since, as they are on or near the surface of the deck, they are more affected by the reciprocating action of the table. The lighter particles, as they are at or near the top of the moving bed or particles, travel slower in a longitudinal direction and thus become separated from the heavy particles.

Wash water and the diluting water in the feed cause the movement of the lighter gravity particles down the short axis of the table. Thus the feed material fans out from the feed box, where it enters on to the table with heavy gravity particles moving in one direction and lighter particles moving in another. By proper adjustments of all the factors previously mentioned, a clear-cut separation is possible between high and low-gravity particles, or between refuse and coal.

Riffles are of great importance to tabling and are responsible for their considerably greater capacity over smooth decks and for greater accuracy of separation. Behind the riffles, particles are delayed in their downward movement along the slope of the deck. Hindered settling, consolidation trickling, and

other phenomena occur as the particles tumble over the riffles and stratification ensues. The heavier particles are settled to the deck of the table and trapped behind the riffles thus insuring their positive movement in the direction of the end of the table.

Since stratification and separation of particles are not complete as a result of any one riffle, a series of riffles are used, repeating the cycle of stratification and hindered settling from riffle to riffle, obtaining purer products as the particles fan out and progress forward and downward over the table.

The smallest heaviest particles stratify themselves downward to the surface of the deck more quickly and are carried out by table movement toward the refuse end at a faster rate than coarse heavy particles. Light-gravity larger pieces ride on the top layer of particles and flow on down the slope of the deck as a result of the cross flow of wash water at right angles to the shaking movement of the table. Heavy large particles settle down next to the fine heavy particles and are carried behind the riffles towards the longitudinal end but do not ride as high up the table as the heavy fine particles.

There is a tendency for the fine or small size light-gravity particles to lie between the coarse heavy particles and work over into the zone with the coarse refuse, and, if the only separating action on a table were that of straight stratification, this would result in fine coal getting over into the refuse. Fortunately, however, hydraulic classification occurs under hindered settling conditions which tends to counteract this and by proper use of water dilution and wash water as well as riffle design the small light particles of coal are washed off with coarse light particles or coarse coal.<sup>6, 10</sup>

When there is a definite clear-cut gravity difference between coal and refuse, a wide size range or an unclassified feed may be treated very efficiently. When there is a large proportion of material intermediate in gravity between coal and refuse, the separation of an unsized feed will become increas-

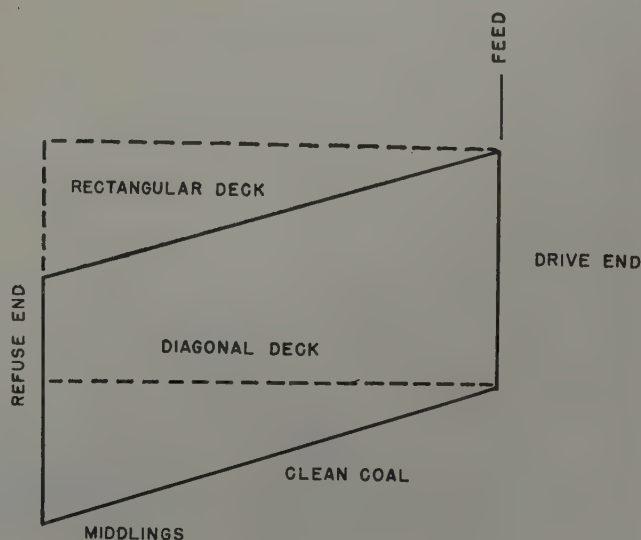
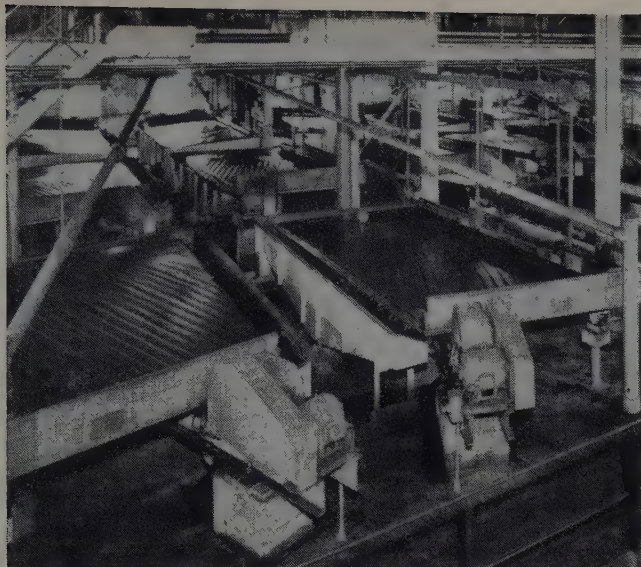


Fig. 5—Comparison of diagonal deck with rectangular deck.





**Fig. 6—Installation of Deister No. 7 coal washing tables in a modern coal washing plant.**

ingly difficult. Fig. 2 illustrates the zone where this material will fall.

In only rare instances, on coals of very difficult cleaning characteristics, is it necessary to cut a middling or a third product from a table. When it is done, these middlings are normally disposed of as a secondary fuel since little can be accomplished by retableting such material.

The advantage of a classified feed where a large proportion of middlings or near gravity material exists is that the danger of having small light-gravity particles of coal get over into the refuse or middling zone vanishes. Sharper lines of demarcation can be made, which result in very high efficiencies. B. M. Bird and H. F. Yancey<sup>8, 9, 11, 12</sup> have demonstrated many times the advantages of classified feed to the tables where coal is difficult to treat because of high percentages of near gravity material or where middlings from another cleaning process are to be cleaned on tables.

For most coals, however, it has been demonstrated that under proper conditions of operation an unclassified feed may be used for cleaning sizes ranging from  $\frac{1}{2}$  in. to 0, and produce products containing relatively few misplaced particles. On the extreme fines, separation may be effective on particles as small as 200-mesh with reasonably efficient recoveries on the 100x200-mesh sizes.

Brief mention should be made of the effect of particle shape in tabling. Generally speaking, flat particles have a tendency to pass over with the refuse. Flat particles adhere to the table and have a tendency to move along with heavier gravity particles. Fortunately, in most instances coal particles are cubical or splintery and many refuse particles are naturally flat in shape. H. F. Yancey,<sup>8</sup> who conducted an investigation on the effect of particles shape, states, however, that "shape of particle is a factor of minor importance in tabling of most unsized coals insofar as the overall efficiency of the process is concerned."

An unmistakable advantage of a concentrating table which should be mentioned is that the pro-

ducts being made are always visible. The operator can observe his refuse, clean coal, or middlings and can note quickly the effect of any change in feed conditions or table adjustments.

### **Description of the Deister Diagonal-deck Table**

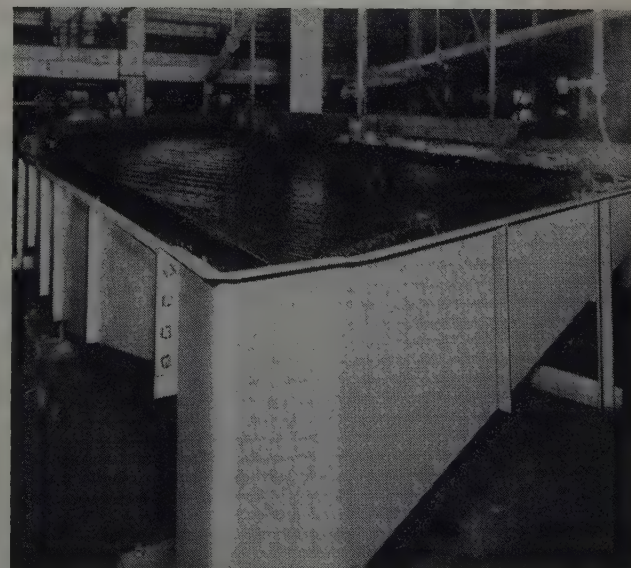
Although there are other wet concentrating tables in use, the writer's experience and test performance data concerns the Deister SuperDuty diagonal-deck table and a brief description of this particular table is in order.

The current Deister table was developed from the widely used Deister-Overstrom diagonal-deck table. Improvements in design included greater ruggedness and Concenco Anti Friction Bearing Headmotion. This motion is similar in principle to the Overstrom, being of the pitman and toggle design, but it is of stronger construction with ball and roller bearings lubricated by a circulating oil system. The table most often used for coal cleaning is their No. 7, SuperDuty size, either linoleum or rubber covered and with wooden or rubber riffles. The bulk of the recent tables are rubber covered with rubber riffles.

Fig. 3 shows a photograph of the No. 7, coal washing table. It can be made either right hand or left hand to suit conditions in the plant. The head motion can be operated from a line shaft with belt and pulley, but invariably new installations use the individual drives. The No. 7 table utilizes a 3-hp motor, although power consumption is seldom over 2 hp.

Fig. 4 shows the general dimensions and plan view of the coal washing table with one type of riffle pattern. Other riffle designs can be used. The most common are the line of motion type in which all the riffles are in a diagonal line parallel to the motion of the table.

Riffing is characterized by a large number of shallow riffles, interspersed with higher strips, which provide pools to settle finer sized particles. All riffles are high at the feed end and taper or feather



**Fig. 7—Closeup view of Deister table showing launder arrangements for handling products.**



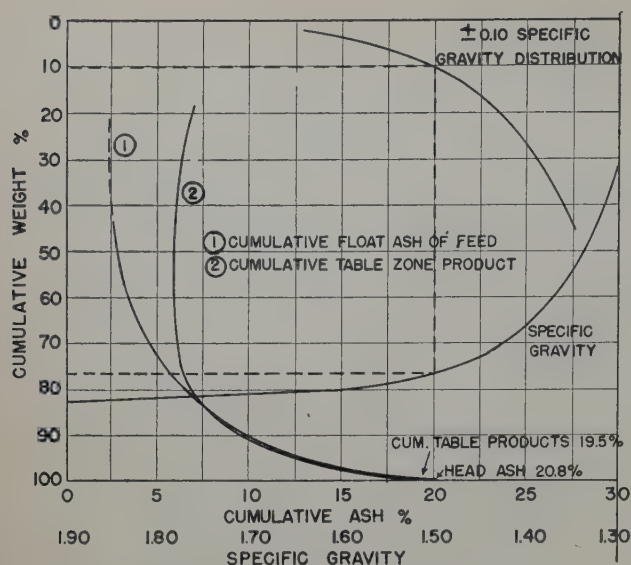


Fig. 8—Case study "A" washability of  $\frac{1}{2}$  in. x 0 table feed and cumulative table products.

towards the refuse end. The actual height of the riffles varies, depending on the sizes of coal being cleaned. The highest may reach 2 in. at the deep end.

Feed mixed with water is fed into the feed box in the upper right hand corner of the table illustrated in fig. 4. In addition to the water with the feed, water is also added at two points along the top edge of the table, where it runs through troughs equipped with adjustable louvers, permitting water to be distributed uniformly or in variable quantities along the length of the deck.

The table base is a pair of heavy parallel channels running diagonally to the opposite corner from the drive with the head motion and rocker arm mounted at one end on an extension of the channels. Another rocker and a helical compression spring are mounted at the other end of the channels. A subframe constructed of lighter channels and beams supports the table deck. This subframe straddles the upper edges of the main channels resting on saddles at four points along its diagonal line, and the two outer corners rest on blocks of which the elevations are adjustable by oppositely facing wedges manipulated by hand wheel and spur gearings. Maximum tilt on the No. 7 table is approximately  $8^\circ$ . The tilting mechanism has an operating range of  $3^\circ$ , which may be employed in any portion of the overall range. End or longitudinal tilt is obtained by shimming up the main channel supports at the spring or refuse end.

The table deck is built of cypress in strips laid diagonally to its framework. The deck is covered with either linoleum or rubber and riffles made of pine or hardwood tacked on, or if rubber riffles are used, glued on by a special rubber cement.

The speed of the Concencho head motion may be varied from 250 to 300 rpm depending upon the type of coal being washed. Stroke length is also adjustable, generally between the range of 7/16 to  $1\frac{1}{4}$  in.

End tilt or end elevation of the table may be anywhere from 0 to 12 in. For  $\frac{1}{4}$  to 0 coal one seldom operates higher than 3 in. The coarser the

coal, the greater the end elevation. The side tilt, or slope, is adjusted by turning the hand wheel that operates the subframe tilting mechanism. One seldom operates at a greater slope than  $6^\circ$ .

The amount of water required will vary with particle size and other characteristics of the coal and refuse. In most cases, feed is at approximately 40 pct solids, and dressing water is added to bring the overall water to solids ratio to 2 to 1, by weight.

As the name implies, the deck dimensions are diagonal or rhombohedral in shape. This has advantages over the ordinary, rectangular shape. Fig. 5 illustrates this point. Generally speaking, the area of a table actually occupied by the coal and refuse takes on the form of a diagonal band from the feed to the refuse or high-gravity end. This is illustrated in fig. 1, where the lines of force are shown, and in fig. 2, where the zones of various products are shown.

Since the material tends to distribute in a diagonal direction, a table of the diagonal shape illustrated in fig. 5 more effectively uses deck area and can utilize more riffles, a desirable characteristic for higher tonnages, unclassified feed, and where there is a great deal of near gravity or middling material to be separated.

Fig. 6 is a photograph of one of the more recent installations of Deister coal washing tables. At this plant they are cleaning  $\frac{1}{4}$  in. x 0 bituminous coal. Fig. 7 is a close-up photograph of one of the tables and shows launder arrangement for discharging products.

The capacity of the No. 7 coal washing diagonal-deck table will vary with the type of feed, being greater for coarser particles (up to the practical size that can be treated) than for extreme fines, greater for classified feeds than for unclassified, and, as in all processes, greater for coals containing a minimum of near gravity material.

Table I. Screen Sizes, Ash<sup>a</sup> and Sulphur<sup>a</sup> of  $\frac{1}{2}$  in. x 0 Feed to Deister Table

Size, Mesh	Direct Analysis, Pct			Cumulative Analysis, Pct					
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
+ $\frac{1}{2}$ in. rd.	19.2	26.3	1.39	19.2	26.3	1.39	100.0	20.8	1.50
$\frac{3}{8}$ x4	21.5	20.7	1.43	40.7	23.3	1.41	80.8	19.5	1.53
4x8	21.9	20.7	1.41	62.6	22.4	1.41	59.3	19.1	1.57
8x14	15.2	18.8	1.51	77.8	21.7	1.43	37.4	18.2	1.66
14x28	8.6	17.2	1.54	86.4	21.3	1.44	22.2	17.8	1.76
28x48	6.0	17.2	1.70	92.4	21.0	1.46	13.6	18.2	1.91
48x100	3.2	16.9	1.98	95.6	20.9	1.48	7.6	19.0	2.07
—100	4.4	20.6	2.13	100.0	20.8	1.50	4.4	20.6	2.13
Total	100.0	20.8	1.50						

<sup>a</sup> Analyses on "Dry Basis".

Table II. Case Study "A" Zone Analysis of Table Products from  $\frac{1}{2}$  in. x 0 Feed

Zone	Direct, Pct			Cumulative Recovery, Pct			Cumulative Reject, Pct		
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
1	23.4	6.6	1.05	23.4	6.6	1.05	100.0	19.5	1.50
2	16.5	5.2	1.14	39.9	6.0	1.09	76.6	23.5	1.64
3	23.9	5.8	1.29	63.8	5.9	1.16	60.1	28.5	1.78
4	15.4	8.8	1.61	79.2	6.5	1.25	36.2	43.5	2.10
5	2.0	16.4	2.43	81.2	6.7	1.28	20.8	69.2	2.47
7	1.9	41.6	2.95	83.1	7.5	1.32	18.8	74.8	2.48
8	16.7	78.5	2.36	99.8	19.4	1.49	16.9	78.5	2.42
9	0.2	79.6	7.59	100.0	19.5	1.50	0.2	79.6	7.59
Total	100.0	19.5	1.50						



On average Western Pennsylvania bituminous coals, capacities of 9 to 10 tph per table for ¼ in. x 0 size feed can very well be attained with excellent efficiencies. By increasing the maximum size to ⅜ in., the feed rate can be raised to 12 tph. With a classified feed, that is where there is a close size range of particles, the table capacity may be sharply increased to as much as 15 to 25 tph or higher. B. M. Bird and H. F. Yancey state that "classification (of table feed) approximately doubles the capacity of the table while permitting efficiencies equal to those of unsized feeds."<sup>11</sup>

## Results of Test Data on Unclassified Table Feeds

To study the results of cleaning the fine sizes of bituminous coal, particularly under unclassified feed conditions, numerous tests have been made under a wide variety of conditions on many different coals. To illustrate the performance of the Deister Super-Duty diagonal-deck coal washing table, test data on four different coals are presented for analysis.

Since interest is also general concerning the effect of tabling coal of different size consists, the four case studies presented are tests of a wide variation of sizes. Examination of the washability curves and ash and sulphur analyses also show them to be considerably different from the viewpoint of their natural characteristics. For the sake of brevity, the numerous operating variables, such as table speed, stroke, side tilt, end elevation, solids-water ratio, etc. are not included in this paper. In general, it can be assumed that table settings were under optimum conditions.

### Case Study "A"

The data shown under this study come from the testing of ½ in. x 0 Pittsburgh seam coal from Washington County, Pa. The table feed was obtained as a result of screening the run-of-mine coal on ½-in. round hole screens.

Zone samples were taken of the products from the table by marking off linear distances along the table, dividing it into 10 zones, with zones 1 through 5 taken along the coal side and zones 6 through 10 along the refuse end. In this particular test no particles came over in zones 9 and 10, which are along the upper side of the refuse end.

In sampling the zonal products, samples were taken simultaneously. A special zone sample cutter was arranged so that all the products from each zone were collected, each container being large enough to hold all the material passing into it. Each

Table III. Float-and-Sink at 1.50 Sp Gr on Table Products (½ in. x 0), Case Study "A"

Zone	Prod-uct, Wt, Pct	Float, Pct			Sink, Pct			Cum. Wt Pct Sink in Product
		Wt	Ash	Sul.	Wt.	Ash	Sul.	
1	23.4	92.9	5.0	1.02	7.1	27.7	1.47	7.1
2	16.5	98.7	4.7	1.11	1.3	36.7	2.84	4.7
3	23.9	98.7	5.4	1.26	1.3	35.6	3.46	3.4
4	15.4	91.1	6.2	1.40	8.9	36.3	3.76	4.4
5	2.0	72.9	7.9	1.77	27.1	39.2	4.29	5.0
6	1.9	30.7	7.6	1.63	69.3	56.7	3.54	5.4
7	16.7	7.7	6.5	1.44	92.3	84.5	2.44	20.9
8	0.2							a
Total	100.0							

<sup>a</sup> Not enough sample in zone 8 for analysis at 1.50.

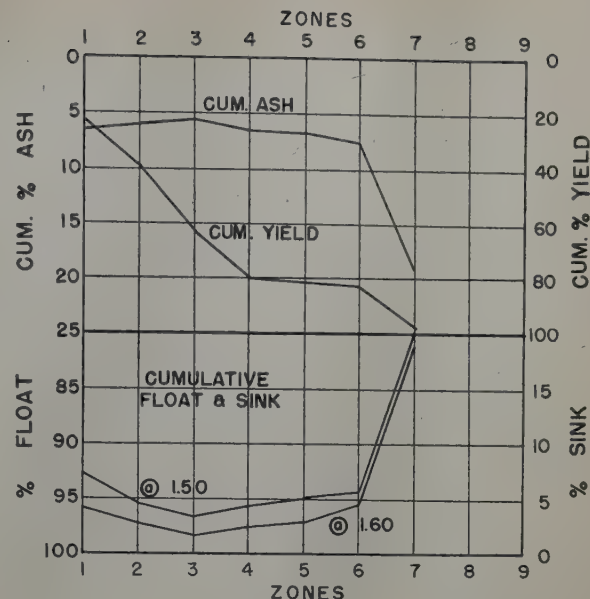


Fig. 9—Case study "A" ½ in. x 0 feed showing cumulative ash, yield, float-and-sink byproduct zones.

zone's product was then filtered, dried, and weighed. Table capacity calculated as 10.5 tph.

Table I shows the size distribution of the feed together with the ash and sulphur analyses of each size increment. Analyses are on a "dry" basis; sieve sizes are Tyler standard. One notes that this coal runs 20.8 pct ash and 1.50 pct total sulphur.

It is a relatively coarse feed containing 19.2 pct of ½x⅜ in. and with 77.8 pct as +14 mesh. The +⅜ in. is the highest in ash, being 26.3 pct and the lowest in sulphur content, 1.39 pct. The middle sizes are the lowest in ash percent.

Fig. 8, shows the washability curve of this feed, its ±0.10 sp gr distribution, as well as a curve (2) indicating the cumulative ash yield of the zonal products from the table. One could term this coal a moderately difficult coal to clean. The specific gravity distribution curve indicates that any washing below 1.50 sp gr would be difficult, the amount of near gravity material below 1.50 being quite high.

Assuming the lower practical gravity of separation as being 1.50, such a separation would give a theoretical ash in the clean coal product of 5.5 pct with a yield of 76.5 pct. At this same yield point the clean coal product of the table analyzed 6.4 pct. Also one observes that at 1.60 sp gr the cumulative zonal

Table IV. Float-and-Sink at 1.60 Sp Gr on Table Products (½ in. x 0), Case Study "A"

Zone	Prod-uct, Wt, Pct	Float, Pct			Sink, Pct			Cum. Wt Pct Sink in Product
		Wt	Ash	Sul.	Wt	Ash	Sul.	
1	23.4	96.0	5.5	1.02	4.0	34.1	1.76	4.0
2	16.5	99.2	4.8	1.11	0.8	44.9	3.56	2.7
3	23.9	99.5	5.6	1.27	0.5	48.3	4.45	1.8
4	15.4	95.0	7.1	1.49	5.0	42.6	3.91	2.4
5	2.0	79.8	9.5	1.96	20.2	43.6	4.29	2.8
6	1.9	36.9	10.9	1.99	63.1	59.6	3.52	4.2
7	16.7	8.3	8.1	1.62	91.7	84.9	2.43	19.0
8	0.2							a
Total	100.0							

<sup>a</sup> Not enough sample in zone 8 for analysis at 1.60 shows 97.5 pct sink at 1.90.

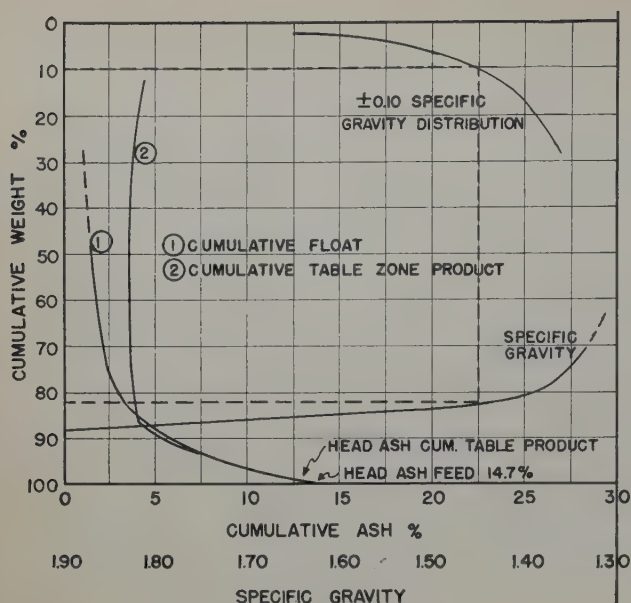


Fig. 10—Case study "B" washability of ¼ in. x 0 table feed.

curve approximates the theoretical curve almost perfectly.

Table II gives the analyses of the table products by zones. The zone 1 product is somewhat higher ash than the next two consecutive zones. This is characteristic, as this first zone near the feed end contains some high-ash fines carried over by the feed water.

The first five zones produced 81.2 pct of the product and ran 6.7 pct ash. The refuse, that is, the products from zones 6, 7 and 8, ran 74.8 pct ash.

One realizes, of course, that the entire product on the long side of the table, generally called the "coal" side, does not have to be sluiced to clean coal. The products from the zones near the refuse can be directed to refuse or separated out as a third product.

Although sulphur was not considered the major problem in this test, it is noted that the cumulative sulphur of zones 1 through 5 ran 1.28 pct, that from zones 1 through 3 ran 1.16 pct. The inherent sulphur of this particular coal is approximately 0.95 pct.

Tables III and IV show the results of float-and-sink analyses at 1.50 and 1.60 sp gr, respectively, of the various product zones. At 1.50 sp gr which, as has been pointed out previously, is the lower limit for practical washing, the first five product zones contain 5.0 pct sink. At 1.60 this is reduced to 2.8 pct.

Table V. Screen Size and Ash Analysis of ¼ in. x 0 Feed to Deister Table, Case Study "B"

Size, Mesh	Direct, Pct		Cumulative, Pct			
	Wt	Ash	Wt	Ash	Wt	Ash
+4	6.9	26.0	6.9	26.0	100.0	14.7
4x8	23.5	18.5	32.8	20.1	93.1	13.9
8x14	18.0	13.0	50.8	17.6	67.2	12.1
14x35	26.7	11.5	77.5	15.5	49.2	11.8
35x100	13.9	11.1	91.4	14.8	22.5	12.2
—100	8.6	13.9	100.0	14.7	8.6	13.9
Total	100.0	14.7				

Fig. 9 shows graphically the cumulative ash percent, yield percent and float-and-sink at 1.50 and 1.60 sp gr for the products from the various zones along the table discharge, beginning with zone 1 and cumulating the products up through zone 8.

### Case Study "B"

The data shown under this study are the results of testing ¼ in. x 0 screened run-of-mine from the Pratt seam, taken from a mine near Birmingham, Ala.

Zone samples were taken along the table as in study "A", except that the long or coal side was divided into six zones and the refuse end into three with zone 7 at the diagonal corner. Feed rate of the table at the time samples were taken was 10.0 tph.

Table V gives the size distribution of the feed together with the ash analyses of the size increments. This coal is finer and lower in ash than the coal of the previous study. Only 6.9 pct of it is +4-mesh, while 8.6 pct is —100-mesh. The calculated head ash of this coal is 14.7 pct. The +4-mesh is by far the highest ash increment, running 26.0 pct. The —100-mesh size contains only 13.9 pct ash.

Fig. 10 shows the washability curve of this Pratt seam ¼ in. x 0 table feed, the ±0.10 sp gr distribution, as well as a curve (2) indicating the cumulative ash yield of the products from the zone samples mentioned above.

Table VI. Case Study "B" Zone Analysis of Table Products from ¼ in. x 0 Feed

Zone			Cumulative Recovery, Pct		Cumulative Reject, Pct	
	Wt, Pct	Ash, Pct	Wt	Ash	Wt	Ash
1	11.8	4.6	11.8	4.6	100.0	13.8
2	40.6	3.4	52.4	3.7	88.2	15.1
3	27.1	4.7	79.5	4.0	47.6	25.0
4	4.0	8.6	83.5	4.3	20.5	51.9
5	2.6	13.1	86.1	4.5	16.5	62.4
6	0.7	17.1	86.8	4.6	13.9	71.6
7	0.6	30.3	87.4	4.8	13.2	74.5
8	9.8	74.2	97.2	11.8	12.6	76.6
9	2.8	85.0	100.0	13.8	2.8	85.0
Total	100.0	13.8				

This coal is not only a lower ash coal but compared to case study "A" is a much easier coal to clean. The ±0.10 sp gr curve shows that the lower limit for efficient cleaning is at 1.45, where a theoretical ash in the float coal is only 3.5 pct producing an 82 pct recovery. At this same yield point the cumulative zone products from the Deister table gave 3.8 pct ash, a very close check with the theoretical. This gap, of course, widens beyond this point at the lower gravities.

Table VI gives the zone analysis of table products in case study "B". As is general, the zone 1 showing 4.6 pct ash gives a somewhat higher ash percent than the next zone, which produced a 3.4 pct ash product. Zones 1 through 6 gave a cumulative ash of 4.6 pct with a combined yield of 86.8. Even adding the corner zone 7 into the clean coal side brought the overall clean coal ash only to 4.8. Refuse or reject ash combining zones 7, 8, and 9 was 74.5 pct.

In table VII, float-and-sink analyses are shown for each zone product at 1.40 and 1.60 sp gr. The combined products of zones 1 through 6 show 13.7



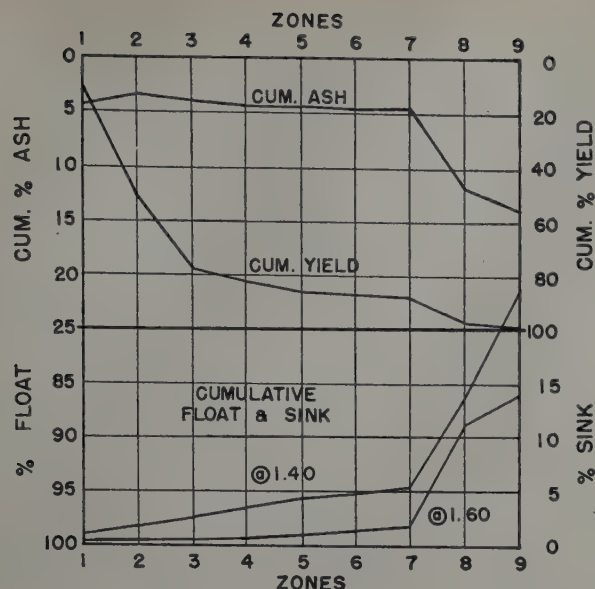


Fig. 11—Case study "B" 1/4 in. x 0 feed showing cumulative ash, yield, float-and-sink byproduct zones.

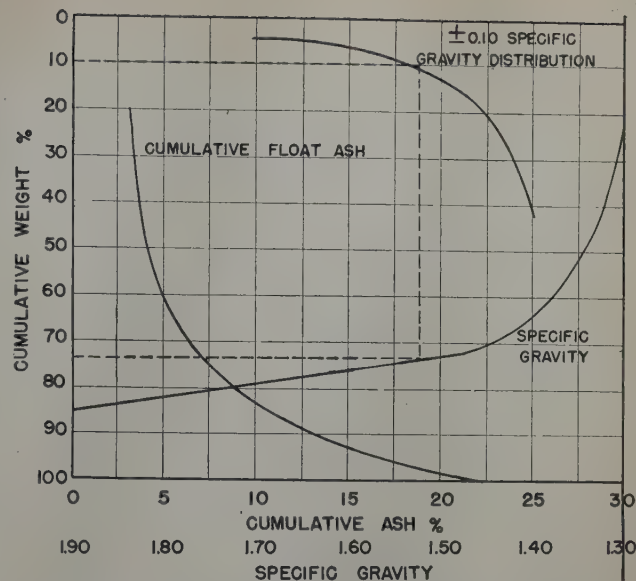


Fig. 12—Case study "C" washability 3/16 in. x 0 table feed.

pct sink at 1.40 and only 2.6 pct sink, however, from zones 1 through 4. Since 1.40 sp gr is lower than the desirable minimum gravity according to the  $\pm 0.10$  sp gr distribution curve, it is logical that one would run into difficulties near the middling zone of the table.

Testing these same products at 1.60 sp gr, less than 1.0 pct sink is observed in the cumulated products of zones 1 through 5 and, including the sixth zone, only 1.05 pct sink was found. Combining the refuse zones 7-8-9, one finds these zones contain 5.3 pct float coal at 1.40 and 6.9 pct float coal at 1.60 sp gr.

Fig. 11 shows graphically the results plotted from the accompanying test data and gives the cumulative ash and yield percent as well as the cumulative float-and-sink percent at 1.40 and 1.60 sp gr, cumulating the products from zones 1 through 9.

#### Case Study "C"

This study is on 3/16 x 0 coal from the Ohio No. 8 seam in Jefferson County of eastern Ohio. The table feed is a mixture of 80 pct total seam deep mined and 20 pct strip coal and is the resultant product of screening the cleaning plant raw feed on 3/16 in. round hole screens.

Zone samples were not taken at various points along the table. Rather, the entire product from the

long or coal side of the table was considered as the clean coal product and the material discharged off the end as refuse. Screen and ash analyses of the feed, clean coal, and refuse were taken and thus a comparison can be made of ash reduction at various size increments. Table feed rate at the time samples were taken was 8.7 tph.

Table VIII gives the size consist together with the ash analyses of the various sizes. This is a dirty coal, the calculated head ash being 22.2 pct. Unlike some of the other coals reported in this paper, the coarsest increment, namely the 3/16 in. x 8 mesh, had the lowest ash, being 14.8 pct. The —100-mesh contained 29.8 pct and the —200-mesh, 31.7 pct ash.

The table feed was considerably finer than those in studies "A" and "B", containing 10 pct —100-mesh and 5.2 pct —200-mesh particles. These extreme fines are difficult to clean by any gravity process.

Fig. 12 shows the washability curves for the table feed. The  $\pm 0.10$  sp gr distribution curve indicates that a gravity of 1.52 must be reached before one finds less than 10 pct of the particles between a 0.10 difference in sp gr, an amount generally accepted as the maximum limit for good separation. Of the feed 12 pct lies between 1.40 and 1.60 sp gr. The theoretical yield at 1.52 is 74.0 pct, giving a cumulative float-ash percent of 7.0.

Table VII. Float-and-Sink at 1.40 and 1.60 Sp Gr on Table Products from 1/4 in. x 0 Feed, Case Study "B"

Table Zone	Prod- uct Wt, Pct	At 1.40 Sp Gr <sup>a</sup>				Cum. Sink, Pct	At 1.60 Sp Gr <sup>a</sup>				Cum. Sink, Pct
		Float, Pct		Sink, Pct			Float, Pct		Sink, Pct		
		Wt	Ash	Wt	Ash		Wt	Ash	Wt	Ash	
1	11.8	99.0	2.4	1.0	19.4	1.0	99.9	2.5	0.1	44.1	0.10
2	40.6	98.3	2.7	1.7	17.6	1.6	99.8	2.9	0.2	35.2	0.18
3	27.1	95.1	3.1	4.9	20.2	1.7	99.6	3.8	0.4	35.7	0.25
4	4.0	79.6	3.5	20.4	24.9	2.6	96.2	6.7	3.8	49.5	0.42
5	2.6	69.5	3.6	30.5	31.8	11.7	84.8	7.3	15.2	39.5	0.86
6	0.7	61.9	3.7	38.1	35.0	13.7	76.3	7.4	23.7	42.3	1.05
7	0.6	32.0	4.4	68.0	40.9	18.2	48.1	11.1	51.9	46.2	1.40
8	9.8	4.0	4.4	96.0	77.3	26.1	6.4	12.6	95.6	77.9	11.05
9	2.8	0.0	0.0	100.0	85.1	28.9	0.2	16.9	99.8	85.2	13.54
Total	100.0										

<sup>a</sup> Float-and-sink analyses on 1/4 in. x 100-mesh, data do not include the —100-mesh.

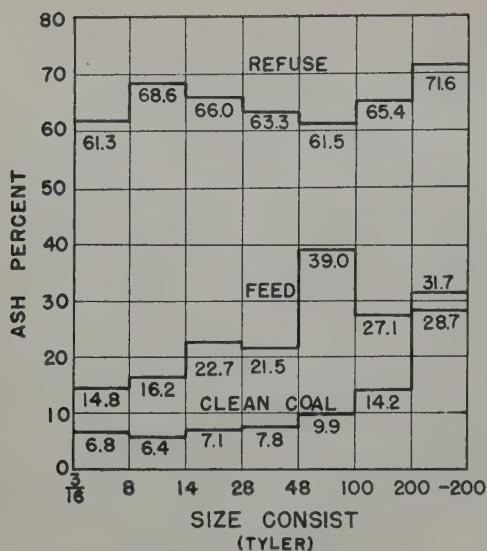


Fig. 13—Case study “C” 3/16 in. x 0 feed, showing ash distribution according to size consist.

Table IX shows the screen sizes of the table products with ash analyses. The calculated head ash of the clean coal was 9.4 pct with 67.3 pct for the refuse. Based on the formula that

$$\text{Recovery, } R = \frac{t-f}{t-c} 100$$

where  $t$  is the ash of the refuse,  $f$  is the ash of the feed, and  $c$  is the ash of the clean coal product, the recovery or yield was 78.0 pct. Examining the theoretical yield of 78.0 pct on the raw feed washability curve in fig. 12, one notes that the float coal shows a theoretical ash of 8.0 pct.

Again examining the results shown in table IX, one finds that the cleanest increment in the clean coal product is the 8x14-mesh which analyzed 6.4 pct ash. The highest ash is in the —200-mesh

Table VIII. Screen Sizes and Ash Analyses of 3/16 in. x 0 Feed to Deister Table, Case Study “C”

Size, Mesh	Direct, Pct		Cumulative, Pct			
	Wt	Ash	Wt	Ash	Wt	Ash
3/16x8	16.8	14.8	16.8	14.8	100.0	22.2
8x14	25.9	16.2	42.7	15.7	83.2	23.9
14x28	20.6	22.7	63.3	17.9	57.3	27.4
28x48	13.9	21.5	77.2	18.6	36.7	30.0
48x100	12.8	29.0	90.0	21.0	22.8	35.0
100x200	4.8	27.1	94.8	21.2	10.0	29.8
—200	5.2	31.7	100.0	22.2	5.2	31.7
Total	100.0	22.2				

Table IX. Screen Sizes and Ash Analysis of Clean Coal and Refuse from 3/16 in. x 0 Table Test, Case Study “C”

Size, Mesh	Clean Coal						Refuse					
	Direct, Pct		Cumulative, Pct				Direct, Pct		Cumulative, Pct			
	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash
3/16x8	15.7	6.8	15.7	6.8	100.0	9.4	16.6	61.3	16.6	61.3	100.0	67.3
8x14	25.9	6.4	41.6	6.5	84.3	10.2	26.9	68.6	43.5	66.0	83.4	66.1
14x28	21.7	7.1	63.3	6.7	68.4	10.4	23.1	66.0	66.6	63.3	56.5	64.9
28x48	13.8	7.8	77.1	7.0	46.7	12.0	16.5	53.3	83.1	61.5	33.4	63.0
48x100	10.3	9.9	87.4	7.4	32.9	12.1	9.7	61.5	92.8	65.4	16.9	64.7
100x200	4.6	14.2	92.0	7.7	12.6	23.4	3.1	65.4	95.9	71.6	7.2	68.9
—200	8.0	28.7	100.0	9.4	8.0	28.7	4.1	71.6	100.0	67.3	4.1	71.6
Total	100.0	9.4					100.0	67.3				

analyzing 28.7 pct. Cumulating the sizes down to 28-mesh, one obtains a 7.0 pct ash product. Cumulating the sizes down farther to 100-mesh, one obtains a 7.4 pct ash product, and at the 200-mesh level 7.7 pct ash results. Obviously the —200-mesh is the offender in bringing up the composite ash of the total clean coal to 9.4 pct.

It is considered noteworthy that the 48x100-mesh coal has been reduced from 29 pct ash to 9.9 and the 100x200-mesh increment from 27.1 pct ash to 14.2 pct.

Table X records the results of float-and-sink analyses of the clean coal and refuse at 1.50 and 1.60 sp gr only on the +100-mesh sizes because of the difficulty of float-and-sink testing of the —100-mesh sizes. As such, it is only partly indicative of the actual results. One notes that the clean coal +100-mesh had 3.5 pct sink and the refuse had 4.7 pct float material at 1.50 sp gr. At 1.60 sp gr there was 1.8 pct sink in the clean coal and 8.5 pct float in the refuse. Fig. 13 illustrates graphically the ash distribution, by various size increments, in the table feed, clean coal, and refuse.

### Case Study “D”

This study is on a 3/8 in. x 0 coal from Greene County, Pa. It is from the Pittsburgh seam and was screened from the run-of-mine. It is an example of high-sulphur coal of which there are large reserves available in that area and which would make excellent metallurgical coking coal if the sulphur content could be reduced to a reasonable amount. The object of the testing was primarily one of sulphur reduction rather than ash.

Washability studies were made showing both ash and sulphur reduction. The Deister table was set to operate at a low gravity. Zone samples were taken along the table deck, with zones 1 through 6 being progressively along the coal discharge side of the table, zones 7, 8, and 9 along the refuse end with zone 7 at the diagonal corner. Table feed rate was estimated as 8.5 tph.

Table XI gives the screen, ash, and sulphur analyses of the table feed. The head ash of this coal is low, analyzing only 10.4 pct. The sulphur content is generally progressively higher in the finer sizes, increasing from a low of 2.30 in the +3/8 in. size to 2.98 pct in the 48x100-mesh and dropping slightly in the —100-mesh to 2.90 pct.

Since these analyses are “total” sulphur on a dry basis, it is important to know what the “inherent” or combined organic and sulphate sulphur content is for this coal, as no cleaning process will remove this “inherent” sulphur but only extraneous sulphur in the form of the iron sulphides of pyrite or marcasite.

In this particular test the exact “inherent” sulphur content was not available, but analyses of



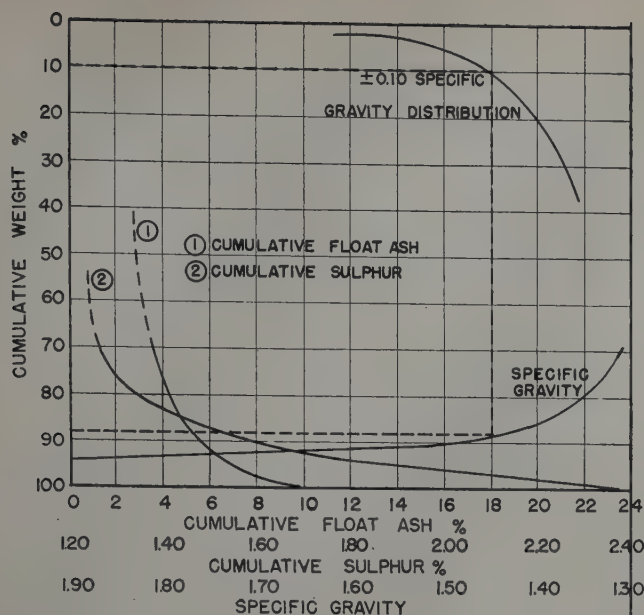


Fig. 14—Case study "D" washability of  $\frac{3}{8}$  in. x 100-mesh high-sulphur table feed.

other similar samples showed it to vary between 0.95 and 1.05 pct with possibly an average of 1.00 pct. One can assume, therefore, that if 100 pct of the extraneous sulphur were removed, the coal would still contain about 1.00 pct sulphur.

Fig. 14 shows the results of float-and-sink analyses of the table feed in the form of washability curves. From the viewpoint of ash removal, the coal is quite easy to clean. The  $\pm 0.10$  sp gr distribution curve indicates that one can separate as low as 1.45 without running into difficult washing.

At this 1.45 sp gr a yield of 88.0 pct can be theoretically obtained, giving a cumulative float ash of 5.5 pct. The sulphur content of the 1.45 float coal is 1.52 pct. If it were possible to separate this coal at 1.35 sp gr, the sulphur in the float coal would coincidentally be also 1.35 pct, assuming a perfect separation.

Table XII gives the yield, ash, and sulphur of the table product zones of case study "D". The lowest sulphur content of any of the zones is that of zone 2, which analyzed 1.64 pct. The cumulative product from zones 1 through 6 gave a total sulphur content of 1.80 pct with the combine refuse zones 7, 8, and 9 containing 11.89 pct sulphur.

From a table recovery standpoint it is interesting to note that 86.3 pct of the total table product came off the first three zones with a sulphur content of 1.72.

Tables XIII and XIV show the results of float-and-sink tests at 1.35 and 1.60 sp gr on the products

Table X. Float-and-Sink Analysis 3/16x100-Mesh Products

Product	At 1.50 Sp Gr				At 1.60 Sp Gr			
	Float, Pct		Sink, Pct		Float, Pct		Sink, Pct	
	Wt	Ash	Wt	Ash	Wt	Ash	Wt	Ash
Clean coal	96.5	6.4	3.5	34.3	98.2	6.7	1.8	55.5
Refuse	4.7	18.6	94.3	69.8	8.5	21.2	91.5	71.0

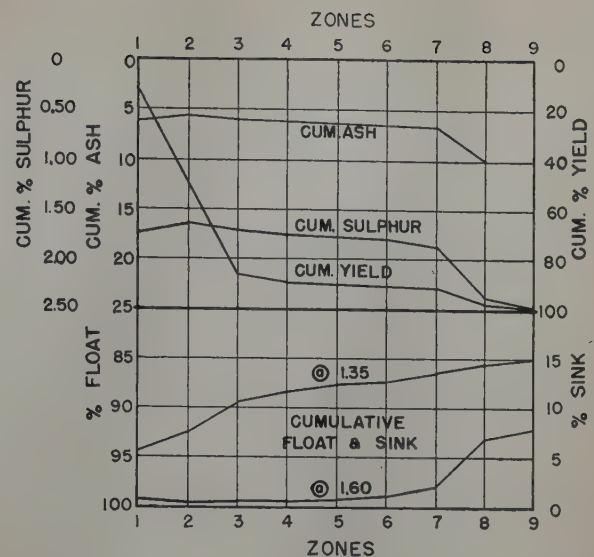


Fig. 15—Case study "D"  $\frac{3}{8}$  in. x 0 high-sulphur feed showing cumulative ash, yield, sulphur, float-and-sink byproduct zones.

from the various zones. In these tests the —100-mesh particles were previously screened out. Fig. 15 presents graphically the cumulative ash, sulphur, and yield percent and the cumulative float-and-sink at the two above-mentioned gravities, starting cumulatively with zone 1 and continuing through all nine zones.

As the washability curves in fig. 14 show, any separation below 1.45 sp gr would be quite difficult because of the high percentage of near gravity material which would, as expected, show up in the high percentage of sink in the table clean coal products at 1.35 sp gr as well as in the percentage of float coal in the refuse. At 1.60 sp gr the results,

Table XI. Screen Size, Ash and Sulphur Analyses of  $\frac{3}{8}$  in. x 0 High-Sulphur Feed, Case Study "D"

Size	Direct, Pct			Cumulative, Pct					
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
+ $\frac{3}{8}$ in. rd	12.8	12.8	2.30	12.8	12.8	2.30	100.0	10.4	2.41
$\frac{3}{8}$ x $\frac{1}{4}$ in.	21.3	10.1	2.40	34.1	11.1	2.36	87.2	10.1	2.43
$\frac{1}{4}$ x 14 mesh	37.2	9.5	2.31	71.3	10.3	2.33	65.9	10.1	2.44
14x48 mesh	19.6	9.8	2.46	90.9	10.2	2.36	28.7	10.8	2.61
48x100 mesh	3.4	11.7	2.98	94.3	10.2	2.38	9.1	12.9	2.93
—100 mesh	5.7	13.7	2.90	100.0	10.4	2.41	5.7	13.7	2.90
Total	100.0	10.4	2.41						

Table XII. Zone Analysis of Table Products from  $\frac{3}{8}$  in. x 0 High-Sulphur Feed, Case Study "D"

Zone	Direct, Pct			Cumulative Recovery, Pct			Cumulative Reject, Pct		
	Wt	Ash	Sul.	Wt	Ash	Sul.	Wt	Ash	Sul.
1	10.8	6.2	1.74	10.8	6.2	1.74	100.0	10.2	2.51
2	40.0	5.7	1.64	50.8	5.8	1.66	89.2	10.7	2.60
3	35.5	6.2	1.80	86.3	6.0	1.72	49.2	14.7	3.38
4	5.3	9.1	2.36	91.6	6.1	1.76	13.7	36.7	7.47
5	1.1	17.9	4.36	92.7	6.3	1.79	8.4	54.1	10.70
6	0.3	28.4	6.20	93.0	6.4	1.80	7.3	59.6	11.66
7	1.1	37.3	7.20	94.1	6.7	1.86	7.0	60.9	11.89
8	5.6	65.2	12.00	99.7	10.0	2.43	5.9	65.3	12.76
9	0.3	67.9	27.22	100.0	10.2	2.51	0.3	67.9	27.22
Total	100.0	10.2	2.51						

**Table XIII. Float-and-Sink at 1.35 Sp Gr on Table Products from  $\frac{3}{8}$  in. x 0 High-Sulphur Feed,<sup>a</sup> Case Study "D"**

Table Zone	Product Wt, Pct	Float, Pct			Sink, Pct		
		Wt	Ash	Sul.	Wt	Ash	Sul.
1	10.8	94.5	4.2	1.42	5.5	15.8	4.06
2	40.0	91.4	4.6	1.44	8.6	14.8	3.53
3	35.5	85.1	4.3	1.40	14.9	15.9	3.90
4	5.3	71.3	4.8	1.47	28.7	19.6	4.52
5	1.1	29.8	5.6	1.63	70.2	22.0	5.23
6	0.3	2.3			97.7		
7	1.1	2.4	5.5	1.79	97.6	37.2	6.97
8	5.6	2.2	5.6	1.73	97.8	66.6	12.07
9	0.3	0.0			100.0	26.3	26.05
Total	100.0						

<sup>a</sup> Float-and-sink analyses on +100-mesh sizes only.

as shown in table XIV, are excellent as far as the percentage of misplaced particles is concerned.

The minimum practical gravity of separation appears to be at 1.45. This is, of course, on an unclassified feed and with a fairly wide size range attempting to clean down to zero sizes. Better results could be achieved by closer sizing and also by making a middling or steam coal grade secondary product.

**Table XIV. Float-and-Sink at 1.60 Sp Gr on Table Products from  $\frac{3}{8}$  in. x 0 High-Sulphur Feed,<sup>a</sup> Case Study "D"**

Table Zone	Product Wt, Pct	Float, Pct			Sink, Pct		
		Wt	Ash	Sul.	Wt	Ash	Sul.
1	10.8	99.5	4.6	1.51	0.5	39.3	12.98
2	40.0	99.9	5.4	1.61	0.1	35.7	7.59
3	35.5	99.4	5.9	1.75	0.6	33.3	5.30
4	5.3	96.0	8.0	2.22	4.0	35.3	5.40
5	1.1	83.3	12.4	3.94	16.7	33.4	5.26
6	0.3	54.1	21.2	6.25	45.9	34.9	5.65
7	1.1	14.0	20.2	6.02	86.0	39.0	6.98
8	5.6	4.4	12.6	3.71	95.6	67.6	12.22
9	0.3				100.0	68.3	26.05
Total	100.0						

<sup>a</sup> Float-and-sink analyses on +100-mesh sizes only.

### Summary and Conclusions

An attempt is made to discuss some of the theory of wet tabling and the phenomena surrounding tabling action. The Deister diagonal-deck table is described. In discussing the performance of this table in the cleaning of the finer sizes of bituminous coal, a study was made of four tests run on unclassified feeds, each test being made on different types and sizes of coals. The author has attempted to be objective in his analyses, presenting the facts of the different studies based on actual table performance.

The separation of different minerals on a wet concentrating table is the result of their differences in specific gravities, size and shape of particles, and the phenomena of stratification and hindered settling. These factors can be controlled and aided by many possible table adjustments, including speed, stroke, side tilt, end elevation, deck area, riffle design, shape of deck, and water distribution.

A diagonal-deck type of concentrating table offers the advantage of a more effective deck area at the section of the deck where it is most needed. For most coal washing a riffle design where the riffles are parallel to the differential head motion appears to give the most satisfactory results. A desirable table for wet washing or concentration of coal is one that provides the flexibility of adjustments so that it can be operated to fit the particular coal being treated and produce the desired results.

Capacities of tables will vary with their size, shape, and other operating characteristics, as well as with the particle sizes being treated, the inherent washability of the coal itself, and the results desired by the operator.

A study of the four cases incorporated in this report indicates that a high degree of performance was achieved under a wide variety of conditions. In comparing the results of any process, it must be remembered that the inherent characteristics of the coal must always be taken into consideration and that a true comparison between different processes can be made only when treating the same kind and size of coal.

It is believed that the type of coal washing table herein studied will produce a satisfactory product from fine sizes of coals and fulfill most requirements while operating on an unclassified feed. Test data indicate that bituminous coal fines of size consist ranges of  $\frac{1}{2}$ ,  $\frac{3}{8}$ ,  $\frac{1}{4}$  and  $\frac{3}{16}$  in. x 0 can be treated effectively. If a higher degree of separation accuracy is required, a closely sized or classified feed may become necessary. This latter course may be especially desirable if the primary purpose is sulphur removal.

### References

- <sup>1</sup> R. H. Richards and C. E. Locke: Textbook of Ore Dressing. 1925. New York. McGraw Hill Book Co.
- <sup>2</sup> A. M. Gaudin: Principles of Mineral Dressing. 1939. New York. McGraw Hill Book Co.
- <sup>3</sup> W. B. Phillips: Practical Results from the Campbell Coal Washer. *Eng. and Min. Jnl.* (1893) **55**, 128-129.
- <sup>4</sup> G. A. Richardson: Rosedale Uses Bumping Tables to Wash its Products, Mixing it for Coking Purposes with Gas Coal. *Coal Age* (1923) **23**, 285-292.
- <sup>5</sup> B. W. Gandrud: Concentrating Tables. Chap. XIII. (1943) Coal Preparation. AIME.
- <sup>6</sup> B. M. Bird and H. S. Davis: The Role of Stratification in the Separation of Coal and Refuse on a Coal Washing Table. U. S. Bur. Mines, R.I. 2950 (1929).
- <sup>7</sup> Charles C. Boley: Analyses of Coal Cleaning on a Concentrating Table. Ill. State Geol. Survey, R.I. 136 (1949).
- <sup>8</sup> H. F. Yancey: Determination of Shapes of Particles and Their Influence on Treatment of Coal and Tables. *Trans. AIME* (1931) **94**, 365-368.
- <sup>9</sup> H. F. Yancey and C. G. Black: The Effect of Certain Operating Variables on the Efficiency of Coal Washing Tables. U. S. Bur. Mines, R.I. 2755 (1926).
- <sup>10</sup> Arthur F. Taggart: Handbook of Ore Dressing. 1948. New York. Wiley and Sons.
- <sup>11</sup> B. M. Bird and H. F. Yancey: Hindered Settling Classification of Feed to Coal Washing Tables. *Trans. AIME* (1930) **88**, 250-271.
- <sup>12</sup> Byron M. Bird: The Sizing Action of a Coal Washing Table. U. S. Bur. Mines, R.I. 2755 (1926).
- <sup>13</sup> W. R. Chapman and R. A. Mott: The Cleaning of Coal. 1928. England. Chapman and Hall, Ltd.
- <sup>14</sup> R. E. Zimmerman: Fine Coal Tabled and Dried. *Coal Age* (Jan. 1948).



# Specific Safety Problems Applicable in West Virginia Mines

by Arch J. Alexander

**This paper is a resume of a study undertaken by the West Virginia Department of Mines. The underlying and direct causes of accidents are determined in each occupational group. Then from this study, accident prevention courses, visual and written, are to be taken to the various working groups in the mines.**

THE causes of accidents in West Virginia mines follow the main pattern of coal industry accidents in the United States—falls of roof, ribs and face, 56 pct; from haulage, 25 to 30 pct; blasting, 3 pct; electrocutions, 3 pct; explosions, 2 pct; and other causes, about 10 pct.

To arrive at some satisfactory conclusions regarding causes and prevention of mine accidents, the Department of Mines has made a study of all mine accidents, from 1945 to 1948, inclusive. During the last two years of this period, we did not have a major explosion, one that took the lives of as many as five men. The direct cause of these accidents is apparent in the reports but we who are making a close study into their frequency think that the virus lies much deeper, and we want to suggest an underlying cause.

The rapid growth of the coal industry in West Virginia, the largest producer of bituminous coal, has resulted in a demand for manpower which probably has never been equaled in mining history. At the turn of the century, West Virginia had 24,635 men employed in and about her mines, and they produced 21,153,341 tons of coal a year. In 1948, there were 125,669 men employed, and coal production for the year was 168,589,033 tons, an average increase of 35 pct for each year since 1900. Over 100,000,000 tons of this increase were mined in the southern counties of the state.

A safety analyst might read into the history of this development an underlying cause for accident occurrence. He might say that in such a vast, rapid development the available supply of trained miners became exhausted and that young, inexperienced men were employed in the mines by the thousands. Being unacquainted with the hazards of mining, it would be only natural for these young men to be injured or killed in too great a number. This assumption would not be borne out by the records. Men were not injured or killed in disproportionate numbers during their first years in a mine. The only reason we can give is that, during these years, they possessed an attribute of safety—CAUTION—to a much greater degree than in the years following.

The coal companies in West Virginia that have been operating over a number of years have a nucleus of employees who have been with them ten, twenty, thirty, and forty years; many of them having gained all of their mining experience with one company. Their habits of work have been

---

ARCH J. ALEXANDER is Chief, West Virginia Department of Mines, Charleston, W. Va.

AIME New York Meeting, February 1950.

TP 2909 F. Discussion (2 copies) may be sent to Transactions AIME before Oct. 31, 1950. Manuscript received Feb. 10, 1950.

---

steady and their proneness to accidents less than in any other group. They have remained long enough in one place to have acquired safety habits. When they make up a large percentage of the total employees of any one company, safety promotion is given a head start.

On the other hand, our accident figures indicate that there is a large segment of mine workers, especially in the rapidly developing fields, which is continually on the move. These miners have accumulated experience in two or more states and in the various fields of each state. Their restlessness is not psychologically conducive to the acquiring of safety habits.

They are of all ages, the young man in his twenties and the mine worker of sixty, and they claim experience in all the occupations in and about our mines. This roving band suffered an average of 42 pct of all our fatal accidents and as high as 60 pct in the counties having the highest accident rates.

The average for the group: Age, thirty-five; total mining experience, fifteen years; and length of service with the mine where they were killed, 4.7 months. They were injured or killed within the year not because of unfamiliarity with any new danger but because of their disregard for the old ones.

Our records show that a county with settled workers, many of whom are second and third generation miners, owning their own homes, and standing out as an integral part of their community, has a better safety record than a county in which there is a heavy labor turnover. In the counties having a heavy labor turnover, we see an answer to the problem by a breakdown of the accident records. In one of our largest producing southern counties, one fourth of the companies, large and small, mining one half of the production, have a safety record comparable to any group of mines in the nation despite their heavy labor turnover. Others could follow their example.

From this analysis, we also can see that the good worker is the safe worker. By the good worker, we mean the man who stays on the job year in and year out, who, in all probability, owns his home and is a steady, everyday man. He is the man who does not change companies too often, and every effort should be made to keep this type of worker at your particular mine as long as possible. This not only brings up the question of safety, but also a social question, which must be solved at each individual mine.

In West Virginia we have three well-known methods of mining, following plans that differ somewhat in pattern, but all with the same objective, and that is the extraction of the coal from the working faces. These methods of coal extraction have changed somewhat during this 4-yr study: Hand loading into mine cars, 1945, 42.7 pct; 1948, 31.7 pct. Mechanical loading by machines, 1945, 43.9 pct; 1948, 51.6 pct. Hand loading into conveyors, 1945, 13.4 pct; 1948, 16.7 pct.

#### **Coal Loading by Hand into Mine Cars**

We are experiencing a sharp decline each year in the amount of coal hand loaded into mine cars. However, this group of hand loaders still makes up the largest single group of men within our mines in a single occupation. They averaged, for the 4-yr period of study, 21.3 pct of all our mine workers,

including supervisors. They made up 41 pct of all the men whose principal duties keep them at the working face. They still exceed the combined number of cutting machine, loading machine operators and helpers, and hand loaders into conveyors.

In this group of hand loaders into mine cars occurred 20 pct of our fatal accidents and the largest number of our serious nonfatal accidents. Even with such a showing, their record is better than the average for all groups, nine others proving more hazardous.

Of their total fatalities, 75 pct were from falls of roof, ribs or faces at the face. Being crushed against the face by cars, being caught in mountain bumps, and blasting, account for another 12 pct, making a total of 87 pct at the working faces. Of the accidents to these face men, 16 pct were from causes beyond their control, either carelessness on the part of a fellow worker or a hazard under the control of management and supervision. Few, if any, were not preventable.

Since most of these accidents, both fatal and non-fatal, to hand loaders were caused by the fall of roof, we immediately arrive at the conclusion that there is insufficient timber being set. This, without a question of a doubt, is the answer. Timber is not being set for one of three reasons; (1) The timber itself is not available, (2) the coal loader is not conscious of his danger, or (3) insufficient or ineffective supervision. Lack of discipline has been a great factor in accidents in our mines. Our records also show that the injuries or deaths of men in a working face were the result of insufficient timbering. Therefore, there can be only one conclusion—timber the working face and you will save the life and limb of the miners.

#### **Machinemen, Cutting**

Machinemen, cutting, come after brakemen in the number killed. So that we may keep together the groups with the most roof fall fatalities, we will discuss haulage later.

Machinemen account for only 5.25 pct of our total employees, but for 11.10 pct of the total fatalities. Of the fatalities, 68 pct were from falls at the face and 22 pct were from causes such as careless tramming, handling of the machine at the face, loose clothing, and being caught between machine and low top. Again, 10 pct were from causes beyond the control of the men.

Men cutting coal are within the group that are termed "skilled workers." They are a rather independent crew and, in most cases, work by themselves. Lack of training as to procedure in their work is the great killer of this group. Machinemen must be taught to reset timbers which they themselves knock out and to examine the roof before and during the cut. They must be taught to examine for gas before and during the process of cutting a place. However, examining for gas is secondary to inspection of the roof. Many a machineman knocks a timber down and never resets it. Again, a system of timbering at the face, carried out, even though many posts must be reset, will go a long way toward saving lives in this group.

#### **Loading Machine Operators and Helpers**

This group is very much like that of the machinemen and accounts for 4.95 pct of our employees and for 10.20 pct of the total number of men killed in our mines. Their record is only a little better than



that of brakemen and shuttle car operators. Eighty-three percent of the loading machine operators and helpers were caught under roof or rib falls at the face while loading coal, or on entries while loading slate. About 8 pct of them were killed by causes beyond their control.

Again, these men must be taught the dangers of unsupported roof. Since they are working around a great deal of noise, they cannot hear when the roof cracks or breaks. Therefore, we must timber for them. Loading machine operators should never be permitted to take a loading machine past the last crossbar or the last row of timbers, even though the machine must be stopped until the place is made safe by additional timbers. If this were carried out, coal production would be higher and a much better accident record would result.

### **Timber and Slate Men**

Timber and slate men comprise 5.02 pct of our workmen and accounted for 7.60 pct of the total fatalities. Ninety percent of these fatalities were the result of falls of roof or rib, causes which were under their control. Six percent of these men were killed from causes beyond their control. A great many of the injuries and fatalities in this group could be avoided if the men would make themselves safe before starting to work. If they would set the necessary timber in the working faces or take the necessary precautions while cleaning up a slate fall, the fatalities would be reduced to nil.

Let me point out again the importance of supervision and discipline.

### **Hand Loaders into Conveyors**

Men hand loading coal into either shaker or chain conveyors make up 8.12 pct of our total employees. They account for 6.9 pct of our fatalities. This is by far our safest group but, of the men killed loading conveyors, 77 pct of them were killed by the fall of roof and rib.

I believe the fine record made by hand loaders into conveyors is due to closer supervision. The average supervisor does not have too much territory to cover on a conveyor section because the coal is usually thin and a systematic method of timbering is carried out in most cases. The small percentage of men killed during conveyor loading may be used as a yardstick for all other occupations. Still, even with this fine record, there is room for vast improvement. A better method of safety timbering and better supervision would go a long way toward saving lives in this group.

### **Supervision**

While supervisors are only 6.97 pct of our total employees, they account for 6.60 pct of our fatal accidents. Forty percent of the supervisors' deaths, fewer than in any other group, were from slate falls and 60 pct were from all other known causes. All of them could have been prevented and none took place in the saving of lives. The supervisor, traveling from place to place, constantly on the move, has a more dangerous job than almost any other worker within the mines.

In West Virginia we have an average of one supervisor for every eight workers and it seems that the leaders should set a better example for the men working under them by being more safety conscious themselves. I believe most face supervis-

ors are far better production men than they are safety men. It is the way they have been taught. Now, if we intend to reduce hazards to this group, we must teach them the proper way to supervise as well as proper safety practices. This is strictly a local problem which must be worked out by the management of each mine.

### **Trackmen**

Trackmen constitute 6.12 pct of our workmen and account for 4.20 pct of all fatalities. They can be credited with being, from a fatality standpoint, one of our safest groups of inside workers. Yet, 70 pct of their fatalities are the result of slate falls. There is no question that this group must be taught not to work in unsafe areas. This is a matter of roof testing and either keeping workers from dangerous places or seeing that they properly protect themselves. Again, 10 pct of the fatalities in this group were from causes beyond the control of the workers.

### **Drillers**

Drillers, both coal and rock, are a small group accounting for 1.56 pct of our total workmen, but accounting for 2.04 pct of our fatal accidents. Slate falls at the face caused 46 pct of these accidents that could have been prevented. Thirty percent were from causes beyond the workers' control. Among these were accidents caused by runaway machines, runaway shuttle cars, the use of defective equipment, and loose clothing. Such accidents should never have happened. The drillmen at the face should take the necessary safety precautions of checking the roof and setting the necessary safety timbers. They should become aware of the other equipment being used near them.

### **Shot Firemen**

This is one of the smallest groups in the mines, constituting only 1.65 pct of our total workmen. Yet, they had 2.30 pct of all the fatal accidents. Of these men, 83 pct were killed from falls at the face. Again, proper face precautions, the setting of safety timbers, and the awareness of roof fall dangers must be emphasized if we intend to save the lives of this group.

### **Haulage Crews**

**Brakemen:** While all the groups in the mine are involved to a certain extent in haulage accidents, they occur mainly to our locomotive crews. Of all the workmen in a coal mine, brakemen have the most injuries. While accounting for only 5.25 pct of our total employees, about the same as cutters, they are charged with 12.50 pct of all underground fatalities. Their nonfatal injuries run very high in permanent disabilities and there are about eleven nonfatal accidents for every fatal accident. Fifty percent of these fatalities were caused by dangerous practices, such as riding front bumpers of cars or locomotives, riding between cars or locomotives, jumping in front of trips, and riding on the sides or top of locomotives and cars. All of these practices violate the rules of the Department of Mines. All of these brakemen were killed because they would not follow known safety rules.

Brakemen should be thoroughly schooled before they are permitted to get on a locomotive. A brakeman who continues to violate company rules, state rules and laws, etc., should be transferred from that particular job to another or discharged.



Twenty-seven percent of our brakemen were killed from causes beyond their control. These were by hazards which the supervisors or management should have discovered and removed. The remaining 23 pct were killed as a result of collisions, rerailling cars with a stiff jack, excessive speed, being caught in tight places and in thin coal, or by low roof over cars and locomotives. None of these are causes which cannot be corrected by the mine management. It is a well-known fact that haulage crews get the least supervision of any group of workers in a coal mine.

**Motormen:** Motormen constitute about the same number of the total employees as brakemen, 5.62 pct, but their fatalities are fewer than those of the brakemen, being 9.30 pct of all fatalities or about 3 pct less than the average for the brakeman. A great many of these fatalities to motormen are caused by slate falls, defective equipment, collisions, and runaways from trips other than their own. To reduce haulage accidents, which are a disgrace, there must be better control over haulage, the removal of dangerous entry roof, and an educating process for motormen and brakemen. Discipline enters the picture to a great extent and, in mines where discipline is somewhat lax, the danger of accidents to haulage crews is ever present.

### Mechanized Mining

When mechanized mining was introduced in our state, we were told that the workers on the various mechanical units would be more concentrated, that they would have the benefits of instruction and supervision not possible for hand loaders into mine cars, and that safety would be advanced. To see if our expectations have been realized, we can look at the record:

	Tons Loaded Per Fatality	Man Hours Worked Per Fatality
Hand loaders into mine cars	1,070,607	823,450
Hand loaders into conveyors	1,318,064	906,381
Loading machine operators and helpers	2,780,041	373,694

As long as we measure our fatality rate on a tonnage-wide basis, the mechanical loading of coal, with the greater tonnage mined per fatality as compared with hand-loading methods, will present a better record. On the other hand, with the variables we now have in mining coal, an accident rate on a production basis does not express the actual safety record of a mine or of a company. Computing on the basis of man-hours exposure, the present high accident rate of loading machine operators and helpers shows that we have not improved the record but that it is somewhat worse.

The various principal occupations within a coal mine in their relative importance, as far as safety is concerned, and the accident record of each occupation have been presented. We are able to arrive at some definite conclusions very quickly.

The first conclusion is that there is a lack of roof support. About 60 pct of all mine fatalities are caused by roof falls. Therefore, if we are going to improve this situation, we must be sure that each working place has sufficient timber of the right length and that this timber is set. When we accomplish this, we will have gone a long way toward cutting down fatal accidents in the fall of roof.

In the past year, there has come into vogue a new system of roof support in coal mines and that is roof bolting. This system controls the roof either by bridging or by holding the roof directly to an above strata. At the present time, we have some 103 mines in West Virginia using this system of timbering successfully, and there is no question in our minds that roof bolting has a great future. Like every other system of roof support, it must be properly studied so that the right equipment can be used, a definite system set up as to its use, and that system carried out. We are gathering information on this subject and are encouraging the use of bolts under a controlled system. We believe that roof bolting is the most progressive step that has been made to date in coal mining. However, as a system of roof support, it is no better than the people who use it.

Supervision is one of the weakest links in our chain of mining. We select a man as foreman because he happens to be an expert motorman, machineman, or outstanding man in his line of work. We never think, generally speaking, that this man needs special instruction in the many things over which he has supervision. The first step in any accident prevention program must start with the supervisor. This instruction takes time and costs money, but we believe it is the only way that any mine can improve its accident record.

Many years ago, when mechanized mining was first started, it failed in a good many places. Our ignorance of the job caused this failure. It did not take us long to study each step of the work or cycle in mechanized mining. After we had made this study, we trained our men, and the results were very gratifying.

The West Virginia Department of Mines, profiting by this experience, has been studying mine accidents by occupation. For instance, we have broken down mine accidents as to timbermen, trackmen, haulage crews, and machinemen; in fact, every category of underground work, to see what, besides roof, was killing our men. The results of this study were amazing. For example, we found that there were only 11 ways that 98 pct of the trackmen in the mines of West Virginia were being killed. There were even fewer ways in other mining occupations.

It is our intention to set up an accident prevention course that can be taught very rapidly. It is planned to present in one 2-hr session a course on accident prevention to each group of workers, one for trackmen, one for timbermen, etc. We will give this course in simple language. We intend to illustrate with slides, and, when a man has finished his 2-hr course, he will be given a booklet covering the subject matter of the course.

This will be a slow process since, under our present setup, it is not probable that we can reach more than 100 mines a year, and, in West Virginia, we have some 2600 coal mines. It is our intention to keep a close record on the mines where this instruction has been given in order to determine whether the instruction has been worthwhile. We believe that we will find that it has been.

In conclusion, let me say that we believe that it will be necessary that the top bracket of management be interested in safety and in seeing that the safety programs, as set up by their company, are carried out. If top management is not sold on safety, there is very little chance for safety in coal mines.



# Air Pollution by Industrial Fumes, Gases, and Dusts

by Louis C. McCabe

The control of dusts and fumes of submicron size is involved in many process industries. This paper presents in tabular form the quantitative data from a number of metallurgical operations and discusses allowable discharge in terms of effluent-weight ratio. Typical installations for recovery of stack effluents are cited.

THERE are many sources of smoke, fumes, gases, and dusts that may cause undesirable contamination in an industrial area. Fumes and gases are the result of chemical action or volatilization, while dusts are produced in processes of disintegration and dispersion. Commonly, smokes are suspended particles formed by combustion or destructive distillation. Methods for controlling smoke are generally known, and great improvements have been made in equipment for removing fine dusts and fumes from gases.

In most of the industrial centers of the country, the chief duty of the air-pollution abatement agency is the elimination or reduction of emissions resulting from improper combustion. Correct methods of firing, proper fuel selection, and installation of properly designed equipment may accomplish this. It is becoming more widely recognized that much of the contamination of an industrial atmosphere is caused by dusts, fumes, and gases that are not related to the process of combustion. As such emissions are usually a functional part of plant operation, they rarely can be prevented even by better design and careful operation. When dusts, fumes, and gases are an integral part of the operation, mechanical equipment must be designed to remove these emissions.

Most of the dusts and fumes that should be controlled are less than 1 micron in size. Control of such small particles is largely a new and untried

field in many industrial areas. However, where conditions have been critical, effective control equipment for dusts, fumes, and gases has been in operation for many years. The problem is not one of smoke abatement and requires an entirely different set of standards for effective solution. Identical standards need not be applied to all industrial areas because of differences in meteorology, physiography, plant location, plant density, and nature of stack effluents. In this paper, discussion of control of atmospheric contamination by industrial plants refers specifically to the Los Angeles area, but the method of approach is applicable to any area where similar problems are encountered.

Quantities of particulate matter reported in analysis of stack effluents indicate the need for standards of emission. Many types of dusts and fumes that contribute materially to the air-pollution problem are discharged into the atmosphere at a lower rate than 0.40 grain per cu ft at standard conditions. On the other hand, some stacks discharge at a higher

---

LOUIS C. McCABE, Member AIME, is Chief, Office of Air and Stream Pollution Research, U. S. Bureau of Mines, Washington, D. C.

Papers by authors of the U. S. Bureau of Mines staff are not subject to copyright.

AIME Columbus Meeting, September 1949.

TP 2915 F. Discussion (2 copies) may be sent to Transactions AIME before Oct. 31, 1950. Manuscript received Oct. 26, 1949.

---

Table I. Summary of Data

Type of Process	Process Weight, Lb per Hr	Process Time	Stack Loading Grain per Cu Ft at Std.		Effluent to Weight Ratio, Cu Ft at Std.	Hourly Max Rate of Discharge, Pct	Average Max Rate of Discharge, Pct
			Avg	Max per Hr			
Red brass	500	72 min	0.250	0.393	273	1.53	0.97
Red brass	435	84 min	0.254	0.395	289	1.63	1.04
Red brass	1,500	152 min	0.320	0.559	172	1.38	0.79
Red brass	1,620	130 min	0.595	0.903	147	1.92	1.26
Red brass	470	102 min	0.410	0.780	231	2.57	1.35
Yellow brass	470	102 min	0.410	0.780	231	2.57	1.35
Yellow brass	535	90 min	0.452	0.762	197	2.15	1.27
Brass smelting	2,000	25 hr		0.390	226	2.11 <sup>a</sup>	1.26
Brass smelting	630	95 min		0.985	282	3.95	
Lead smelting	5,000	24 hr		0.355	355	3.01 <sup>a</sup>	1.80
Steel, open hearth	14,500	8 hr	0.798	2.000	55.6	1.58	0.63
Steel, open hearth	10,200	10 hr	0.863	1.720	53.1	1.31	0.65
Gray iron	9,250	80 min	1.604	1.604	32.5	0.75	0.75
Gray iron	13,450	156 min	1.110	1.110	91.5	1.45	1.45
Gray iron	18,900	320 min	0.798	0.798	49.5	0.56	0.56
Gray iron	8,350	Data by outside laboratory				1.07	1.07
Gray iron discharges are constant throughout melt cycle. Average and hourly rate of discharge are equal.							
Steel, electric	3,325	130 min	0.306	1.17		1.02	0.29

<sup>a</sup> Pct, calculated, based on average increase found in the nonferrous industry.

rate, but because of the coarseness of the material the nuisance created is limited to the immediate neighborhood and perhaps only to the property of the company concerned. Much of the characteristic blue haze, which contaminates the atmosphere and contributes to low visibility, is due to particle sizes near the low range of the wave length of visible light, that is, 0.3 to 0.5 micron diam.

It is not possible to separate the industries into well-defined groups according to the physical state of their stack discharges, but for the purpose of this discussion they are grouped as follows: Dust: metallurgical, clay and rock products. Fumes: food processing, fertilizer, paint, roofing and asphalt. Gases: refining, chemical.

Dusts and fumes standards now in effect in Los Angeles County were developed after a year's study of the stack discharges of representative plants. Three or four furnaces are represented in each type of metallurgical operation chosen for basic information. The data (table I) are representative of four types of metallurgical industry, namely, the non-ferrous, steel, gray iron, and electric iron. The non-ferrous operations comprise the largest group numerically, but in gross tonnage of processed material the combined steel industry is larger.

The development of these standards<sup>1</sup> has been

<sup>1</sup>L. C. McCabe, A. H. Rose, W. J. Hamming, and F. H. Viets: *Ind. and Eng. Chem.* (Nov. 1949) 41, No. 10.

discussed elsewhere; significant factors in their use are:

1. Effluent-Weight Ratio—This is the total volume of effluent in cubic feet at standard conditions per pound of material processed and indicates the average dilution ratio for each plant in a given industry. Throughout each industry this ratio tends to hold constant.

2. Hourly Mass Rate of Emission—This is the maximum discharge of solid material occurring in any 1-hr interval expressed as a percentage of the total hourly process weight.

3. Average Mass Rate of Emission—This is the average discharge for the entire process cycle expressed as a percentage of the total process weight per cycle.

Maximum stack loadings, in grains per cubic foot at standard conditions, range from 0.393 in a red-brass operation to 2.0 in a steel open hearth. The maximum allowable discharge, expressed in pounds per hour for selected weights, is presented in tabular form in table II. This was done at 50-lb intervals up to 20,000 lb per hr and in 10,000-lb intervals up to 50,000 lb per hr. A further restriction on the allowable discharge was made in that it becomes constant when 40 lb per hr is reached, regardless of the process weight involved. The collection efficiencies used were those that can be obtained with available equipment within the economic reach of industry. It was felt the additional cost of high-efficiency collection equipment can be carried better by large industries; the smaller industries will be required, therefore, to operate their collecting equipment at 80 pct efficiency and the large industries to operate with collection efficiencies of approximately 90 pct.

Open-Hearth Steel: Two pilot installations on the open-hearth steel furnaces were completed during the year: (1) a venturi scrubber and (2) an electrostatic precipitator. The pilot tests proved that both of these types of equipment will meet Los Angeles standards successfully in the collection of fume and dust from the open-hearth furnaces. Economics, involving water and power consumption, are important factors in the system selected.



Operating data from one open-hearth is as follows:

a. Average metal charge	129,498 lb
b. Average lime charge	5,620 lb
c. Average coke charge	1,450 lb
d. Average total charge, Jan. 1 to June 1, 1949	136,568 lb
e. Average heating cycle (start of charge to finish tap)	8.083 hr
f. Average charge per hr	16,900 lb
g. Combination firing	20,000 cfm natural gas and 72 gph fuel oil
h. 100 pct oil firing	250 gph fuel oil used when on gas curtailment
i. Volume of flue gases	12,500 to 14,000 cfm at standard conditions

The temperature of the gases entering the stack was approximately 1300°F.

The dust loading rate in the stack gases was at the maximum, 2.00 grains per cu ft at standard conditions; the charging average was 0.87 grain per cu ft; and the melt-down average was 0.51 grain per cu ft.

The average charge per cycle for the second open-hearth plant was 111,000 lb on a 10.25-hr cycle. The temperature entering the stack ranged from 1100° to 1250°F. The maximum dust loading was 1.44 grains per cu ft and the minimum, 0.08 grain per standard cu ft. The average was 0.60 grain per standard cu ft.

**Electric Steel:** The fume-discharge problem of the electric steel furnace was solved by using a hood system, which utilized a limited amount of air and discharged through a wet dynamic precipitator. Operating installations of this type of equipment were placed in use during the year, and tests proved these units to be most satisfactory.

**Gray Iron:** This division of the metallurgical industry is farthest from a solution of its problem. Tests in cooperation with the Air Pollution Control District were made in an effort to determine characteristics of the effluent discharges from the cupolas. As a result of these tests, the District believes that this problem will be solved through the application of equipment similar to that employed in the steel industry.

**Nonferrous Industry:** Three types of collection equipment have been tried with varying success on the submicron fume discharge encountered in this industry. Modified wet scrubbers and mechanical separators were discarded as unsatisfactory. Filter-type collectors, however, proved satisfactory when proper hooding was employed. Changes in operating techniques were also tried with limited success. Furnaces were slagged throughout the entire operating cycle, including the pouring phase. On alloys of moderately low zinc and lead content, this method was most satisfactory. As the lead and zinc content increases, the slagging technique appears to be less effective. It is probable that collection equipment will be required in the high zinc alloy operations.

Special air-pollution control methods are required in a wide variety of industries such as rendering plants; chemical fertilizer plants; chromate manufacturing; varnish, paint, and resin plants; detergents; vegetable-oil processing; oil refining, sulphur production; acid plants, paper coating; and food processing.

Table II. Maximum Allowable Discharge per Hour

Process Wt per Hr (Lb)	Allow- able Disch. per Hr (Lb)	Process Wt per Hr (Lb)	Allow- able Disch. per Hr (Lb)	Process Wt per Hr (Lb)	Allow- able Disch. per Hr (Lb)	Process Wt per Hr (Lb)	Allow- able Disch. per Hr (Lb)
50	0.24	1,300	3.26	3,500	5.52	8,500	9.03
100	0.46	1,400	3.40	3,600	5.61	9,000	9.36
150	0.66	1,500	3.54	3,700	5.69	9,500	9.67
200	0.852	1,600	3.66	3,800	5.77	10,000	10.0
250	1.03	1,700	3.79	3,900	5.85	11,000	10.63
300	1.20	1,800	3.91	4,000	5.93	12,000	11.28
350	1.35	1,900	4.03	4,100	6.01	13,000	11.89
400	1.50	2,000	4.14	4,200	6.08	14,000	12.50
450	1.63	2,100	4.24	4,300	6.15	15,000	13.13
500	1.77	2,200	4.34	4,400	6.22	16,000	13.74
550	1.89	2,300	4.44	4,500	6.30	17,000	14.36
600	2.01	2,400	4.55	4,600	6.37	18,000	14.97
650	2.12	2,500	4.64	4,700	6.45	19,000	15.58
700	2.24	2,600	4.74	4,800	6.52	20,000	16.19
750	2.34	2,700	4.84	4,900	6.60	30,000	22.22
800	2.43	2,800	4.92	5,000	6.67	40,000	28.3
850	2.53	2,900	5.02	5,500	7.03	50,000	34.3
900	2.62	3,000	5.10	6,000	7.37		
950	2.72	3,100	5.18	6,500	7.71		
1,000	2.80	3,200	5.27	7,000	8.05		
1,100	2.97	3,300	5.36	7,500	8.39		
1,200	3.12	3,400	5.44	8,000	8.71		

Where the process weight falls between the figures stated, the values of allowable discharge per hour will be interpolated. In no case will particulate matter in excess of 40 lb per hr be allowed to emit from any one source.

Process weight shall be defined as the total weight of raw materials or materials entering the process net weight of finished product. Solid fuels charged will be considered as part of the process weight but liquid and gaseous fuels and combustion air will not.

In continuous operation, the average rate of feed shall be used. In batch operation, the total batch weight divided by the operating time of a batch cycle shall determine the process weight per hour.

In the rendering industry, as in others where odors are the chief air-pollution problem, much progress has been made. Among the methods used for odor reduction were incineration, wet scrubbing, and chemical treatment by proprietary formulas. Incineration is proving to be by far the most successful and practical method. Removal of odors by wet scrubbing has been used successfully in the ammonia-odor nuisance but is generally limited to odors of a water-soluble nature. The method has been used also in the varnish cooking industry. An experimental installation was made by one rendering plant for neutralizing odors by means of a chemical additive in the effluent gases. This proved to be unsuccessful and was abandoned in favor of incineration.

During 1949, the Hancock Chemical Co. built a plant at Long Beach for converting hydrogen sulphide to sulphur. This plant is presently producing 50 tons of sulphur from  $H_2S$  delivered by three refineries and is expanding its facilities so that it can process hydrogen sulphide from three additional major refineries. Prior to construction of the Hancock plant, this hydrogen sulphide was burned as fuel along with the other refinery gases, and the sulphur dioxide produced was released to the atmosphere.

Some pilot-plant work is being done on ultrasonic methods of agglomerating and recovering dusts and mists. Fume problems in the coffee-roasting industry can best be solved by incineration. One installation has been made to test the design factors used in construction.

# Drying Fine Coal in the Entrained and Fluidized State

by V. F. Parry and E. O. Wagner

This paper summarizes investigations during 1949 on three pilot plants for drying low-rank fine coal by entrainment in hot gases. Detailed operating results on processing seven coals having moisture ranging from 24 to 62 pct, formulas and data for calculating performance of large units, and general conclusions on the problem of drying fine coal are presented.

THIS paper presents a summary of investigations by the Coal Branch of the Bureau of Mines during 1949 on the removal of internal moisture from low-rank coals by processing them in the entrained and fluidized state. A paper on this subject, giving a summary of the early work, was presented at the 1949 AIME annual meeting<sup>1</sup> and gave a the-

<sup>1</sup> V. F. Parry, J. B. Goodman, and E. O. Wagner: Drying Low-Rank Coals in the Entrained and Fluidized State. *Trans. AIME* 184, 89-98; *Min. Eng.*, April 1949.

oretical discussion of the time and heat required to dry various sizes of low-rank coal. The investigations on two pilot plants were presented in the foregoing paper. During 1949, three additional pilot plants were built to study phases of the drying problem and to investigate different techniques of handling fine coal in the entrained and fluidized state. The present paper gives operating data on the new units and presents detailed data on the drying of seven coals having bed moisture ranging from 62 to 24 pct.

The problem of removing surface moisture from fine coal is principally one of heat exchange and dispersion of the coal particles so that heat can reach the liquid moisture to permit evaporation, and

it does not involve the factors of size and time of contact to any great extent. If enough heat can reach the surface of the particles at temperatures higher than 250°F, moisture evaporates almost instantly, and the factors of time and size are of secondary importance compared with the factors of heat balance and dispersion of the coal. Therefore, it does not appear to be necessary to employ a fluidized bed when removing surface moisture, and the most efficient process would be one in which the coal particles are entrained in a hot gas stream of optimum temperature traveling at a velocity that causes the particles to disperse into a low-density phase.

When low-rank coals are dried, the fluidized bed is necessary to provide control of time of contact. This is done by adjusting the superficial velocity which controls the density or dispersion of the particles. The density of beds having fluidized properties suitable for drying low-rank coals appears to range from about 5 up to about 15 lb per cu ft. When the density is below 5 lb per cu ft, which represents a low-density phase, optimum conditions for removal of surface moisture should be found, and this requires superficial velocities of about 15 fps.

The problem of removing bed moisture from low-rank coals involves consideration of size of coal, temperature, mass flow, heat balance, pressure, time of contact, and method of dispersing the coal in the heating mediums. The experimental work presented here gives data on these factors. Coal sizes up to ½ in. x 0 were dried with hot gases having initial temperatures ranging from 1900° to 2400°F. Coal

V. F. PARRY, Member AIME, is Supervising Engineer, and E. O. WAGNER is Chemical Engineer, Coal Branch of Region IV, Bureau of Mines, Denver, Colo.

AIME New York Meeting, February 1950.

TP 2905 F. Discussion (2 copies) may be sent to Transactions AIME before Oct. 31, 1950. Manuscript received Jan. 18, 1950.

Papers by authors on the U. S. Bureau of Mines staff are not subject to copyright.



was fed pneumatically and by screw conveyors, and the time of contact in the drying column ranged from 19 to 230 sec.

The principal object of the experimental work cited in this report was to ascertain the relationship of the various factors affecting the drying of fine low-rank coals containing various amounts of natural bed moisture. A few experiments were made on the removal of surface moisture, but the results are not presented as they were not conclusive, and additional work on optimum temperature of hot gases is necessary before results on this problem can be reported.

### Summary and Conclusions

This report presents results of experimental work on the fluidized drying of low-rank coals from Colorado, Wyoming, Alaska, Texas, and Greece, which contain 24 to 62 pct natural bed moisture. They were processed in three pilot plants to find optimum conditions for removing 90 to 95 pct of the moisture. The factors of temperature, time of contact, heat balance, and size were studied, and two methods of dispersing the fine coal in the fluid stream were employed.

The fluidized process for drying fine coal has been under investigation for about two years by the Coal Branch of the Bureau of Mines at Denver, Colo. A preliminary report on the work was presented at the 1949 annual meeting of the AIME. The main features of the drying process are described as follows:

Hot gas under moderate pressure is generated in a combustion chamber by the combustion of pulverized coal or gas. The temperature of the hot gases issuing from the burner is regulated by recirculating inert products of combustion to reduce the free-oxygen content and to produce a temperature suitable for drying the coal. This hot gas is jetted at velocities of about 80 fps into a drying column or heat-transfer zone where it contacts the fine coal and creates a fluidized or entrained mixture of gas and coal in which heat exchange occurs rapidly to evaporate the moisture. The mixture of hot gases and fine coal moves upward in an entrained and fluidized condition to suitable cyclone separators, where the dried coal is deposited and the moisture vapor is discharged. Part of the discharged gases is recirculated to control the temperature and velocity of the gases in the drying column.

This process causes extremely rapid rates of heat interchange between the hot gases and coal particles and results in maximum thermal efficiency. The time required to pass the coal through the system ranges from about 20 to 230 sec, depending upon the size of the coal treated.

Experimental work on various sizes and ranks of coal dried under various conditions in three pilot plants supports the following general conclusions:

1. The capacity of a fluidized drying unit is expressed approximately by the following empirical formula, which shows the relationship of the major variables:

$$C = \frac{3.2ST_1^{0.95}}{M_1}$$

where  $C$  = lb of raw coal per hr per sq ft,

$S$  = mean superficial linear velocity of the hot gas and vapors in the column at the temperature and pressure of the system, ft per sec,

$T_1$  = temperature of hot gases jetted into the raw coal, °F,

$M_1$  = moisture content of the raw coal, pct.

The foregoing formula was derived for a mixture temperature of 300°F and it assumes that 93 pct of the moisture in the coal is removed. The capacity of a drying column is directly proportional to the linear velocity and nearly directly proportional to the temperature of the hot gases while being inversely proportional to the moisture content of the coal.

2. For the average range of coals tested, the practical capacity of a drying unit is about 500 lb of moisture liberated per hr per sq ft. This means that if 20 pct of moisture is liberated from a coal, the capacity of the unit should be about 2500 lb per hr per sq ft, whereas if 35 pct moisture is liberated, the capacity will drop to about 1430 lb per hr per sq ft of drying-column area.

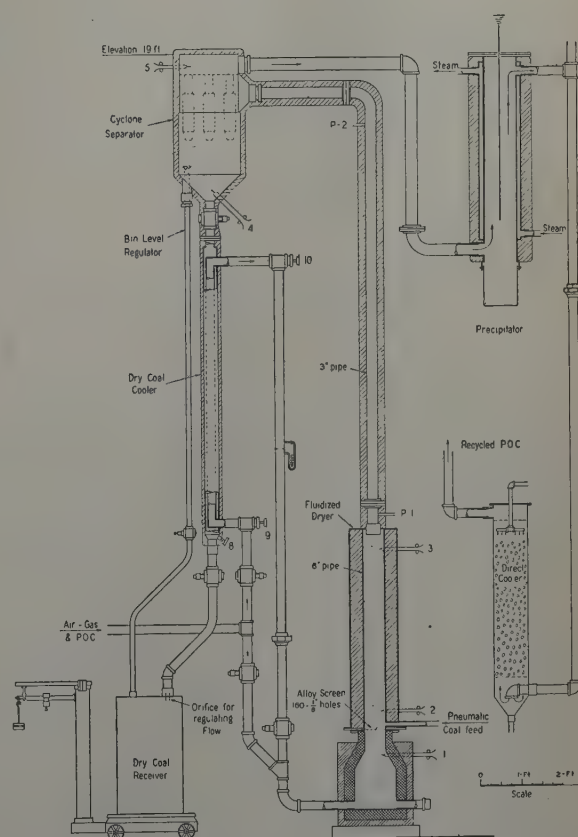


Fig. 1—No. 3 pilot plant for drying low-rank coal in fluidized state, Denver, Colo.

3. The efficiency and capacity of the fluidized drying system increase in almost direct proportion to the hot gas temperature, however, the practical limit of temperature is around 2000°F because temperatures higher than 2000°F promote the formation of ash deposits near the jets. Temperatures as high as 2500°F were studied with good operation except for the formation of ash deposits.

4. Coals having high bed moisture can be jetted with high-temperature gases with virtually no effect on the coal substance when 90 pct of the moisture is liberated. When all of the moisture is liberated,

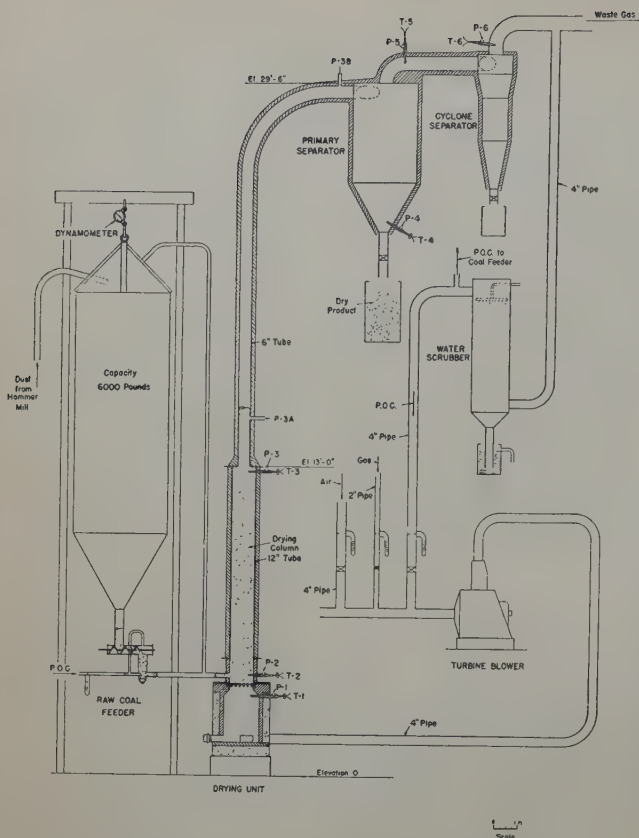


Fig. 2—No. 4 pilot plant for drying fine coal, Golden, Colo. Capacity 2400 lb per hr.

some devolatilization of the coal substance occurs, depending upon the temperature of the mixture.

5. The thermal efficiency of the fluidized drying process is 90 to 93 pct, excluding radiation and with no recovery of heat from the dried coal. In the small pilot plants, the overall thermal efficiency ranged from 73 to 85 pct because of the considerable radiation. However, in a large drying unit, the overall efficiency should approach 90 pct.

6. The density of the coal and gas mixture in the drying column is controlled by the superficial velocity and the size distribution of the coal particles. With equal velocity, the density of the mixture will increase with the average size of coal treated and this increases the time of contact.

7. Considerable degradation of the coal particles, caused by rapid evolution of moisture and shrinkage of the coal, occurs in the drying column. The measured degradation in average size ranged from 15 to 81 pct when Alaskan subbituminous coals and Greek lignites were treated. In general, the amount of degradation approximately equals the bed moisture of the coal treated, but the size of the particles and physical structure of the coal have some bearing on degradation.

8. It has been demonstrated that it is feasible to dry any coal completely and to effect devolatilization of the coal in the fluidized system. The amount of heat transmitted to the coal particles governs the extent of drying or devolatilization. There appears, however, to be an optimum point in removing bed moisture, taking into account capacity and estimated cost of removing moisture. A disproportionately longer time is required to remove the last traces of moisture, and the capacity decreases. It appears

feasible to reduce the moisture in any coal to 2 to 5 pct while attaining maximum capacity and efficiency.

9. The size of the coal particle automatically regulates the time it will remain in the fluid system—the smallest dust particles evidently move with the gas at the same superficial velocity and, as the particle increases in size, it is carried less rapidly by the gas owing to its increased weight and drag. This conclusion is indicated from measurements of pressure drop and the residual moisture in different sizes of treated coal. The size distribution of the coal particles also affects operation of the fluid bed. A preponderance of large pieces will clog the column, and it seems necessary to float the larger pieces with adequate fine material.

10. The drying of a given coal to any degree is a straightforward heat-balance problem. The heat required to evaporate the moisture and raise its temperature to the mixture temperature plus the sensible heat deposited in the dried coal up to the mixture temperature is balanced against the sensible heat of the drying gas between the high and mixture temperatures. This gives the heat balance of materials excluding radiation. The extra sensible heat from the hot gas to supply radiation is then calculated. The unit quantity of materials thus calculated is then fed to the system at predetermined rates to establish the desired superficial velocity.

11. The dried coal dusts are free flowing because of the low moisture content. Although the compacted bulk density ranges from 45 to 56 lb per cu ft, the normal angle of repose of the coal is approximately 45°. The dried dusts flow through orifices by gravity at rates similar to the flow of liquids; for example, the flow of dust, in pounds per minute, equals  $14.5 D^{2.65}$ , where  $D$  is the diameter of the orifice in inches. When the dust is discharged through a 7-ft long pipe, the rate of discharge is approximately four times greater than through an orifice of equal diameter.

12. It is estimated that the investment cost of fluidized drying units should be in the range of \$150 to \$200 per ton of daily capacity, depending upon the moisture content of the coal. The cost of drying a lignite to remove 670 lb of moisture per ton is estimated at \$0.286 per ton. It is indicated that the unit cost of removing moisture should range from \$0.85 to \$1.00 per ton of water removed.

### Experimental Work, Description of Pilot Plants

The No. 3 pilot plant is shown in fig. 1. The object in building this unit was to prepare samples of dried coal for briquetting tests and to study some of the problems of handling and cooling of dried fine coal. In this unit, coal was fed pneumatically to the drying column from a weighing and charging system described in the previous paper. The separating cyclones were about 19 ft above the ground level and connected to a dry-coal cooler through which the hot, dried coal moved to the coal receiver. The mixture of gas and air going to the combustion chamber circulated through the cooler. In the cyclone separator, three small cyclones were used for dust separation. This vessel was insulated to avoid condensation of steam, which had a concentration of about 50 pct in the waste gases.

The fine dust in the waste gas passing the primary cyclone separators was sent through an electrostatic precipitator. This unit did not function very satisfactorily because of shorting out caused by condensation, but when operating without shorting, the



collection of dust was satisfactory. Part of the waste gas was drawn back through a water scrubber to supply recycled products of combustion for the heating system.

A larger drying unit was constructed to study several factors of operation and to treat large batches of coal. This No. 4 plant is shown in fig. 2. The basic principles of operation employed in other units were incorporated in this plant, but the capacity was quadrupled, and provisions were made to gain further experience in handling materials.

The raw crushed coal was stored in a large cylinder suspended on a dynamometer. A screw feeder driven by a variable-speed motor was attached to the cone of the cylinder and transferred coal into the line feeding the drying unit. A jet of products of combustion moved the fine coal through the 1¼-in. id conduit to the drying column. The distance between the feeder and the dryer was about 30 ft. This coal feeder handled ¼-in. coal at any desired rate up to 3000 lb per hr, but the transfer conduit and the carrying gas were adjusted to provide 1 to 2 cu ft of gas per lb of coal at a velocity of about 75 ft per sec.

Table I. Calculation of Drier Operation

Test No.	21	30-B	32-A
Moisture to be removed from coal, pct, (D)	95	95	90
Moisture in raw coal, pct, (M <sub>1</sub> )	58.8	62	37.6
Moisture in dried coal, pct calc., (M <sub>2</sub> )	6.7	7.1	5.7
Improvement ratio $(100 - M_2) \div (100 - M_1)$ , (R)	2.263	2.442	1.512
Moisture removed per lb of coal, = $(1 - 1/R)$ , (X)	0.558	0.59	0.838
Dried coal per lb of raw coal, $(1 - X)$	0.442	0.41	0.662
Temperature of combustion gases (developed), °F, (T <sub>1</sub> )	2,250	2,100	2,100
Temperature of combustion gases (theoretical flame), °F	2,500	2,400	2,400
Temperature at top of drying column, °F, (T <sub>2</sub> )	300	275	250
Temperature of dried coal, °F, (T <sub>3</sub> )	275	250	225
Superficial velocity in column, ft per sec, at T <sub>2</sub> and P, (S)	15	10	10
Heat required to dry coal, Btu per lb: Latent heat in moisture evap. = 1058X	591	624	358
Sensible heat in steam, 60°F to T <sub>2</sub>	61	52	28
Sensible heat in dried coal, 60°F to T <sub>3</sub>	24	19	27
Total heat required excluding radiation, (H <sub>1</sub> )	676	695	413
Sensible heat in POC <sup>b</sup> at T <sub>1</sub> , Btu per cu ft (from chart)	48.2	44.5	44.5
Sensible heat in POC at T <sub>2</sub> , Btu per cu ft	5.0	4.2	4.0
Heat transferred to coal, Btu per cu ft, (H <sub>2</sub> )	43.2	40.3	40.5
POC required = $H_1/H_2$ , cu ft per lb of coal	15.7	17.2	10.2
Volume of moisture liberated = 21X	11.7	12.4	7.1
Total gas and vapor in column, cu ft per lb coal, (V <sub>1</sub> )	27.4	29.6	17.3
Volume of total gas at T <sub>2</sub> and P, cu ft per lb coal, (V <sub>2</sub> )	48.1	50.0	28.3
Dryer capacity, lb per hr per sq ft = $S \times 3600/V_2$	1,122	720	1,270
Gas required:			
Overall efficiency of drier, assumed, (E)	0.85	0.75	0.75
Net heat required = $H_1/E$ Btu per lb coal, (H <sub>3</sub> )	795	927	510
Cu ft of gas per lb of coal H <sub>3</sub> per 880	0.904	1.053	0.580
Air required = 10 x gas cu ft per lb POC recirculated (from combustion chart) <sup>c</sup>	9.04	10.53	5.80
	4.20	5.78	3.19

<sup>a</sup> The theoretical flame temperature is 200° to 350°F higher than the developed temperature, depending upon the design of the combustion chamber.  
<sup>b</sup> Products of combustion.  
<sup>c</sup> A combustion chart is calculated for gas or coal showing the ratio of air/POC vs. theoretical flame temperature when burning with 10 pct excess air.

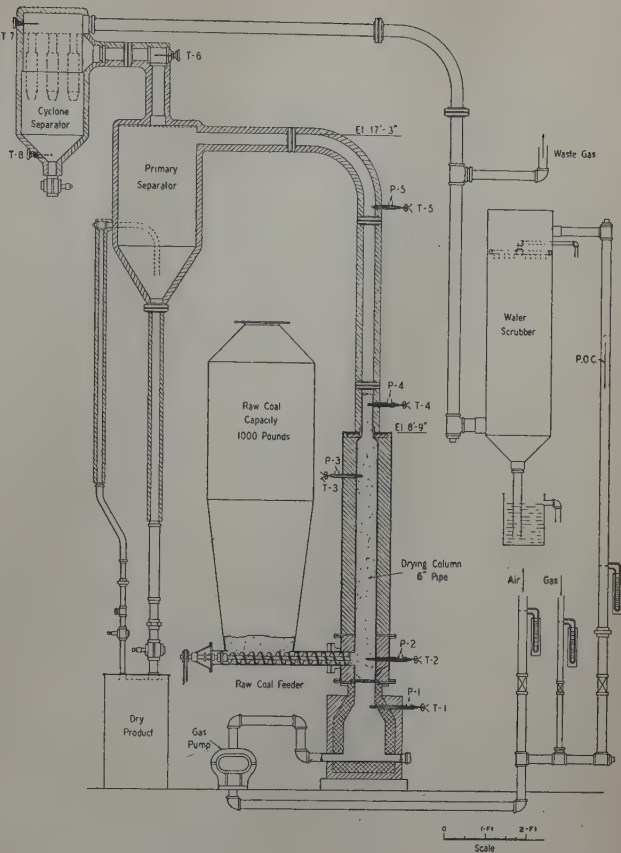


Fig. 3—No. 5 pilot plant for drying coal, Golden, Colo.

In the No. 4 plant, a turbo blower was used to feed the gas and air to the combustion chamber. This introduced the factor of constant pressure on the system, which caused a little trouble. The pressure drop or resistance through the drying column varies somewhat, and with constant pressure available, the flow will vary with the resistance, resulting in pulsations during operation. However, the No. 4 unit performed very well, particularly on the light-density fluid beds.

In this plant a primary separator consisting of a plain cylinder with tangential inlet and center off-take was installed. About 99 pct of the dried product was caught in this separator with little pressure drop. The off-gas passed through a single cyclone separator where fine dust, down to about 10 microns, was separated with a pressure drop of 4 to 6 in. of water.

The grid above the combustion chamber of the No. 4 plant was constructed of plastic refractory approximately 1¼ in. thick with twenty-nine 5⁄8-in. holes cast in a circular pattern. The lower 12 in. of the drying column was lined with refractory, but the upper section was a plain 12-in. steel tube insulated on the outside.

The No. 5 pilot plant, illustrated in fig. 3, was constructed to study the factors of size and degradation and also to provide a simple unit for testing various coals. The drying column and combustion chamber used in plant No. 3 were incorporated in this unit, the feeding system was changed to a screw feed from a closed hopper, and a simple primary separator similar to the one employed in the No. 4 plant was included. The perforated grid above the combustion chamber was made of ¼-in. alloy plate drilled with thirty-six 17⁄64-in. holes. A constant-

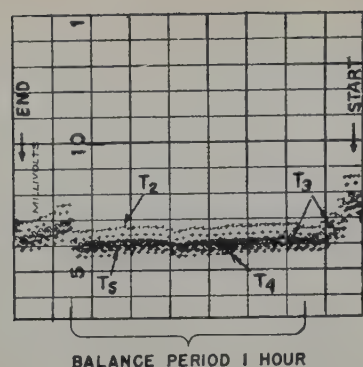


Fig. 4—Temperature data for test 30-B, pilot plant No. 5.

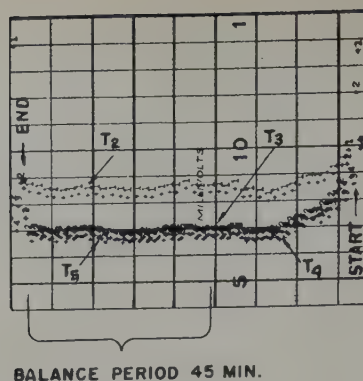


Fig. 5—Temperature data for test 29, pilot plant No. 5.

volume gas pump was used for introducing the gas and air mixtures into the combustion chamber under pressure.

The No. 5 plant has advantages for testing the drying characteristics of small batches of coal up to 1000 lb and is useful for research on the several factors affecting drying. Its operation is quite simple.

**Operating Procedure:** The procedure in making tests is approximately the same on each of the pilot plants and is described as follows with respect to plant 5, fig. 3:

The coal to be dried is prepared by crushing in a hammermill to the desired size. For routine tests,  $\frac{1}{4}$  in.  $\times$  0 size is employed. The coal is sampled to determine its size and analysis and then weighed and placed in the charging bin. The combustion chamber is heated up to about 1700°F by burning a mixture of air and gas, and when the 1700°F temperature is reached, recirculation of products of combustion is begun and the furnace temperature raised to about 2000°F. The coal feed is then started, and the desired rates of feeding and heat supply are adjusted. The system soon comes into balance, which is maintained for the period of the test. The length of a test ranges from 1 to 5 hr, depending upon the rate of feed and object of the test.

During the balanced period, routine operating data are recorded periodically. The temperatures in the combustion system and drying column are recorded continuously, and the pressures in various places, P-1 to P-5 of fig. 3, are determined by direct reading with a portable, water-filled U-gauge. The dried product is removed at regular intervals to determine the rate of yield and to test for moisture

removed. When the coal feed stops, the gas circulation is shut off, and all dried coal recovered is weighed and sampled for a material balance.

Before a test is started, theoretical heat and material balances are calculated to establish the desired rate of coal feed, the temperature and velocity of gases in the column, and rates of air, gas, and POC (products of combustion). This calculation is shown in table I, which gives typical calculations for two plants and two coals.

The approximate quantities of air, gas, and coal, calculated as shown, are then fed to the plant. The unit soon comes into operating balance, and the established rates of feed are not changed. Figs. 4 and 5 show the history of temperatures during typical balanced periods when operating plant No. 5.

**Results of Tests:** Approximately 40 tests have been made on the four fluidized drying units to study various problems of drying different grades of fine coal. Table II presents results of 14 tests showing typical experimental and operating data for seven different coals in plants 3, 4, and 5.

The static pressures in the combustion chamber and the pressure drop across the drying column are shown in table III. The chemical and physical properties of the coals before and after drying are shown in table IV.

**Discussion of Results:** Experience from operation of the three pilot plants on several grades and ranks of coal has shown that the fluidized drying process is relatively simple. The operation of the units coincides quite closely with theoretical operation as indicated by comparing the calculated data in table I with the experimental data of table II. It is con-



Table II. Summary Data on Drying of Subbituminous Coals and Lignites

Pilot plant	(1)	3	3	3	3	3	4	4
Test number	(2)	8	9-A	10-A	11-A	14	17	18
Rank of coal	(3)	Subc	Subc	Subc	Lignite	Lignite	Subc	Lignite
Source	(4)	Colorado	Alaska	Wyoming	Texas	Greece	Colorado	Texas
<b>Materials and moisture data</b>								
Coal charging rate, lb per hr = $W_1$	(5)	345	332	277	273	262	1,682	1,166
Coal charging rate, lb per hr per sq ft	(6)	1,716	1,652	1,378	1,358	1,303	2,143	1,485
Moisture in raw coal, pct (as charged)	(7)	24.8	25.5	32.6	37.6	47.3	27.4	39.2
Moisture in dried coal, pct	(8)	4.4	4.2	4.0	4.6	9.0	4.3	3.4
Dry coal recovered, total lb per hr = $W_2$	(9)	269	256	194	178	151	1,273	727
Improvement ratio by weight loss = $R_w$	(10)	1.283	1.297	1.428	1.534	1.735	1.321	1.604
Improvement ratio by moisture determination = $R_m$	(11)	1.271	1.291	1.424	1.529	1.727	1.318	1.589
Ultimate improvement ratio = $U$	(12)	1.330	1.348	1.484	1.603	1.898	1.377	1.645
Degree of drying, pct = $D$	(13)	85.9	87.3	91.3	92.0	89.0	88.1	94.5
Dust loss, pct = $L$	(14)	0.9	0.5	0.4	0.4	0.5	0.2	0.9
Inert gas used for moving coal, cu ft per lb	(15)	1.6	2.2	2.6	2.7	3.3	1.4	1.8
<b>Heating system data</b>								
Natural gas used, cfh (60-30 dry)	(16)	156	161	176	163	197	760	723
Heat supplied, MBtu per hr, net	(17)	137.3	141.7	154.9	143.4	173.4	668.8	636.2
Air used with gas, cfh (60-30 dry)	(18)	1,536	1,656	1,986	1,622	1,896	7,584	7,406
Heat supplied per lb of raw coal, net Btu	(19)	398	427	559	525	662	398	546
Hot gas used to dry coal, cu ft per lb	(20)	9.4	11.5	14.2	13.9	16.4	9.1	12.7
Products of combustion recirculated, cfh	(21)	974	1,246	1,015	1,242	1,319	4,686	4,561
Analysis of gases leaving column:								
H <sub>2</sub> O, pct	(22)	38.6	35.1	36.8	40.1	40.9	41.9	44.0
CO <sub>2</sub> , pct	(23)	6.9	6.9	6.1	6.7	6.8	6.4	6.1
O <sub>2</sub> , pct	(24)	0.7	1.4	2.5	0.8	0.4	0.8	1.1
N <sub>2</sub> , pct	(25)	53.8	56.6	54.6	52.4	51.9	50.9	48.8
Mass gas velocity in column, lb per hr per sq ft	(26)	1,597	1,827	1,897	1,909	2,187	2,010	1,992
Mean superficial velocity in column, ft per sec	(27)	11.2	12.4	14.2	13.5	16.7	14.6	14.2
Contact time of coal, sec <sup>a</sup>	(28)	117	122	153	160	174	42	36
<b>Temperatures in system</b>								
Theoretical flame, °F	(29)	2,410	2,190	2,300	2,240	2,380	2,400	2,350
Combustion chamber, °F, (Point 1)	(30)	2,070	1,915	2,005	1,950	2,060	2,080	2,105
Bottom of column, °F, (Point 2)	(31)	290	285	360	345	420	450	460
Top of column, °F, (Point 3)	(32)	275	270	335	300	335	285	300
Gas outlet from primary separator, °F, (Point 5 or 6 <sup>b</sup> )	(33)	260	255	280	250	225	240	290
Coal in receiver, °F	(34)	225	225	245	215	170		
<b>Heat balance</b>								
Net heat used, Btu per lb of raw coal charged <sup>c</sup>	(35)	321	346	465	498	620	363	498
Net heat required to dry coal, Btu per lb of raw coal <sup>d</sup>	(36)	298	315	418	459	558	327	464
Drying efficiency, excluding radiation, pct <sup>e</sup>	(37)	92.8	91.0	89.9	92.2	90.0	90.1	93.2
Overall efficiency, pct <sup>e</sup>	(38)	74.9	73.8	74.8	87.4	84.3	82.2	85.0
<b>Pilot plant</b>								
Test number	(2)	19	21	22-A	29	30-B	31-A	32-A
Rank of coal	(3)	Subc	Lignite	Subc	Subc	Lignite	Lignite	Lignite
Source	(4)	Wyoming	Greece	Colorado	Colorado	Greece	Greece	Texas
<b>Materials and moisture data</b>								
Coal charging rate, lb per hr = $W_1$	(5)	1,464	810	349	409	142	196	232
Coal charging rate, lb per hr per sq ft	(6)	1,865	1,032	1,736	2,035	706	975	1,154
Moisture in raw coal, pct (as charged)	(7)	30.2	58.8	24.0	24.8	62.0	46.0	37.6
Moisture in dried coal, pct	(8)	3.2	6.8	3.5	2.2	4.6	7.4	5.2
Dry coal recovered, total lb per hr = $W_2$	(9)	1,053	354	273	313	54	114	149
Improvement ratio by weight loss = $R_w$	(10)	1.390	2.288	1.278	1.307	2.630	1.719	1.557
Improvement ratio by moisture determination = $R_m$	(11)	1.387	2.262	1.270	1.301	2.511	1.715	1.519
Ultimate improvement ratio = $U$	(12)	1.433	2.427	1.316	1.330	2.632	1.852	1.603
Degree of drying, pct = $D$	(13)	92.3	94.9	88.5	93.2	97.0	90.6	90.8
Dust loss, pct = $L$	(14)	0.2	1.2	0.7	0.4	4.7	0.3	2.5
Inert gas used for moving coal, cu ft per lb	(15)	1.5	2.6	none	none	none	none	none
<b>Heating system data</b>								
Natural gas used, cfh (60-30 dry)	(16)	707	749	147	212	146	146	150
Heat supplied, MBtu per hr, net	(17)	622.2	659.1	129.4	186.6	128.5	128.5	132.0
Air used with gas, cfh (60-30 dry)	(18)	7,500	7,420	1,470	2,122	1,449	1,449	1,501
Heat supplied per lb of raw coal, net Btu	(19)	425	814	371	456	783	656	569
Hot gas used to dry coal, cu ft per lb	(20)	9.8	17.4	6.9	8.6	16.5	12.2	10.5
Products of combustion recirculated, cfh	(21)	3,938	3,813	762	1,168	737	783	774
Analysis of gases leaving column:								
H <sub>2</sub> O, pct	(22)	43.5	46.7	47.0	43.6	50.5	49.2	47.8
CO <sub>2</sub> , pct	(23)	5.9	6.0	5.9	6.2	5.5	5.7	5.8
O <sub>2</sub> , pct	(24)	1.5	0.7	0.8	0.9	0.7	0.7	0.8
N <sub>2</sub> , pct	(25)	49.1	46.6	46.3	49.3	43.3	44.4	45.6
Mass gas velocity in column, lb per hr per sq ft	(26)	1,917	1,939	1,266	1,791	1,299	1,303	1,308
Mean superficial velocity in column, ft per sec	(27)	13.3	14.9	9.1	13.8	9.8	9.2	9.2
Contact time of coal, sec <sup>a</sup>	(28)		19	144	111	205	231	182
<b>Temperatures in system</b>								
Theoretical flame, °F	(29)	2,400	2,530	2,530	2,480	2,560	2,510	2,530
Combustion chamber, °F (Point 1)	(30)	2,140	2,240	2,120	2,120	2,110	2,125	2,100
Bottom of column, °F (Point 2)	(31)	400	990	320	400	315	260	270
Top of column, °F (Point 3)	(32)	280	410	275	335	285	235	240
Gas outlet from primary separator, °F (Point 5 or 6 <sup>b</sup> )	(33)	270	310	255	305	250	225	225
Coal in receiver, °F	(34)	230	235	255	305	200	220	225
<b>Heat balance</b>								
Net heat used, Btu per lb of raw coal charged <sup>c</sup>	(35)	380	738	307	367	783	532	456
Net heat required to dry coal, Btu per lb of raw coal <sup>d</sup>	(36)	355	684	281	322	715	494	419
Drying efficiency, excluding radiation, pct <sup>e</sup>	(37)	93.4	92.7	91.5	87.7	91.3	92.8	91.9
Overall efficiency, pct <sup>e</sup>	(38)	83.5	84.0	75.7	70.6	79.0	75.3	73.6

<sup>a</sup> Calculated for plant 3 assuming average bed density of 10 lb per cu ft. Other tests calculated on basis of pressure drop.

<sup>b</sup> Point 5 for pilot plants 3 and 4 and point 6 for pilot plant 5.

<sup>c</sup> Total net heat excluding radiation.

<sup>d</sup> Excludes radiation.

<sup>e</sup> Includes radiation.

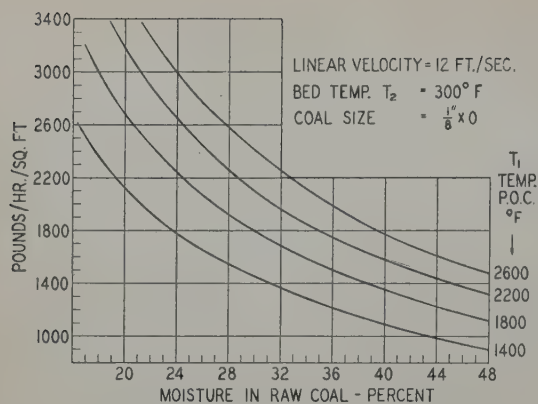


Fig. 6—Capacity of drying column on  $\frac{1}{8}$  in.  $\times$  0 coal.

cluded that the probable performance of any unit on any low-rank coals can be calculated in advance.

The data in table II indicate that the maximum practical superficial velocity to achieve 90 to 95 pct moisture removal from  $\frac{1}{4}$  in.  $\times$  0 coal is approximately 15 fps. Considerably higher velocities can be attained by reducing the percentage of drying, but for best practical results, superficial velocities between 10 and 15 fps appear to be desirable.

The rapid rate of heat transfer between the hot gases and the fluid bed is shown by comparing temperatures at points 1 and 2, shown in table II. In a distance of less than 6 in., the temperature of the gases drops nearly  $1800^\circ\text{F}$ , indicating almost instantaneous evaporation of the moisture from near the surface of the particles. The nearly uniform temperature of the fluid bed is characteristic, but there is a slight decrease in temperature toward the top of the column.

Despite the rapid exchange of heat and the high initial temperature of the hot gases, very little change occurs in the coal substance when removing 90 pct of the moisture. This is illustrated by the analysis of coal before and after drying, as shown on the moisture- and ash-free basis in the data of table V. There is a slight but insignificant reduction in volatile matter averaging about 0.5 pct. The heating value of the coal substance shows very little change, the average heating value increasing from 12,340 to 12,388, an increase of 0.39 pct. The slight devolatilization and increase in heating value indicates some liberation of  $\text{CO}_2$ , which is one of the first products of thermal decomposition of coal. By increasing the bed temperature to effect 100 pct drying and further devolatilization, the heating value

of the treated coal can be substantially increased through liberation of  $\text{CO}_2$ .

The theoretical performance of a fluidized drying unit, when operating at a superficial linear velocity of 12 fps at sea level, is shown in fig. 6. In this figure, no radiation is considered and 93 pct of the moisture is liberated. The effect of moisture in the coal and the temperature of the initial gases is clearly shown by this graph.

Attention is called to the degradation of the coal during drying. The initial size of the particle and the amount of moisture in the coal influence the extent of degradation. Reference is made to tests 22-A and 29 in table IV on a Colorado subbituminous coal where the degradation is shown to range from 10.6 to 47.6 pct. Relatively little degradation is suffered by the  $\frac{1}{4}$  in.  $\times$  0 coal used in test 22-A, but the  $\frac{1}{2}$  in.  $\times$  0 coal used in test 29 degraded five times as much. The excessive degradation of the large size is caused by rapid shrinkage of the particle which forms big cracks and reduces the strength. The agitation in the boiling bed causes the weakened pieces to disintegrate.

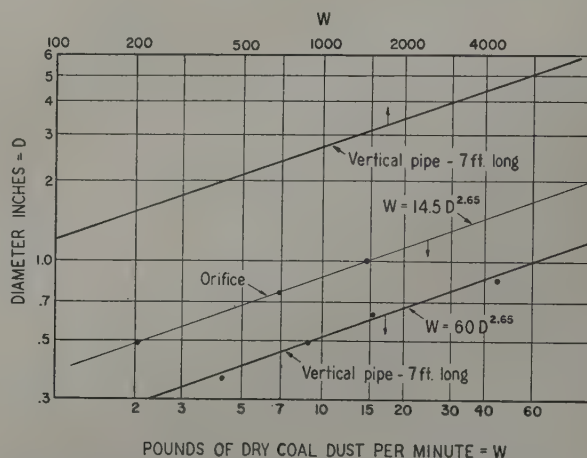


Fig. 7—Flow of  $\frac{1}{8}$  in.  $\times$  0 dry coal dust by gravity through sharp edged orifices and vertical pipes.

The extreme degradation of Ptolemais lignite shown in test 30-B is interesting. This lignite has an earthy structure, with 62 pct natural bed moisture. The particles apparently explode when heated rapidly. Similar degradation is suffered by the Megara lignite and the Texas lignite. It is probable that, because of the rapid degradation, these low-rank coals might respond favorably in a fluid bed with initial sizes of 1 in.  $\times$  0 or more.

The available heat in the several coals before and after drying is shown in table IV. The utilization factor, which is the ratio between the available heat and the gross Btu, is also given. The raw coals have utilization factors ranging from 0.65 to 0.81. This means that 65 to 81 pct of the gross heat in the raw coal can be recovered in an efficient combustion system. However, the dried coals have higher utilization factors, ranging from 0.83 to 0.85, which means that uniform high efficiencies can be obtained from the treated coal.

The physical properties of the dried dusts are similar, which is encouraging for utilization because uniformity is an essential factor in efficient plant performance. It is indicated that a boiler plant designed for burning dried low-rank coals would have

Table III.

Static Pressures in Drying Pilot Plants—Inches  $\text{H}_2\text{O}$

Plant No.	Test No.	Point 1, Combustion Chamber	Point 2, Above Screen	Point 3, Top of Drying Column	Drying Column Differential	Riser Differential
3	8		5.5		4.0	0.7
3	9-A		5		3.5	0.7
3	10-A		13		9.5	0.3
3	11-A		15		11.5	0.2
3	14	17	10	9.5	0.5	1.0
4	17	18	11.6	9.0	2.6	1.4
4	18	20	12.0	9.5	2.5	1.5
4	19	20	9.2	8.5	0.7	1.1
4	21	19	11.8	10.8	1.0	2.0
5	22-A	19	13	3	10	0.2
5	29	22	16	7	9	0
5	30-B	14	11	5	6	0
5	31-A	17	13	4	9	0
5	32-A	19	15	7	8	1



Table IV. Chemical and Physical Properties of Coals Before and After Drying

Drying plant number	(1)	3		3		3		3		3		4		4	
Test Number	(2)	8		9-A		10-A		11-A		14		17		18	
Mine	(3)	Pike View		Suntrana		Wyodak		Sandow		Megara		Sterling		Sandow	
Location	(4)	Colorado		Alaska		Texas		Texas		Greece		Colorado		Texas	
Kind of coal	(5)	Subc		Subc		Subc		Lignite		Lignite		Subc		Lignite	
Condition <sup>a</sup>	(6)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)
Proximate analysis, pct															
Moisture	(7)	24.8	4.4	25.8	4.2	32.6	4.0	37.6	4.6	47.3	9.0	27.4	4.3	39.2	3.4
Volatile matter	(8)	31.9	40.5	37.1	47.0	30.0	42.3	27.2	41.5	22.8	39.2	28.9	39.7	26.5	40.9
Fixed carbon	(9)	37.7	48.0	32.3	42.9	32.1	46.2	27.1	41.6	15.4	28.3	39.4	50.2	23.7	39.3
Ash	(10)	5.6	7.1	4.8	5.9	5.3	7.5	8.1	12.3	14.5	23.5	4.3	5.8	10.6	16.4
Ultimate analysis, pct															
Hydrogen	(11)	6.1	5.0	6.6	5.3	7.0	5.2	7.2	5.0	7.1	4.3	6.5	5.1	7.0	4.8
Carbon	(12)	50.4	64.2	48.6	63.0	46.1	65.2	40.2	61.6	25.3	45.2	52.6	69.3	36.5	58.5
Nitrogen	(13)	0.6	0.8	0.6	0.7	0.7	1.4	0.8	1.2	0.5	0.8	1.2	1.6	0.7	1.2
Oxygen	(14)	37.0	22.5	39.3	24.8	40.4	20.1	42.6	18.3	50.0	21.8	35.1	17.7	43.9	17.3
Sulphur	(15)	0.3	0.4	0.1	0.3	0.5	0.6	1.1	1.6	2.6	4.4	0.3	0.5	1.3	1.8
Btu per lb (gross)	(16)	8,470	10,840	8,450	10,900	7,980	11,330	7,000	10,740	4,250	7,590	9,050	11,970	6,380	10,200
Btu per lb (net)	(17)	7,910	10,380	7,850	10,420	7,340	10,860	6,340	10,280	3,600	7,200	8,460	11,510	5,740	9,760
Ash softening temp., °F	(18)	2,180		2,310		2,130		2,130		2,130		2,150			
Available heat, Btu per lb <sup>b</sup>	(19)	6,880	9,100	6,830	9,160	6,350	9,540	5,440	9,020	2,990	6,260	7,360	10,080	4,920	8,560
Utilization factor <sup>c</sup>	(20)	0.812	0.839	0.808	0.840	0.796	0.842	0.777	0.840	0.704	0.830	0.813	0.842	0.771	0.839
Size consist, cum. pct ret.															
On 0.530 in.	(21)														
Mesh On 0.375 in.	(22)														
(3) On 0.263 in.	(23)														
(4) On 0.185 in.	(24)														
(8) On 0.093 in.	(25)	0.1						0.2		1.5		1.7	0.5	1.9	
(16) On 0.0469 in.	(26)	20.0	7.9	21.4	13.5	26.9	6.1	31.2	2.9	18.4	2.7	22.0	12.1	28.3	5.1
(30) On 0.0232 in.	(27)	52.4	40.7	62.4	51.1	60.5	29.1	65.5	24.8	40.5	27.4	54.0	39.7	59.9	30.0
(50) On 0.0117 in.	(28)	78.3	73.6	83.7	78.6	81.1	56.6	82.4	56.3	60.2	59.1	77.7	67.8	82.7	61.0
(100) On 0.0059 in.	(29)	89.5	89.8	92.9	92.1	90.6	76.5	90.3	75.8	73.2	75.8	88.9	84.4	92.6	77.5
(200) On 0.0029 in.	(30)	96.2	97.4	97.5	97.5	97.3	90.2	95.0	88.8	85.1	86.9	95.1	94.5	99.2	88.7
Avg size, in.	(31)	0.031	0.025	0.034	0.029	0.035	0.020	0.038	0.018	0.027	0.018	0.033	0.025	0.037	0.020
Degradation, pct	(32)		19.4		14.7		42.9		52.6		33.3		24.2		45.9
Bulk density, lb per cu ft															
Modified ASTM	(33)	43.8	44.9	43.2	41.4	40.9	39.6	43.2	39.6	40.6	41.4	45.7	43.9	45.9	42.3
Compacted	(34)	49.8	51.1	51.7	49.6	49.1	45.6	49.2	45.8	54.1	52.7	51.6	50.6	55.4	50.9

Drying plant number	(1)	4		4		5		5		5		5		5	
Test number	(2)	19		21		22-A		29		30-B		31-A		32-A	
Mine	(3)	Wyodak		Ptolemais		Pike View		Pike View		Ptolemais		Megara		Sandow	
Location	(4)	Wyoming		Greece		Colorado		Colorado		Greece		Greece		Texas	
Kind of coal	(5)	Subc		Lignite		Subc		Subc		Lignite		Lignite		Lignite	
Condition <sup>a</sup>	(6)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)	(1)	(2)
Proximate analysis, pct															
Moisture	(7)	30.2	3.2	58.8	6.8	24.0	3.5	24.8	2.2	62.0	4.6	46.0	7.4	37.6	5.2
Volatile matter	(8)	31.5	42.2	19.4	43.1	31.1	40.9	32.0	41.6	18.1	44.4	23.5	40.1	27.2	40.8
Fixed carbon	(9)	28.9	41.7	14.9	33.3	39.3	48.4	37.4	49.3	13.5	35.4	16.8	28.8	24.4	37.8
Ash	(10)	9.4	12.9	6.9	16.8	5.6	7.2	5.8	6.9	6.4	15.6	13.7	23.7	10.8	16.2
Ultimate analysis, pct															
Hydrogen	(11)	6.6	5.0	8.3	4.8		4.9		4.6		4.5		4.3		4.7
Carbon	(12)	45.1	61.4	23.0	50.8		64.8		66.1		53.3		45.3		57.1
Nitrogen	(13)	0.6	0.9	0.6	1.5		0.8		0.8		1.5		0.8		1.1
Oxygen	(14)	37.5	18.6	60.5	24.9		21.9		21.2		23.8		21.5		18.8
Sulphur	(15)	0.8	1.2	0.7	1.2		0.4		0.4		1.3		4.4		2.1
Btu per lb (gross)	(16)	7,740	10,760	3,820	8,640		10,940		11,290		8,870		7,680		9,980
Btu per lb (net)	(17)	7,140	10,300	3,060	8,200		10,490		10,870		8,460		7,290		9,550
Ash softening temp., °F	(18)														
Available heat, Btu per lb <sup>b</sup>	(19)	6,170	9,050	2,490	7,170	7,210	9,210	6,850	9,580	2,160	7,400	3,210	6,350	5,060	8,380
Utilization factor <sup>c</sup>	(20)	0.797	0.841	0.652	0.830	0.816	0.842	0.811	0.848	0.624	0.834	0.715	0.827	0.774	0.840
Size consist, cum. pct ret.															
On 0.530 in.	(21)							3.6		1.1		1.8			
Mesh On 0.375 in.	(22)							10.0	1.5	4.6		4.7		1.1	
(3) On 0.263 in.	(23)							21.0	3.0	12.8		15.9	0.3	9.8	0.4
(4) On 0.185 in.	(24)							31.0	7.3	27.0		29.1	0.6	25.8	1.9
(8) On 0.093 in.	(25)	1.9	0.3	4.6		1.2	0.9	58.2	33.5	53.9	0.5	58.1	4.6	59.4	14.3
(16) On 0.0469 in.	(26)	37.6	14.5	29.4	2.3	46.0	41.3	77.9	62.9	71.6	6.9	75.0	24.8	77.6	39.5
(30) On 0.0232 in.	(27)	70.4	46.4	54.5	24.4	70.5	68.5	89.4	81.5	82.7	44.2	85.0	61.1	87.7	66.4
(50) On 0.0117 in.	(28)	86.9	73.8	73.5	54.0	86.2	85.4	95.5	91.8	90.9	71.0	91.7	81.6	93.7	83.3
(100) On 0.0059 in.	(29)	93.1	87.2	85.9	71.6	94.0	93.6	98.0	96.5	95.9	84.4	95.4	90.0	96.9	91.2
(200) On 0.0029 in.	(30)	96.4	94.7	98.9	86.9	98.0	98.0	99.5	99.1	99.6	93.7	97.9	94.9	98.7	96.2
Avg size, in.	(31)	0.043	0.028	0.038	0.017	0.057	0.051	0.166	0.087	0.137	0.025	0.147	0.039	0.133	0.053
Degradation, pct	(32)		34.9		55.3		10.6		47.6		81.8		73.5		60.2
Bulk density, lb per cu ft															
Modified ASTM	(33)	44.7	42.7	35.4	38.7	45.5	44.7		42.7	37.7	35.0	44.1	42.8	43.0	40.8
Compacted	(34)	53.0	50.5	48.8	46.7	52.8	51.9		50.0	51.4	43.6	51.9	52.7	50.7	47.2

<sup>a</sup> (1) Before drying; (2) After drying.<sup>b</sup> Burned with 30 pct excess air, and stack temperature = 500°F.<sup>c</sup> Ratio of available heat to gross Btu per lb.

Table V. The Effect of Drying as Shown by the MAF Analysis of the Raw and Dried Coals

Test No.	Mine County and State	Condition <sup>a</sup>	Proximate Analysis		Ultimate Analysis					Btu per Lb	Drying Temp., °F <sup>b</sup>
			Volatile Matter	Fixed Carbon	Hydrogen	Carbon	Nitrogen	Oxygen	Sulphur		
8.	Pike View El Paso County, Colo.	(1)	45.9	54.1	4.9	72.5	0.9	21.3	0.4	12,170	275
		(2)	45.7	54.3	5.1	72.5	1.0	21.0	0.4	12,240	
9-A	Suntrana Suntrana, Alaska	(1)	53.3	46.7	5.4	70.1	0.8	23.6	0.1	12,160	270
		(2)	52.3	47.7	5.5	70.0	0.8	23.3	0.4	12,130	
10-A	Wyodak Campbell County, Wyo.	(1)	48.2	51.8	5.3	74.3	1.0	18.6	0.8	12,850	335
		(2)	47.9	52.1	5.4	73.6	1.6	18.7	0.7	12,800	
11-A	Sandow Milan County, Tex.	(1)	50.1	49.9	5.5	74.1	1.4	17.0	2.0	12,900	300
		(2)	49.9	50.1	5.4	74.1	1.4	17.2	1.9	12,910	
14	Kokkinarea Near Megara, Greece	(1)	59.7	40.3	4.9	66.3	1.3	20.8	6.7	11,140	315
		(2)	57.9	42.1	4.9	66.9	1.2	20.6	6.4	11,250	
17	Sterling Weld County, Colo.	(1)	42.1	57.9	5.0	76.9	1.7	15.9	0.5	13,250	285
		(2)	44.1	55.9	5.1	77.2	1.7	15.5	0.5	13,310	
18	Sandow Milan County, Tex.	(1)	52.8	47.2	5.4	72.6	1.4	18.0	2.6	12,680	300
		(2)	51.0	49.0	5.4	72.9	1.4	18.1	2.2	12,710	
19	Wyodak Campbell County, Wyo.	(1)	52.1	47.9	5.4	74.5	1.0	17.7	1.4	12,800	280
		(2)	50.3	49.7	5.6	73.2	1.0	18.7	1.5	12,840	
21	Ptolemais area, Greece	(1)	56.7	43.3	5.0	66.9	1.9	24.2	2.0	11,110	410
		(2)	56.4	43.6	5.3	66.5	2.0	24.6	1.6	11,300	

<sup>a</sup> Condition: (1) Before drying. (2) After drying.<sup>b</sup> At top of column.

uniform operating characteristics despite the source of raw coal.

The dried dusts are free flowing, and since the moisture content is lower than the equilibrium value, they are hygroscopic and their tendency to pack in a bin is reduced provided moisture is excluded. Interesting experiments were made on the movement of dried dusts through orifices and vertical pipes. These are illustrated in fig. 7. It is shown that the dried dusts move through orifices similar to liquids, and a formula is derived showing relationship between size of opening and rate of discharge. The weight of dried dust that will discharge through an orifice per minute is equal to  $14.5 D^{2.65}$ ,  $D$  being the diameter of the orifice in inches. Similar experiments were made to find out how fast dried dust will move through vertical pipes. Pipes of various lengths and diameters were connected to a bin and

the discharge measured. It was found that a pipe 7-ft long had the greatest capacity, and the amount of dust, in pound per minute, that would flow through a 7-ft vertical pipe from a bin is equal to  $60 D^{2.65}$ . In other words, a vertical pipe 7-ft long will discharge about four times as much dried dust as an orifice of equal diameter. This phenomenon is due to the energy developed by the falling particles, which entrains gas and acts as a siphon. The phenomenon may be employed for efficient handling of dried dusts in large plants.

### Commercial Application of the Fluidized Drying Process

Experimental work on several pilot plants has carried the investigation about as far as feasible on a small scale. The next step is to operate a commercial-size unit to determine engineering and economic data. Fig. 8 shows a plant designed to dry 30 tons of Wyoming subbituminous coal per hour and to deliver about 20 tons of dried coal per hour. This schematic arrangement shows the essential ingredients of a plant and indicates the flowsheet and quality of materials.

The cyclone burner for generating primary hot gases under pressure may or may not be applicable for such service, and some development work would be necessary to work out a suitable burner for this drying unit. The problem of burning pulverized coal under pressure is not difficult, and existing equipment can be readily adapted to solve this problem.

The unit illustrated in fig. 8 is estimated to cost approximately \$100,000. The investment and operating costs of three units of this design for processing 2000 tons of coal per day, when removing 670 lb of water per ton, are estimated as follows, in cents per ton: investment cost, 7.0; labor and supervision, 6.2; power, 4.5; fuel, 9.8; maintenance, 1.1; total, 28.6.

These estimated costs, which do not include the cost of crushing the coal, indicate that moisture can be removed from lignite at a cost of about \$0.85 per ton of water removed. The cost will vary with the moisture content of the coal treated, but it is believed that the unit cost for large commercial plants will be in the range of \$0.85 to \$1.00 per ton of moisture removed.

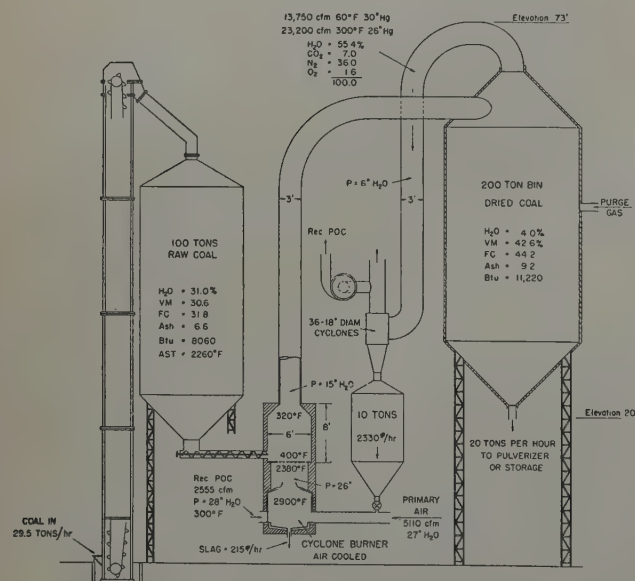


Fig. 8—Schematic design of drying unit for subbituminous coal. Capacity 20 tons of dried coal per hour.



# Roof Studies and Mine Structure Stress Analysis, Bureau of Mines Oil-Shale Mine, Rifle, Colo.

by E. M. Sipprelle and H. L. Teichman

**E**NACTMENT of Public Law 290 by the 78th Congress authorized the U. S. Department of the Interior, Bureau of Mines, to conduct an experimental program to develop the technology for obtaining oil from oil shale. In adopting and later extending this legislation, the Congress recognized the impending necessity of supplementing ground petroleum reserves with synthetic fuels. Under the provisions of this legislation, the Bureau of Mines, among other things, was charged with the responsibility of developing mining techniques, methods, and equipment for mining the oil shales of the Green River formation of Colorado, Utah, and Wyoming.

The oil shales of western Colorado are apparently richer, more accessible, and more amenable to exploitation than elsewhere in the Rocky Mountain region. The site chosen for the Bureau's Experimental mine is about 10 miles west of Rifle in northwestern Colorado. It is within a 1000-sq-mile area from which, it has been estimated, 300 billion barrels of shale oil could be produced from a 500-ft measure near the top of the formation. One hundred billion barrels of this amount could be produced from the Mahogany ledge, a 60 to 100-ft section near the bottom of the 500-ft measure. This ledge is considered to have economic importance at present.

The Green River formation was laid down as sediment in the bottom of vast, shallow inland lakes during Eocene time. The deposit is flat-lying, and there are no faults, fissures, or local rolls. Oil shale is actually a strong, tough magnesium marlstone, which will stand unsupported over relatively wide spans. These and other natural physical characteristics favor mechanized, low-cost mining, which is essential for establishment of an oil-shale industry.

It was realized from the outset that an extensive research program would be necessary to develop mining methods, equipment, and techniques for a mechanized, low-cost operation. The program was designed to include research into all the productive phases of mining, such as drilling, blasting, loading, transportation, and maintenance of the mine structure. The methods, equipment, and techniques developed as a result of this research have established a production of 116 tons per man-shift total labor at a direct cost of \$0.292 per ton.

Another important phase of the research program that has received little publicity because of its theoretical nature is study of the roofstone behavior and determination of mine structure stresses. This paper

purposes to discuss this phase of the research program.

Preliminary studies of the physical properties of the Green River oil-shale formation were made in the Barodynamics Laboratory at Columbia University during the latter part of 1945 and the early part of 1946.\* The purpose of these studies was to

\* F. D. Wright and P. H. Bucky: Determination of Room and Pillar Dimensions for the Oil-Shale Mine at Rifle, Colo. *Trans. AIME*, 181, 352; *Min. Tech.*, Nov. 1948, TP 2489.

determine the maximum size of unsupported underground openings that would be commensurate with safety and still permit the use of large, efficient mining equipment. Also to be determined were the pillar support to extraction ratio and the shape, size, and spacing of supporting pillars. Selected samples of possible roofstones near the top of the Mahogany ledge, as well as representative samples of different rock types found within the ledge, were obtained from the Bureau's oil-shale mine for these studies. The maximum safe unsupported roof span calculated from this work was 200 ft. Using a safety factor of four, it was theoretically determined that openings 60 ft wide could be advanced under a roofstone at the top of the Mahogany ledge. To support the overburden, 60-ft-sq pillars would be left in a checkerboard pattern.

From visual observations made of core samples through the selected roofstone at the oil-shale mine, it was determined that the roofstone was actually a plate 6 to 8 ft thick.

Because the calculations were theoretical and allowance had to be made for unknown cracks and fractures in the formation, openings 50 ft wide and pillars 60 ft sq were originally contemplated in the Bureau's Experimental mine. This would be the minimum allowable width that would permit use of large underground mining equipment. For lower mining costs and greater efficiency larger openings were desirable.

Different but analogous approaches were made to the problem at the Bureau of Mines Applied Physics

*E. M. SIPPRELLE, Member AIME, and H. L. TEICHMAN are Chief and Physicist, respectively, the Oil-Shale Mining Branch, U. S. Bureau of Mines, Rifle, Colo. AIME New York Meeting, February 1950.*

*TP 2942 A. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Jan. 30, 1950.*

*Papers by authors of the U. S. Bureau of Mines staff are not subject to copyright.*

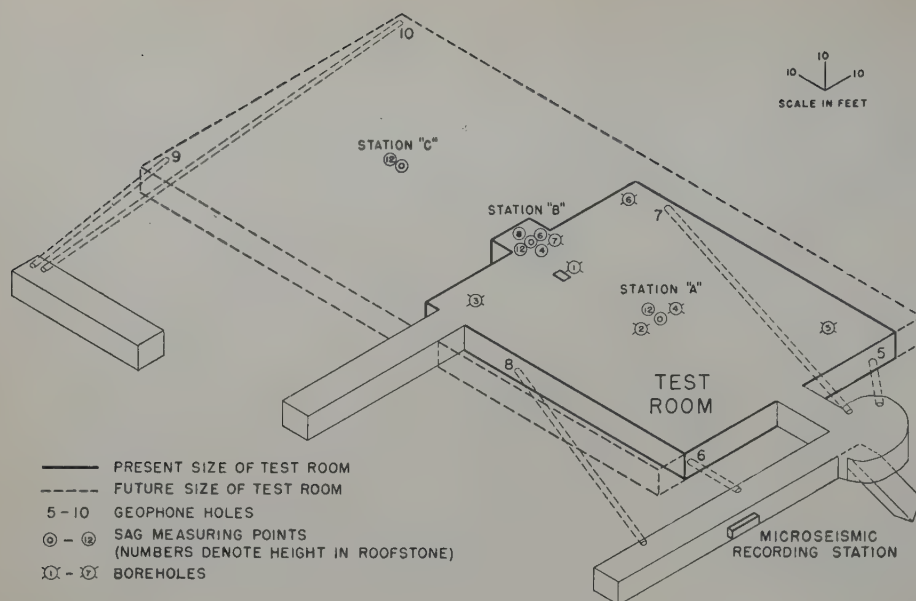


Fig. 1—Isometric drawing of test room showing present and proposed size and recording installations.

Laboratory, College Park, Md., and the Bureau of Mines Pittsburgh Station. Essentially the same results were obtained.

### Seismic Investigations

To check the laboratory data and theory and to prove conclusively the maximum size at which underground openings could be mined safely, it was decided to excavate an experimental test room 50 ft wide by 100 ft long directly under the selected roofstone. In this test room it would be possible to study the various physical properties of oil shale as a formation in place and to demonstrate the practicability and use of scientific instruments in mining research under controlled conditions.

Excavation of the 50x100-ft test room was completed in December 1946. By February 1947 specialized equipment had been installed, and the first scientific studies were started. The procedure has been to widen the room in increments of 10 ft, while using the specialized equipment to make daily observations of the behavior of the roofstone. Each new width was observed for a period of time until it was definitely determined that no excessive stresses were developing. The test room was widened from 50 to 60 ft in June 1947 to 70 ft in August 1947 and to 80 ft in November 1948 (fig. 1). During this period it was observed that small hairline cracks developed in the roofstone, and air slacking of a ½-in. layer on the roofstone occurred to a minor degree. Aside from the hairline cracks and air

slacking, no other failures have occurred. During the entire period of widening from 50 to 80 ft, while under constant observation, no indications of excessive stress have been observed.

One generally accepted method of predicting imminent failure in underground mine openings is by listening to the ground "work," that is, make audible popping and cracking noises. These noises are caused by failure of rock crystals and actual cracking of the rock formation when stressed beyond the breaking strength. The intensity and frequency of these noises offer a semiquantitative method of predicting excessive stresses in underground mine structures. It was reasoned that if these noises could be detected while still in the subaudible range, the maximum safe roof span in the oil-shale formation could be determined.

Seismic equipment (fig. 2) was adapted to detect and record these subaudible noises caused from cracking or rearrangement of minute rock crystals subjected to pressure. The equipment consists of geophones, amplifiers, and recorders. The geophone is a metallic cylinder, 1¼ in. in diam and 8 in. in length, which houses a small Rochelle salt crystal. Subaudible rock noises are picked up and transformed into small electric currents by the salt crystal. These electric currents are amplified and recorded as pulses on the microseismic recorder tape. Four such geophones were installed in the roofstone of the test room near the four corners where maximum stress is most likely to occur. Each geophone makes a separate record. Minute rock noises can be detected by a geophone in solid ground from a radius of 50 to 70 ft. The recording from a geophone closest to a minor local disturbance will show greater frequency and intensity than one farther away. Disturbances of a general nature would be recorded nearly equal in frequency and intensity from all geophones. Records of the rock noises are made each day and the results tabulated.

From a study of these data it has been possible to establish a definite noise level (number of noises per hour) pattern that gives a qualitative indication of the stress conditions in the test-room roofstone. The average normal noise level is less than five noises per hour. This average has been established from the seismic data recorded over a period of

Table I. Typical Noise Levels Compared to Sag Rates

Noises per Hour	Approximate Avg Sag, In.	Remarks
0 to 5	0.0001	Average noise level of test room with no mining activity. Considered very quiet.
20 to 60	0.001	Considered noisy with some possibility of approaching the elastic limit. This is approximate rate immediately after widening is completed. The rate dropped to a normal level within a few days.
100 to 1,000+	0.01	During the widening of test room. Would be extremely dangerous if not due to blasting. This noise level decreased rapidly and several days after blasting the rate dropped to a normal level.



several months between each widening of the test room. During these periods no mining activity has been conducted in the vicinity of the test room. Immediately after blasting in the test room or its vicinity and for a short period after each widening, the noise level has been as high as several hundred noises per hour. In all cases, so far, the noise level has dropped to the normal level within a few days. As long as this condition exists, the stresses are well within the elastic limits of the rock. If the noise level should suddenly start increasing or if it should fail to return to the normal level after blasting or widening, it could be assumed that the elastic limit of the rock was being approached.

### Subsidence

It is an established fact that materials deform when subjected to stresses over long periods of time. This deformation, when smaller than a certain rate, does not materially affect the strength of the material. Laboratory tests have shown that shale has similar plastic properties. From these facts it was reasoned that the test-room roofstone would subside or sag to some extent because of its own weight and the pressure of overburden above it. Early in the test-room studies it was decided to measure this subsidence as accurately as possible and to correlate the subsidence rate with the seismic noise level and other stress data that could be obtained.

The original subsidence station was installed to measure the sag in the immediate roofstone at the center of the room. Stainless steel studs were cemented into the roof and floor at the center of the room. Periodic measurements are taken between these two points with an extensometer (fig. 3), an instrument made of stainless steel with an Ames dial on one end and a thermometer for making temperature corrections at the time of reading. Measurements taken with this instrument are accurate to 0.001 in. As the test room was widened, it became apparent that two main factors were affecting the sag. One was due to the width of the opening, in which case the sag was immediate and relatively large. The other was due to plastic flow causing a fairly constant but relatively small sag.

The greatest sag rate (0.13 in.) occurred between August 1 and August 25, 1947, when the room was widened from 60 to 70 ft. After widening, the rate



Fig. 2—Engineer taking readings at microseismic recording station.

(Photograph by U.S. Bureau of Mines)



Fig. 3—Engineer making sag measurements with extensometer at Station "A."

(Photograph by U.S. Bureau of Mines)

of sag decreased rapidly and stayed at a slow rate for several months, gradually decreasing with time and finally approaching zero. A graph of the sag from the time measurements were started to the end of 1949 is shown in fig. 4.

The magnitude of the immediate sag during widening from 50 to 60 ft and from 70 to 80 ft was less than from 60 to 70 ft. The reason for this is not definitely known. It could be caused by the longer time element and greater number of blasts during widening from 60 to 70 ft. Evidently blasting procedure has some effect on the sag. Exactly how much has not been determined.

The total sag measured from November 1948 to January 1950 (the period after widening from 70 to 80 ft) was only 0.10 in.

**Roof Sag and Noise-Level Correlation:** Correlations were made between roof sag and noise level. It was found that with each increase in sag rate a corresponding increase in noise level was observed.

Some typical noise levels compared to sag rates are shown in table I.

**Apparent Thickness of Roofstone:** From the subsidence data, the dimensions of the room, and the elastic properties of the roofstone, it was possible to calculate the apparent thickness of the roofstone by using the beam formula. Since there are several cracks running across the roofstone, using the beam formula instead of the plate formula will give results toward a safer value and will also conform more with the actual conditions.

From the beam theory

$$c^2 = \frac{p}{128E(d_2 - d_1)}(L_2^4 - L_1^4);$$

where  $c$  =  $\frac{1}{2}$  thickness of roof,  
 $p$  = density of rock = 0.08 lb/in.<sup>3</sup>,  
 $E$  = Young's modulus =  $3 \times 10^6$  lb/in.<sup>2</sup>,  
 $d_1$  = deformation before increasing room width,  
 $d_2$  = deformation after increasing room width,  
 $L_1$  = room width before widening,  
 $L_2$  = room width after widening.

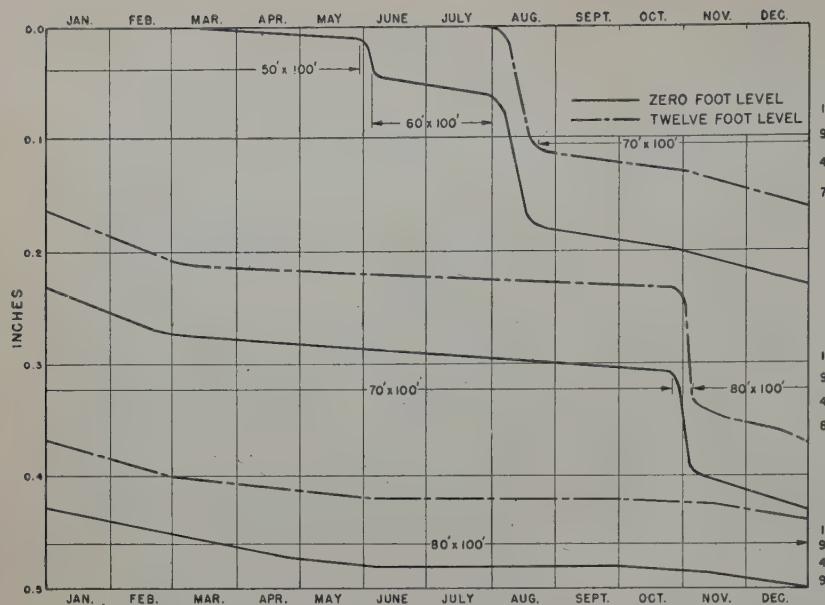


Fig. 4—Test room roof sag, Station "A."

For 50 to 60-ft widths,

$$c^2 = \frac{0.08}{128 \times 3 \times 10^6 \times 0.03} (60^4 - 50^4) 12^4,$$

$$c = 31.08 \text{ in.} = 2.58 \text{ ft,}$$

$$2_c = 5.16 \text{ ft}$$

For 60 to 70-ft widths,

$$c^2 = \frac{0.08}{128 \times 3 \times 10^6 \times 0.12} (70^4 - 60^4) 12^4,$$

$$c = 10.68 \text{ in.} = 1.64 \text{ ft,}$$

$$2_c = 3.28 \text{ ft}$$

For 70 to 80-ft widths,

$$c = \frac{0.08}{128 \times 3 \times 10^6 \times 0.1} (80^4 - 70^4) 12^4,$$

$$c = 88 \text{ in.} = 7.33 \text{ ft,}$$

$$2_c = 14.66 \text{ ft}$$

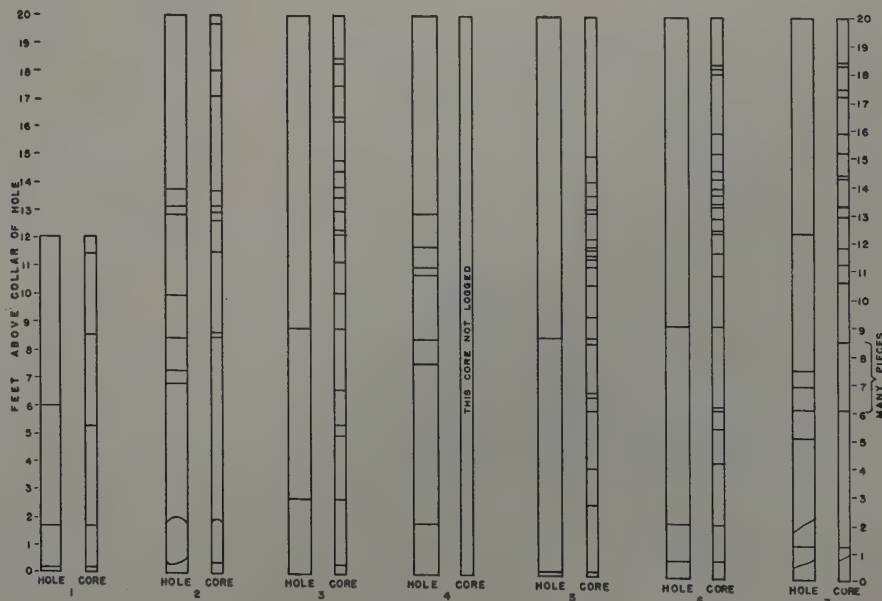
Average of all three determinations

$$\text{Avg} = (5.16 + 3.28 + 14.66) / 3 = 7.7 \text{ ft.}$$

**Differential Sag:** To check this apparent thickness and locate any separation in the roofstone, differential roof sag measurements were started in July 1947. A special 12-ft extension stud was designed to fit into an AX diamond drill hole and to clamp into the roof 12 ft vertically up from the surface. Differential sag measurements were then obtained by subtracting the sag at the 12-ft level from the sag at the 0-ft level, thereby showing any separation that occurs between these two measuring points. Sag measurements 12 ft above the roofstone indicated that a maximum differential of 0.025 in. occurred during the middle of August 1947. This differential decreased with time, and at the end of 1949 the separation in the roofstone layers was only 0.002 in. This differential sag is only what has been measured. The separation is probably greater, since it is likely that separation to some extent occurred before differential measurements were started. This surmise is substantiated by other tests made in examining bore holes in the test room, which will be explained later in this report.

To supplement differential sag measurement, locate

Fig. 5—Stratoscope and core log data from diamond-drill holes in test room roofstone showing location of cracks.





separations in the roofstone accurately, and gain other important information, it was desirable to examine the roofstone layers in the test room visually. One way of doing this is by using an instrument called a boroscope, with which vertical drill holes can be examined for cracks. An instrument of this sort, but called a stratoscope, was obtained from the Roof Control Section, Coal Mine Inspection Division, Bureau of Mines, College Park, Md. Seven vertical NX diamond drill holes averaging 21 ft in depth were examined, and photographs were obtained of cracks in the roofstone of the test room.

A crack 2 ft above the roofstone was observed in boreholes 1, 2, 4, and 6 and in a small raise near the center of the test room. A crack 8½ ft above the roofstone was observed in boreholes 2, 3, 4, 5, and 6. Another crack 13 ft above the roofstone was observed in boreholes 2 and 4. Other minor cracks were observed in some of the bore holes, but correlations with other holes could not be made. A summary of these data together with core logs from the same holes is shown in fig. 5.

Two drill holes 9 ft deep were examined in the roofstone of the Underground quarry. These were percussion drill holes, and the walls were rough, making it difficult to determine accurately the location of cracks, if any. There was some evidence of a small crack 2 ft from the roofstone in one of the holes. It is planned to repeat the examinations in boreholes in the Underground quarry roofstone at some future date.

### Stress Measurements

In addition to the seismic and subsidence information, it was desirable to determine the absolute pressures in the mine structure. A method for obtaining these measurements was devised based on the theory of calculating the stress by the relief of strain.

The procedure followed is to select a site relatively free from blasting fractures. A surface about 4x5 in. is made flat and smooth by sanding. An electric strain gauge of the Baldwin SR4-type is bonded to the surface (fig. 6), and a zero reading is obtained with a strain indicator. Using a specially designed diamond saw and with the gauge in place, a prism section about 3x4 in. to a side is sawed out, containing the gauge (fig. 7). A reading of the strain relief is then obtained (fig. 8). In the laboratory, the moduli of elasticity are determined from the sawed-out specimen. From the moduli of elasticity and the strain relief, the original stresses in the specimen are calculated.

A number of strain measurements were made during 1949 in the oil-shale mine. Two types of measurements were made; one where the site tested was under compression and the other where the site was under tension. So far, all tension tests have proved unsuccessful, largely because tension was relieved by tiny cracks in the specimen. Compression tests, however, have yielded very encouraging results. Many of the measure pressures obtained came within 10 to 15 pct of theoretically calculated values.

For a 45° Rosette strain gauge where  $e_1$ ,  $e_2$ ,  $e_3$  are the measured strains in the 0°, 45°, and 90° directions, respectively, the principal strains are given by:

$$e_p = A + B,$$

$$e_q = A - B,$$

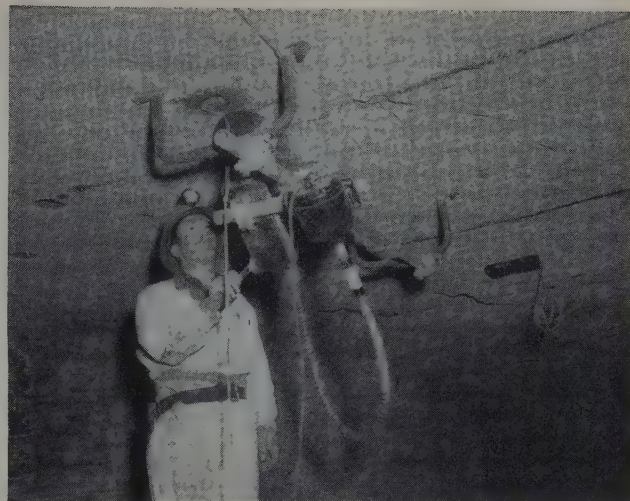


Fig. 6—Engineer preparing to saw a strain specimen from the test room roofstone. Strain gauge may be seen bonded to the roofstone.

(Photograph by U.S. Bureau of Mines)

where

$$A = \frac{e_1 + e_3}{2},$$

$$B = \frac{\sqrt{2}}{2} \sqrt{(e_1 - e_2)^2 + (e_2 - e_3)^2}$$

The principal stresses are given by:

$$p = \frac{E (e_p + \nu e_q)}{1 - \nu^2},$$

$$q = \frac{E (e_q + \nu e_p)}{1 - \nu^2},$$

where

$E$  = Young's modulus

$\nu$  = Poisson's ratio

The directions of the principal strains are given by:

$$\tan 2 \theta_{p,q} = \frac{2e_2 - e_3 - e_1}{e_1 - e_3},$$

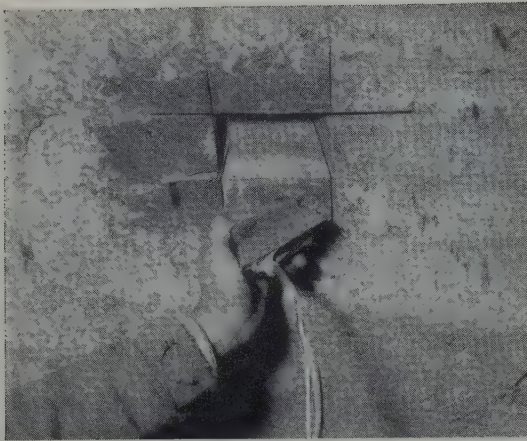
where  $\theta_{p,q}$  is the direction of  $e_p$  and  $e_q$  as measured from  $e_1$  and where  $e_p$  must make an angle of less than 45° with the algebraically greater normal strain  $e_1$  or  $e_3$ .

If a 60° Rosette strain gauge is used, slightly modified formulas are employed for making the computations.

Before field tests were made at Rifle, Colo., laboratory tests were conducted at College Park, Md., during September and October 1947 to determine whether the above method was practical.

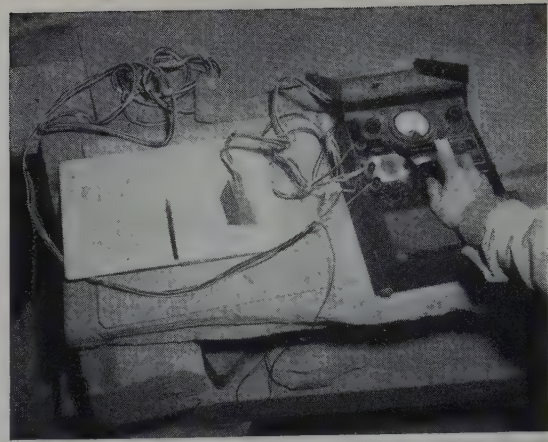
Two blocks of different materials were used. One was an oil-shale block 7x7x10 in. and the other a marble block of the same dimensions. An SR-4 electric strain gauge was cemented at the center of one of the faces of each block. Each block was individually loaded in a Baldwin Southwark compression table to simulate a face in a mine under compression. A prism section containing the strain gauge was sawed out and the strain relief measured with a strain indicator. The moduli of elasticity were de-





**Fig. 7—Strain specimen being removed from the roofstone after sawing.**

(Photograph by U.S. Bureau of Mines)



**Fig. 8—Instrument for measuring expansion or contraction of strain specimen.**

(Photograph by U.S. Bureau of Mines)

terminated by standard methods and the stresses were computed and compared to the applied stresses. Remarkable agreement was noted. The following table shows the results:

Rock Type	Applied Stresses, Lb per Sq In.	Computed Stresses, Lb per Sq In.	Error, Per Cent
Marble	2430	2470	1.6
Shale	2040	1980	3.0

During 1948 and 1949 a number of field tests were made at Rifle, Colo., using practically the same procedure as that employed at College Park, Md., except that stressed mine faces were used instead of the stressed blocks. Locations for the tests were selected where the stresses at the test position could be computed from the overburden and shape of the opening. Measured stresses could then be compared to actual stresses. Several tests had to be discarded because of breakage of the specimen while being

**Table II. Results Obtained in 10-ft Drift of Nearly Circular Shape**

Test Position	TP-1	TP-2	TP-3
Measured vertical compressive stress	680 psi	894 psi	1269 psi
Static vertical compressive stress	328 psi	328 psi	328 psi
Stress concentration factor due to shape of opening	2.1	2.7	3.8
Theoretical stress concentration factor due to shape of opening	3.0	3.0	3.0

**Table III. Results Obtained in a Long Raise of Square Shape with Rounded Corners**

Test Position	Measured Vertical Stresses, psi	Static Load, psi
TP-5	565	491
TP-6	71	491
TP-8	665	491
Avg	433	491
TP-10	59	538
TP-11	1148	538
TP-12	438	538
TP-13	734	538
Avg	595	538

sawed out. Several others have not yet been computed because of the difficulty in obtaining the elastic constants. The results obtained in a 10-ft drift of nearly circular shape are shown in table II.

The results obtained in a long raise of square shape with rounded corners are shown in table III.

## Conclusion

As a direct result of the information obtained from the roof studies and the mine-structure stress analysis, openings in the Bureau's Experimental mine have been increased from 50 to 60 ft wide, leaving 60-ft-sq pillars to support the overburden.

It is planned to continue this phase of the research program and to improve the equipment methods and techniques so that more accurate data may be obtained. The test room is to be lengthened to 200 ft and widened in increments of 10 ft to determine the actual maximum safe roof span. Additional recording stations will be installed as the size of the room is increased.

Instruments developed for measuring mine-structure stresses will be installed in the Experimental mine to determine the roofstone behavior and the increase of pressure on the supporting pillars as the mine openings are advanced. From the information obtained over a period of time, it should be possible to predict what will happen in a large-scale commercial operation.

The equipment and procedures for roofstone studies and mine-structure stress analysis developed by the Bureau of Mines should find wide application throughout the mining industry in predicting safe mine-structure design.

## Acknowledgments

E. D. Gardner, former Chief of the Oil-Shale Mining Branch, and Tell Ertl, former Chief of the Oil-Shale Mining Section, were instrumental in starting the research program. F. D. Wright, Mining Engineer, Bureau of Mines, and P. B. Bucky, Professor of Mining, School of Mines, Columbia University, and consultant to the Oil-Shale Mining Branch, completed most of the early laboratory studies. Full credit must be given to Leonard Obert, Physicist, College Park Station, Bureau of Mines, for equipment and technical advice. All members of the Oil-Shale Mining Section have contributed to the theory and application of the program.



# An Unusual Test Of the Accuracy Of Well-Surveying Methods

by S. H. Williston

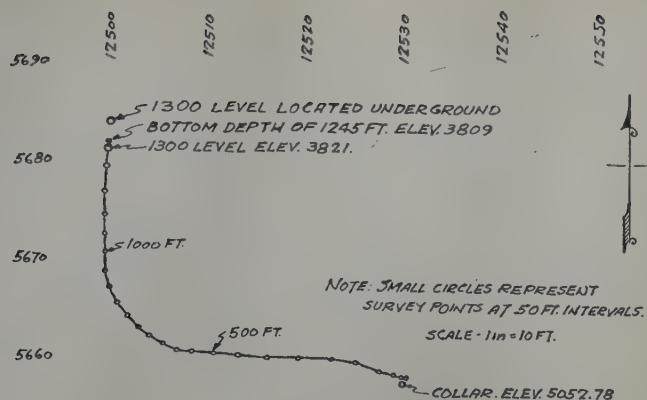


Fig. 1—Horizontal projection of survey churn drill hole No. 201.

Location	Coordinates and Elevations			Survey No. 1395
	Latitude	Departure	Elevation	
Collar	5657.71	12519.88	5052.78	
Bottom (surveyed from top)	5681.98	12489.72		
Depth of 1233 ft (surveyed from top)	5681.25	12489.64		
Depth of 1233 ft (located underground)	5684.16	12490.14	3820.86	

IT IS not often that bore hole surveys can be checked by actual civil engineering methods. A recent Arizona survey was checked by normal surveying methods and the comparison of the results should be of value to both oil and mining men.

During the summer of 1948 the Phelps-Dodge Corporation, at its Copper Queen property near Bisbee, Ariz., drilled a 1245 ft, 8 in. diam, churn drill-hole in a mineralized area and cased part of it, intending to use it to transfer mill tailings for stope fill. The hole, as frequently occurs, was not straight, and, in endeavoring to locate the bottom in the underground workings, they found no evidence of the hole at the underground coordinates directly below the surface location.

The noise of the drilling tools was reasonably clear, but the direction of sound was uncertain. Preliminary tests with available equipment were not successful in locating the bottom of the hole. Because of the mineralized character of the area and the fact there was some casing in the hole, any magnetic method of well surveying would give results of doubtful value.

Sperry-Sun gyroscopic well-surveying instruments were finally used to locate the bottom of the hole. These instruments consist of a gyroscope to determine azimuth and either a pendulum or bubble inclinometer. A multiple shot camera photographs both instruments on a single film and superimposes the photograph of a watch. Coordination of depth with time at the surface makes it possible to select the corresponding picture for any depth.

After making several runs of the empty instrument housing from the top of the hole to the bottom to make sure there were no obstructions in the hole, three surveys on wire line were completed during the afternoon. The three surveys, in which readings were taken at different points in the hole on each survey, were computed and gave the following locations of the bottom of the hole in relation to the surface collar: survey No. 1—24.92 N, 30.30 W; survey No. 2—24.24 N, 31.11 W; survey No. 3—26.54 N, 27.72 W. Then the data from the three surveys were combined into a single set of calculations which gave a location for the bottom of the hole: combined surveys—24.27 N, 30.16 W. (Fig. 1.)

Immediately upon the determination of the coordinates at the bottom of the hole, a drift on the 1300 ft level was started toward the indicated loca-

tion some 38 ft northwest of the coordinates of the surface location. The bottom of the hole was located within the drift round in which it was expected, and the transit survey run to the actual location of the hole indicated N 27.18, W 29.71. This shows a discrepancy between the well survey and the transit survey of 0.45 ft in the westerly direction and 2.91 ft in the northerly direction. All surveys, both gyroscopic and transit, fell well within the width of an ordinary drift. While this is satisfactory for almost any and all mining requirements, a theoretical examination was made as to reasons for the discrepancy.

A study of the course of the hole indicates that considerable right turn or spiral existed, and in all probability the surveying instrument was pulled out of alignment while traversing the turn by approximately 0.05 ft at the top and another 0.05 ft in the opposite direction at the bottom of the instrument. If such an allowance were to be made, the survey calculations would almost exactly correspond with those determined by transit. This sort of discrepancy would be minimized by the use of stabilizing guides.

It is unfortunate that physical laws probably effectively prevent the use of gyroscopic instruments in EX and AX diamond drill holes. The directive power of a gyroscope falls off inversely at some rate between the third power and the sixth power of the diameter. Present instruments can be run in casing 5¾ in. ID or over and might be adapted to somewhat smaller diameters, but the difficulty of reducing these diameters to 1¼ in. or 2 in. is almost insurmountable at the present time.

## Acknowledgment

The author wishes to express his appreciation to the Phelps-Dodge Corporation for permission to publish this article, and to the Operating and Engineering Departments for their cooperation on the survey; also to Donald Hering, of the Sperry-Sun Well Surveying Co., who actually made the survey and calculated the results.

S. H. WILLISTON, Member AIME, is Vice President and Treasurer, Cordero Mining Co., San Francisco, Calif.

AIME St. Louis Meeting, February 1951.

TP 2945 A. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received April 14, 1950.

# Preliminary Report of Massco Circuitron

by Allen E. Craig, William J. Tait, and E. P. McCurdy

**The Circuitron herein described applies current from the classifier motor circuit and energy from the sound of grinding media to move an oscillating disc. The disc through a photoelectric cell controls the speed of the ore feeder and consequently the amount of ore delivered to the mill.**

**T**HE Telluride Mines, Inc. at Telluride, Colo., is operating a flotation concentrator recovering lead, zinc, and copper with some gold and silver. The nominal capacity is 500 tons per 24 hr. The ore, which comes from several different mines, is characteristic of the district, changing from extremely tough rhodonite to soft gouge. The ore is crushed and washed, recrushed, and delivered to the mill bin at about  $\frac{5}{8}$  in. to serve as ball mill feed.

## Details of Grinding Circuit

The material from the fine-ore bin is carried over a belt feeder driven by a Reeves vari-speed device. The belt discharges to a No. 86 Marcy ball mill, which operates in closed circuit, with jigs and a 6-ft Dorr classifier.

A Massco density controller holds the density of the classifier overflow at a predetermined point. The overflow, which contains about 5 pct + 48 mesh, goes to the flotation section.

## Manual Operation

Before the adoption of automatic control for the grinding circuit, this work was all done manually by the operator. The bin gate was set to deliver 40 to 50 lb of ore per lineal foot of belt, and the speed of the belt was adjusted by the operator to deliver at the rate of about 500 tons per 24 hr. As the ore changed in hardness, specific gravity, or in the amount of moisture, or, as the size of the feed changed due to bin segregation, crusher setting, or other causes, the operator made corresponding changes in the rate of feed by manipulation of the Reeves control. Sometimes these changes occurred so rapidly that they caused sharp, pronounced

surges, which created very considerable unbalance from normal conditions in the grinding circuit and, more critically, in the flotation section. Since it is not possible for the operator to anticipate or follow such changes closely, a very considerable unbalance from normal conditions may occur, which will require a drastic change in the rate of feed. These surges upset the fineness of grind and interfere radically with the balance between tonnage and reagent control. All of this tends to reduce the grade of concentrates, increase the loss in the tailing, and thus decrease the recovery of values.

## The Circuitron

Some years ago, The Mine and Smelter Supply Co., in conjunction with the operators at the Telluride Mill, undertook to build a device that would give close control of the rate of feed from the ore bin and to coordinate with this a related device that would regulate the amount of sand load on the classifier rakes. Emphasis was strongly laid on the fact that such a control must apply to the entire grinding circuit and must include the feed to the ball mill, the classifier sand return, and the density of the classifier overflow. If this could be done successfully, we felt that the circuit could be operated under best possible conditions, with compensations

---

ALLEN E. CRAIG, Member AIME, WILLIAM J. TAIT, and E. P. McCURDY are Assistant Manager, Chief Engineer, and Electrical Engineer, respectively, The Mine and Smelter Co., Denver, Colo.

AIME New York Meeting, February 1950.

TP 2948 B. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Jan. 30, 1950.



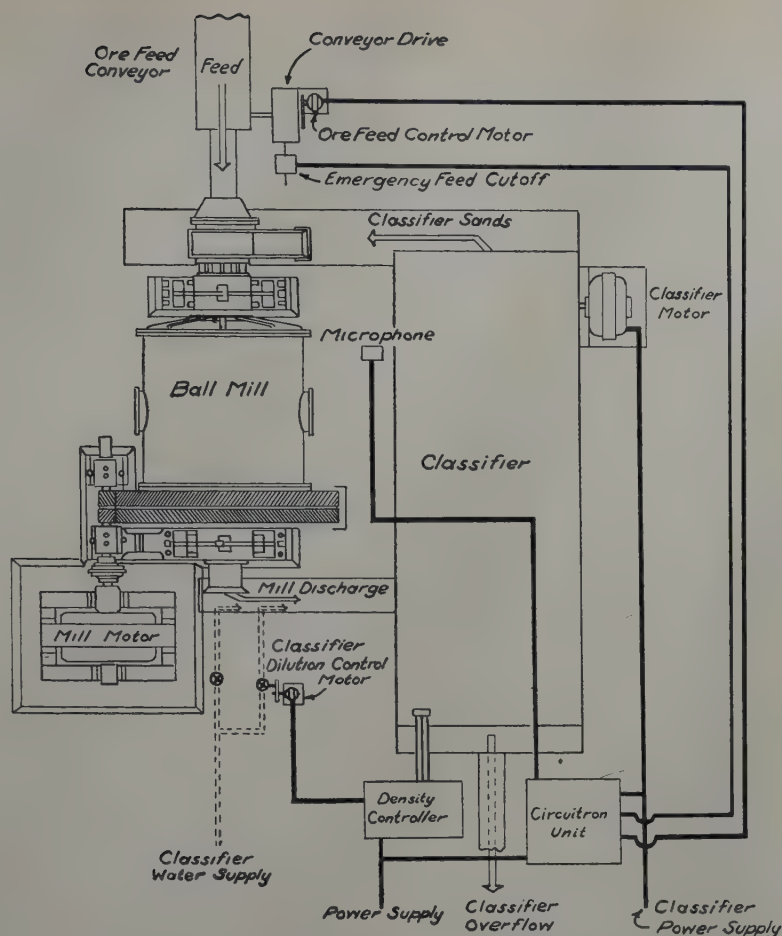


Fig. 1—Massec grinding circuit control.

being made continuously for changes in feed characteristics and feed rates. Further, we felt that if changes occurred, they could be noted and regulated before they became too great and, consequently, corrections could be made more evenly and gradually than could be done by manual control. Our desires in this respect have materialized, and the Circuitron has, in a very large manner, fulfilled our greatest hopes.

The Circuitron maintains constant control at all times so that as changes become necessary they are immediately noted and regulated before any great unbalance occurs. The control is so positive and the changes in regulation are made so gradually that surges are entirely eliminated. There are, indeed, occasions when the ore changes so quickly that the regular adjustments are too small for compensation, and there is a tendency to overload the circuit. Under such emergency conditions, our Circuitron actually stops the feed belt for a short interval, which may be from 5 to 80 sec. This complete cut-off allows the Circuitron to regain control but does not last long enough to interfere with the otherwise smooth operation. The tremendous advantage of this device is its positive control and its ability to make necessary adjustments so gradually that they are hardly noticed in the general mill operation. This is not possible with manual operation as it is now generally practiced.

We call this device the Massec Circuitron. Its position in the grinding circuit is shown in fig. 1. We have experimented for a long time and now have

developed a control using electronic tubes, which gives very sensitive regulation and trouble-free service for long periods of time. Tube replacements can be made as required with a minimum amount of delay or effort. Such replacements become necessary only after months of service.

#### Control of Classifier

It is possible to imagine a condition in the grinding circuit in which the ball mill may be noisy and calling for more feed, but the classifier is loaded so that additional feed will be detrimental. There is no point in feeding more ore to the circuit because by so doing we simply increase the circulating sands or deliver a coarser product to the flotation section. So it is of vital importance to control the amount of sands returned from the classifier as well as to regulate the conditions existing within the ball mill itself. Naturally, the control of conditions, both in the ball mill and in the classifier, is based ultimately on the control of the amount of new ore delivered from the bin. When we control the classifier, we do, in fact, also affect conditions in the ball mill and, conversely, when we control the ball mill, we naturally affect conditions in the classifier. Nevertheless, two entirely separate sources of control are incorporated in our device to give control of the complete circuit.

We take the energy from the classifier motor through a transformer to the current coil of a wattmeter. The current delivered to this wattmeter will vary depending on the sand load on the classifier rakes. The wattmeter is modified to use a spring

tension and to reflect this variation in oscillation rather than in rotation of its disc. If the sand load on the classifier is constant, the wattmeter disc, under spring control, will oscillate as the rakes move over a definite arc an equal distance on each side of a central point. As the sand load increases or decreases, the swing will be greater to one side or the other side of the center point as the pull of the current in relation to the pull of the spring becomes greater or less.

### Control of Ball Mill Load

If a microphone is placed near the shell of the ball mill, the sound of the balls striking on the steel liners will generate a small amount of audio energy. There is, however, interference from conflicting sounds such as that given by the ball mill gear, repair work in the vicinity, and from other outside sources so that, for our purpose, it is essential to narrow the range of the microphone to select only the desired sound of the balls on the steel liners. This is done by insulating and shielding the microphone with the result that a very small voltage, about 1/10 of a volt, is picked up and transmitted to an amplifier to give a resultant 100 v. The 1/10 v picked up by the microphone represents a sound too small to be distinguished by the human ear. By amplifying this to 100 v, we increase the sensitivity of the instrument so that the necessary adjustments and control can be made to a very fine point.

After amplification, the current is rectified and the direct current component is used to control the output of a generator tube, which delivers an alternate current voltage to the potential coil of the wattmeter. This voltage is proportional to the sound of the ball charge in the mill.

### Operation of the Circuitron

The wattmeter now receives current from the classifier motor and voltage from the mill sound as transmitted through the microphone.

The wattmeter disc carries a steel shutter placed so that it will intercept a light beam which plays on a photoelectric cell. The photoelectric cell operates a two-pole, double-throw, heavy-duty relay. One section of the relay is used to change the direction of rotation of a reversible control motor, which operates the Reeves vari-speed drive on the feed belt delivering ore from the bin to the ball mill.

The circuit is adjusted so that when the light is acting on the photoelectric tube the reversible control motor will turn clockwise and increase the feed to the mill. When the shutter interrupts the light beam, the motor will turn counter-clockwise to reduce the feed rate. When the mill is operating properly, and no change in feed is required, the shutter will swing equal distances from the center so that the relay contacts are closed on one side and then on the other side and the Reeves control motor oscillates in balance.

When the ore becomes softer or finer, the mill becomes noisy, and the amount of voltage generated is increased. The torque, resulting from the increased voltage, is also increased to give a stronger pull against the wattmeter disc, and unobstructed light plays on the photoelectric cell for a greater part of the time to cause an increase in feed rate. As the mill becomes loaded again, and the voltage is reduced, the torque is reduced and the shutter swings back to intercept the light and reduce the feed.

In a similar manner, as the sand load on the classifier rakes increases, the amount of current drawn by the motor increases to pull the wattmeter shutter into the light beam and to reduce the feed. Conversely, as the classifier sand load lessens, the current is reduced and the spring tension pulls the shutter out of the light beam to increase the feed.

### Overload Protection

The double-throw relay actuated by the photoelectric cell has another pair of contacts which are wired to a thermal tube. When the light beam is interrupted, this tube acts as an overload unit. If the mill becomes overloaded to such an extent that it cannot be controlled in the usual manner, with enough rapidity for safety, then this overload thermal unit opens the electrical circuit to stop the feed conveyor and actually cuts off the flow of ore to the mill for a few seconds.

### Circuitron Operation

A typical log of the movement of the automatic control is tabulated below:

8:30 a.m. Emergency cut-out 10 sec. Feed on 10 sec., off 30 sec., on again, in balance, for 5 min.  
8:35 a.m. Off 10 sec., on, in balance, 10 min.  
8:45 a.m. Coarse ore on the feed belt. Circuitron reducing feed.  
8:50 a.m. Feed still being reduced, off 55 sec.  
8:53 a.m. Still reducing.  
8:55 a.m. Off 5 sec., on 10 sec., off 20 sec., on and balanced.  
9:05 a.m. Fine ore on the belt—Circuitron increased feed.  
9:10 a.m. Feed increased to same proportion as at 8:45 a.m.  
9:15 a.m. Control unit in balance.  
9:18 a.m. Reducing feed.  
9:20 a.m. Reducing feed.  
9:21 a.m. Feed off 15 sec., then balanced.

### Conclusion

The length of time over which this device has been performing satisfactorily is comparatively short, and we have not had a full opportunity to gather any amount of operating detail.

We do know definitely that the Circuitron gives the following advantages: (1) Surges are eliminated, and necessary changes in feed rates are brought about gradually, (2) operating labor is reduced to a very marked degree, (3) tonnage is definitely increased.

From future study, we hope to gather data which will indicate the amount of tonnage increase. We hope, also, to prove an increase in the value of concentrate and a reduction in tail loss, thereby proving that the Circuitron will pay for itself in a reasonable time.

### Acknowledgments

In the development of this Circuitron, we have had a great deal of valuable help from John Goetzcke, of the Mine and Smelter staff, and Olin J. Hurley in correlating the electronic principles to the mill operations.

We are deeply indebted to H. S. Worcester who, by his quiet insistence, kept us on this job and forced it through to the present state of success. We are also grateful for the whole-hearted cooperation and the patience exhibited by John Ferguson, Jr. and the entire operating force of the Telluride Mill.



# Conductance Electrostatic Separation with Convective Charging

by Foster Fraas and Oliver C. Ralston

VIRTUALLY all commercial use of electrostatic separation has employed separators depending on differences of conductance of the broken, solid mixtures treated by them. The two main types of conductance separators have been: (1) Those in which the separating field is as near a static field between two charged electrodes as it is possible to maintain in the face of the usual leakage that takes place between electrodes with some 5 to 50 kv tension between them and spacing of  $\frac{1}{2}$  to 3 in.; (2) those in which a convective discharge of corona or spray type, but not a spark discharge, takes place between the carrier electrode and a charged electrode facing it, which is of very restricted area, such as a row of needle points or a small diameter wire parallel to the face of the carrier electrode. The present paper deals with this latter type. The principle is usable in separators where the carrier electrode, usually grounded, has the shape of a roller, a conveyor belt, a vibrating table, a sloping chute, or a nearly vertical plate. The innovation studied by us has been the use of the convective field for charging material on the carrier electrode and succeeded by a field between the carrier electrode and an electrode made of a true dielectric which cannot take on a charge by conduction but is charged by being in the radius of the corona discharge. The acquired charge is of the same sign as that of the corona but is not dischargeable by contact with a good conductor. However, high-tension charge tends to leak off because of the ionization of the ambient air. The field between this electrode and the carrier electrode is as near static as it is possible to obtain. There are some differences in behavior of this electrode and that of the gas tube electrode used by Sutton, as will be discussed below. Both electrodes cause increased adhesion of nonconductive particles to the carrier roll. The particles are pinned there because they are convectively charged on their outer surfaces facing the corona electrode. The electrodes also cause increased rate of release of conductive particles tending to be pinned by corona charging but conveying the charge to the grounded carrier roll almost as rapidly as it is received by the particles.

FOSTER FRAAS, Member AIME, is Metallurgist, U. S. Bureau of Mines, College Park, Maryland, and OLIVER C. RALSTON, Member AIME, is Chief Metallurgist, U. S. Bureau of Mines, Washington, D. C.

AIME St. Louis Meeting, February 1951.

TP 2946 B. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Jan. 24, 1950. Revision received June 21, 1950.

Papers by authors of the U. S. Bureau of Mines staff are not subject to copyright.

The sign of the potential on the corona-emitting electrode makes little difference in the separation. An electrode usually emits negative corona somewhat more easily than it emits positive corona, as electrons are then the fundamental particles emitted. With a positive electrode the flow of electrons involved in positive corona discharge is onto the electrode from the ambient gas.

## Theory

In the separator illustrated in fig. 1, the nonionizing electrode is a rotating, nonconductive dielectric cylinder designated as B. The ions emitted by the corona electrode, or wire F, charge both the particles on the carrier electrode, or roll A, and the surface of the dielectric electrode B. Whereas the space between roll A and wire F contains ions, the space between electrode B and roll A is a pure static field free of ions.

The dielectric electrode has been described by Bullock.<sup>1</sup> However, the corona electrode, which Bullock uses for the purpose of charging only the dielectric electrode, is adapted here to also charge the particles on the carrier roll.

In operation the ionic field from the corona electrode is balanced with the static field from the dielectric electrode. The ionic field charges the particle so that it adheres to the roll. The static field intensifies the discharge of the corona charge by interfacial conduction and permits the release of the particle. The ionic and static fields therefore have opposing effects.<sup>2</sup> The values of both fields are adjusted so that the poor conductor will adhere and the good conductor will be released.

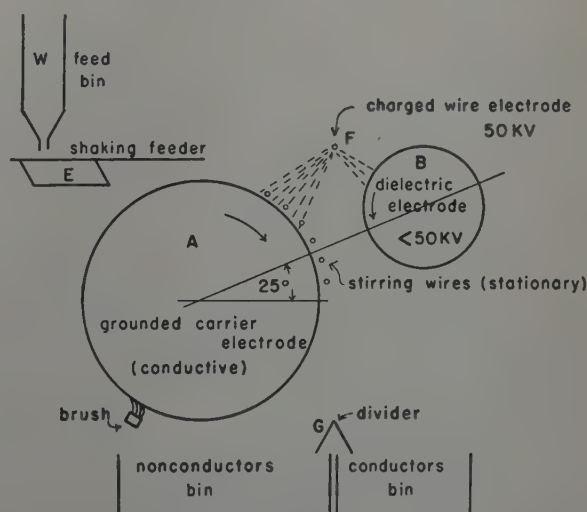


Fig. 1—Diagram of separator.

The intensity of both fields may be increased with this balanced adjustment, but the upper limit is frequently determined by the static field. With the dielectric electrode there is no spark discharge to interrupt the separation and no back deflection of conductive particles which may strike it. The dielectric electrode permits the establishment of a static field intensity that is higher than that possible with metal or conductive electrodes.

By increasing both the static and ionic fields under these balanced conditions, the efficiency of the electrostatic separator is increased. A proof of this follows.

The electrical action occurs in two stages, a preliminary stage in which the particles are charged by the corona, and a second stage where the particles in contact with the roll pass through the static field between the dielectric electrode and the roll (fig. 1). In the second stage the particles lose their corona charge at the rate:

$$\frac{\partial Q}{\partial t} = -(Q_T + Q_B) \frac{SC}{K} e^{\frac{-SCt}{K}},$$

where:

$Q_T$  = a constant charge proportional to  $F$ .

$F$  = field intensity on the carrier electrode in the zone of separation resulting from the dielectric electrode.

$Q_B$  = initial corona charge proportional to  $r^2\sigma$ .

$r, K$  = particle radius and electrostatic capacity.

$\sigma$  = surface density of corona charge.

$S, C$  = interfacial area and conductivity.

$t$  = time

$$Q = -Q_T + (Q_T + Q_B) e^{\frac{-SCt}{K}}, \text{ (Ref. 2).}$$

The signs were determined so that at  $t=0$ ,  $Q=Q_B$ , and at  $t=\infty$ ,  $Q=-Q_T$ . During corona-charge loss,  $Q$  decreases from the value  $Q_B$ , passes through zero, and attains the value  $-Q_T$ .  $Q_T$  and  $Q_B$  have the same polarity. The final value of  $Q$  is of opposite polarity. The rate of loss of charge  $Q$  increases with the increase in  $Q_T$  and  $Q_B$ . Similarly the rate of change of  $Q$  with respect to the change in  $C$  increases with the increase in  $Q_T$  and  $Q_B$ .

$$\frac{\partial Q}{\partial C} = -(Q_T + Q_B) \frac{St}{K} e^{\frac{-SCt}{K}}$$

This relationship proves the statement that the highest efficiency is secured when the ionic and static field strengths are balanced at their highest intensity. In calculations the system of units must be considered.

The motion of a particle on the grounded roll is controlled by the opposing electrostatic and centrifugal forces  $f_e$  and  $f_c$ .

$$f_e = \frac{Q^2}{(2ir)^2} = br^2\sigma^2; \text{ outside of field } F$$

$$f_e = QF + \frac{Q^2}{(2ir)^2} = ar^2\sigma F + br^2\sigma^2; \text{ in field } F.$$

$$f_c = cr^3\rho(\omega^2 R + g \cos \phi)$$

where:

$i$  = charge to image distance correction factor

$\omega, R$  = roll angular velocity and radius

$\rho$  = density of particle

$a, b, c$  = proportionality constants which include shape factors

$\phi$  = angle between verticle and line through particle and shaft center line

$g$  = acceleration of gravity.

The equation for  $f_c$  would be modified with other types of carrier electrodes.

In the field  $F$ , the corona-charged particle has a greater adhesion to the roll than out of the field. When the particle is in the field but off the roll, the attraction toward the roll is less than when the particle is on the roll. This is an explanation for the violent ejection of the conductors from the roll. A great force of adhesion, if present momentarily with the conductive particles, is a factor contributing a favorable interfacial surface conductance.

If for a given particle size the separator is adjusted so that for the nonconductive particle  $f_e = f_c$ , then for all larger particles,  $f_e < f_c$ , and for all smaller particles,  $f_e > f_c$ . The correct procedure is to select an adjustment so that  $f_e \approx f_c$  for the maximum nonconductor particle size. Actually this adjustment is made relative to the position of the dividing edge ( $G$ , fig. 1) and not with respect to release from the roll.

Table I. Beneficiation of Hematite Ore

Fraction	Weight, Pct	Assay by Particle Number, Hematite, Pct	Chemical Assay, Fe, Pct	Total Fe, Pct
Concentrate	82.7	97	67.7	90
Rejects	17.3	19	35.9	10
Composite	100	78	62	100

Table II. Size Analysis of the First Roll Hematite Concentrate

Fraction Mesh Size	—20 + 35	—35 + 65	—65 + 100	—100	Total
Weight, Pct	42.7	35.7	11.2	10.4	100

In the separation of a mixture of a conductor and a nonconductor where the conductivities are highly divergent, all the particles of the conductor, both coarse and fine, will lose their corona charge and be deflected before the nonconductor particles are released. This is illustrated with crushed quartz-pyrite.

When the conductivities are not highly divergent, the corona charge loss rates from the two constituents become competitive. Such a condition arises even though there is still a large difference in the specific conductivities of the compact crystals of the two constituents. This results from the probability factor in the conductance at the particle-roll interface. Measurement has shown that the electrical charges acquired by galena particles on a copper carrier roll with a static field are distributed as a logarithmic function of the normal probability law. Galena is a comparatively good conductor. The charges were measured as each particle fell through a Faraday cage, which was connected to an FP 54 thermionic electrometer tube, a direct current amplifier, and a high-speed recorder.



A frequency distribution of charging rates may result from either variations on each particle or variations between particles. The former has been demonstrated with straight-line probability plots. With the latter, some particles may have, by chance, a low conductance surface exposed. Regrinding would yield a new distribution.

In general, all factors that may improve the interfacial surface conductance, such as good contact and the cleaning of the carrier roll, have an influence on the efficiency of the separation.

When the conductivities are not highly divergent and the adjustments are made so that,  $f_e \leq f_o$ , a high purity cut can be taken of the good-conductor fraction but not of the poor-conductor fraction. Fine good-conductor particles will always be found with the coarse and fine poor-conductor particles. Size classification is required for removing a high purity cut of the poor conductor.

### Application

Although the performance of the separator may be illustrated with a variety of ores, the results with those containing locked particles is most significant.

The repulsion effect of the high intensity static field on the corona charged nonconductors is sufficient to cause the firm adhesion of locked particles of conductor and nonconductor to the carrier roll. The corona-charged nonconductor portion cannot be discharged, although the conductor portion may be in contact with the roll. Such a response has previously been suggested by J. L. Gilson.

The poor-conductor fraction will contain therefore all the locked particles. Since the separator capacity is highest at the coarsest size, it is good practice to separate initially as much of the coarse good-conductor as possible. A most efficient separation is secured by the following procedure:

1. Remove unlocked good-conductor particles at the coarsest size.
2. Grind the undeflected residual mixture of locked particles, nonconductor particles and low-probability good-conductor particles to finer sizes, and repeat the separation.

Table I and fig. 2 summarize the results with a quartz-hematite (specular) ore from Lyon Mountain, N. Y. The carrier roll and dielectric electrode had diameters of 6 and 3 in., respectively, and rotation speeds of 150 and 300 rpm. A simple form of the dielectric electrode consists of a methyl methacrylate plastic tube (0.13-in. wall thickness) placed as a close fit over a precision metal tube. Corona wires having diameters from 0.003 to 0.01 in. and potentials up to 100 kv are satisfactory.

The feed rate was 0.15 ton per hr per ft length of roll. Higher rotation speeds and feed rates up to 0.25 ton per hr per ft length of roll were used with some sacrifice in grade and recovery. The feed and the separator were heated to counteract the effects of high humidity.

Laboratory batch tests were conducted on a single roll separator with the subsequent products carried through the flowsheet steps of fig. 2. Here a single roll will remove a large portion of the unlocked particles of hematite from the —20-mesh grind. Another single roll removes the unlocked particles from the —35-mesh and another two rolls most of the particles from the —65-mesh grind.

The corona electrode at the first unit was close to the carrier roll. On the succeeding units, it was

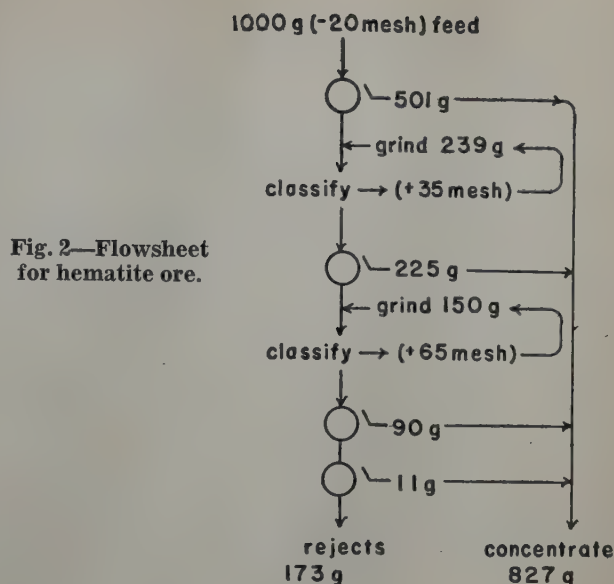


Fig. 2—Flowsheet for hematite ore.

withdrawn from the roll as the particle size decreased. The dividing edge setting at the last three units was close to the roll, and at the first unit, far from the roll. Although a small proportion of low-probability conductors depends on particle size, the major portion of the feed is removed with coarse and fine sizes combined. This is shown in table II by the size analysis of the product from the first roll.

An important feature of this separator is that the product from each roll is a finished concentrate requiring no recleaning.

Ilmenite, high-iron chromite, and magnetite have conductivities which correspond to specular hematite and yield equivalent results. Low-iron chromites have lower conductivities, and when associated with diopside and serpentine having apparent conductivities higher than that of quartz, size classification of the fine fraction is required. Commercial operation would require air or an electrostatic method for classification at sizes finer than 20-mesh. Electrostatic sizing will be discussed in a later paper.

The 0.018-in. diam steel stirring wires shown in fig. 1 were not used in the test described in this paper. They increase the separation efficiency with rutile-zircon mixtures and are more effective with fine than with coarse material. The principle involved is repeated and higher pressure contact of the particles with the roll. Vibration of the carrier electrode or periodic interruption of potential applied should have a similar effect.

A separator is described in which a dielectric electrode and the particles on a carrier roll are simultaneously charged by corona. The characteristics and the procedure for operating the separator on step crushed ores are detailed.

### Acknowledgment

Acknowledgment is given to P. M. Ambrose, chief of the College Park Station, and to F. D. Lamb, in charge of the mineral-dressing unit, for administrative guidance, and to H. W. Jaffe and A. H. Macmillan for petrographic and chemical analysis studies.

### References

- <sup>1</sup>H. L. Bullock: Scope and Economics of Electrostatic Separation. *Ind. Eng. Chem.* (1941) **33**, 1119-1123.
- <sup>2</sup>F. Fraas: The Conductance Electrostatic Separator. *Trans. AIME* (1943) **153**, 576-587.

# Pilot-Plant Investigation of Concentration of Blackbird Cobalt Ore by Roast-Flotation Process

by S. R. Zimmerley and S. F. Ravitz

**High-grade cobalt concentrates were produced from the complex Blackbird ore with very good recovery in continuous pilot-plant operations in which a low-grade bulk cobaltite-pyrite flotation concentrate was roasted in a multiple-hearth furnace to oxidize the pyrite selectively. The cobaltite was floated from the resulting calcine.**

THE existence of the cobalt-copper deposits in the Blackbird district, Lemhi County, Idaho, has been known for many years. However, these deposits have not yet been exploited successfully because of the low grade of the ore, absence of information on ore reserves, and lack of feasible methods for concentrating and recovering cobalt from the ore.

In 1941, when it appeared that World War II might interfere seriously with imports of cobalt from overseas, the U. S. Bureau of Mines started an extensive mining and metallurgical investigation of the Blackbird deposits. Exploration by trenching and drilling<sup>1</sup> indicated that ore reserves were adequate to warrant consideration of large-scale exploitation. A process for producing high-grade cobalt concentrates from the ore was developed at the Salt Lake City Station of the Bureau,<sup>2</sup> and hydrometallurgical procedures for producing cobalt metal or oxide from the concentrates were developed at the Boulder City Station.<sup>3</sup> The Howe Sound Co., through a subsidiary, the Calera Mining Co., carried out further exploratory work, which increased probable ore reserves to a tonnage commensurate with a large investment in mine, mill, transportation facilities, and refinery. Simultaneously, the metallurgical staff of the company, working in cooperation with the Bureau of Mines, further investigated the metallurgical problems. It was the opinion of both groups that pilot-plant testing was required to translate laboratory results into plant operation.

In general, the Blackbird ore has a high content of pyrite and some pyrrhotite and contains 0.5 to 1 pct cobalt and 1 to 2 pct copper, which are present essentially as cobaltite and chalcopyrite. The copper can be recovered readily as a high-grade concentrate by selective flotation. The cobalt then can be recovered as a bulk sulphide concentrate assaying 4 to 5 pct cobalt, but no satisfactory procedure for separating the cobalt from the pyrite

and pyrrhotite directly by flotation had been found.

The method developed by the Bureau for producing a high-grade cobalt concentrate is essentially an application of the old Horwood process of selective roasting.<sup>4</sup> The bulk sulphide concentrate is roasted to form an oxide coating on the iron mineral particles, and the calcine is then treated by flotation. The oxide coating prevents the iron minerals from floating so that it is possible to obtain a high recovery of cobalt in concentrates assaying more than 20 pct cobalt.

Laboratory tests by metallurgists of both the Bureau and the Howe Sound Co. had indicated the optimum roasting temperature to be 425° to 450°C. At lower temperatures the rate of oxidation of the iron minerals is too low, whereas at higher temperatures substantial quantities of cobalt are rendered water-soluble. Because of its high sulphur content (35 to 40 pct), the bulk concentrate tends to continue to burn after it has reached the optimum roasting temperature. Although excessive temperatures could be avoided in the small-scale laboratory furnaces, there was considerable doubt that the temperature could be controlled properly in industrial equipment.

To provide material for the pilot-plant tests, the Calera Mining Co. installed a small, temporary flotation mill at the Blackbird property and treated

---

S. R. ZIMMERLEY, Member AIME, is Chief, Metallurgical Division, Region IV, U. S. Bureau of Mines, Salt Lake City, Utah. S. F. RAVITZ, Member AIME, formerly Assistant to the Chief, Metallurgical Division, Region IV, U. S. Bureau of Mines, is now associated with the Division of Mineral Technology, University of California, Berkeley, Calif.

AIME New York Meeting, February 1950.

TP 2929 B. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Nov. 28, 1949.

Papers by authors of the U. S. Bureau of Mines staff are not subject to copyright.

---



Table I. Results of Continuous Campaign

Period	Feed Rate, Lb per Hrs <sup>a</sup>	Time, Hr	Calcine Assay, Pct			Sulphur Eliminated, Pct <sup>b</sup>	Co Assay, Pct			Co Distribution, Pct <sup>c</sup>		
			Co	Fe	S		Conc.	Mid.	Tail.	Conc.	Mid.	Tail.
Roasting Bulk Concentrate: 4.5 pct Co, 36.6 pct Fe, 38.9 pct S												
1	118	24	5.1	40.7	23.9	45	23.0	5.3	0.65	80.9	10.9	8.2
2	170	28	5.1	40.1	27.9	35	21.7	2.9	0.40	87.3	7.0	5.7
3	187	52	5.0	40.3	30.3	29	17.7	3.2	0.40	87.0	8.1	4.9
4	195	92	5.0	39.8	29.4	30	17.8	3.2	0.38	86.7	8.5	4.8
5	169	64	5.0	41.5	26.7	39	20.7	4.9	0.50	82.1	11.3	6.6
6	172	76	5.0	41.9	25.7	42	21.7	4.3	0.45	84.8	9.8	5.4
7 <sup>d</sup>	172	40	5.4	37.0	20.4	48	21.5	4.3	0.62	82.0	10.3	7.7
Roasting Underroasted Calcine: 5.5 pct Co, 40.0 pct Fe, 34.5 pct S												
8	167	82	5.2	43.0	22.5	51	22.9	4.5	0.43	87.6	6.3	6.1
Roasting Flotation Middlings: 4.2 pct Co, 40.9 pct Fe, 31.5 pct S												
9	169	61	4.6	44.0	23.2	50	21.9	4.9	0.58	81.3	8.8	9.9

<sup>a</sup> Including recycled flue dust, which constituted 6 to 12 pct of feed in all periods except No. 7.

<sup>b</sup> Calculated from iron and sulphur assays of bulk concentrate and calcines.

<sup>c</sup> Exclusive of dissolved cobalt in mill solutions (0.6 to 2 pct of total cobalt).

<sup>d</sup> Feed consisted of approximately 38 pct bulk concentrate and 62 pct flue dust.

<sup>a</sup> Including recycled flue dust, which constituted 6 to 12 pct of feed in all periods except No. 7.

<sup>b</sup> Calculated from iron and sulphur assays of bulk concentrate and calcines.

<sup>c</sup> Exclusive of dissolved cobalt in mill solutions (0.6 to 2 pct of total cobalt).

<sup>d</sup> Feed consisted of approximately 38 pct bulk concentrate and 62 pct flue dust.

350 tons of ore, producing about 50 tons of bulk cobalt concentrates assaying approximately 4.5 pct Co, 37 pct Fe, 39 pct S, and 0.45 pct Cu, which were shipped to the Salt Lake City Station of the Bureau.

#### Pilot-Plant Equipment and Procedure

**Roasting:** The bulk concentrate as received was too wet and lumpy to feed uniformly to the roasting furnace. Therefore, it was air-dried and screened through 10-mesh. The +10-mesh lumps were allowed to air-dry for a day or two and then were crushed and returned to the screen.

The bulk concentrate was roasted in an experimental Herreshoff furnace, which was 6½ ft high, 3 ft in diameter inside the refractory lining, and equipped with eight roasting hearths. The effective hearth area was 49 sq ft, and the time required for the charge to pass over each hearth was 10 to 15 min. The actual temperature of the material on each hearth was determined by means of a bare-junction thermocouple inserted by hand directly into the charge. The furnace gases passed first through a cyclone dust collector, then through a venturimeter for measuring the volume, and finally through a fan to the stack. The flue dust was mixed with incoming bulk concentrate and returned to the furnace.

**Flotation:** The calcine was fed at the rate of approximately 300 lb per hr to a conditioning tank, where it was mixed with water to give a pulp density of about 30 pct. An aqueous emulsion containing 0.4 to 0.5 lb Minerec A and 0.05 lb B-23 (stabilized with 0.03 lb Syntex M) per ton of calcine was added at the conditioner. The pulp then flowed through the mechanically agitated flotation cells. The first two cells produced a rougher concentrate, which was cleaned in a third cell, the cleaner tailing being returned to the first rougher cell. The rougher tailing was retreated in a scavenger cell to produce a middling product and a tailing; approximately 0.1 lb of additional collector and 0.05 lb B-23 per ton of calcine were used in the scavenger circuit. The middling product, which had approximately the same cobalt assay as the original bulk concentrate, was air-dried and stored for reroasting.

Flotation was carried out at the natural pH of the pulp which, with properly roasted calcine, was 3.5 to 4.0. The mill solution also contained 0.15 to 0.4 g of cobalt, up to 0.2 g of copper, and about 4 g of iron per liter. Very sensitive chemical tests were made for arsine and hydrogen sulphide in the exit air from the cells, but none was detected.

#### Experimental Results

To obtain operating information needed for a continuous campaign, the furnace was operated intermittently for the first two weeks. It was found that burning of the charge could not be confined to the upper hearths by ordinary furnace manipulation. Burning gradually advanced downward so that the calcine was discharged at 450°C or higher and continued to burn in the storage piles.

Laboratory tests on various portions of calcine showed that if the calcine is either quenched from roasting temperature or allowed to cool in the absence of air, virtually no selectivity is obtained in flotation. Above 425°C the rate of oxidation of the first atom of sulphur in the pyrite is apparently much greater than that of the second so that the surfaces of the particles consist essentially of FeS (or artificial pyrrhotite) and unaltered cobaltite, both of which float readily. However, if the burning calcine is cooled slowly to below about 320°C, burning ceases and the FeS is oxidized to iron oxide, as evidenced by a change in color from gray to red. The pyrite particles thus are made nonfloatable, and very good selectivity is obtained in flotation.

In view of these results, it was decided to control the cooling entirely within the furnace by means of water sprays in order to maintain a maximum charge temperature of about 450°C and a discharge temperature below 300°C. The furnace was rabbled out, and feeding was resumed at a relatively low rate. By supplying a small amount of heat at hearth 3 and injecting a fine spray of water at intervals as needed on hearths 4 to 7, the following average temperature distribution in the material on the various hearths was readily maintained: hearth 2, 408°C; 3, 459°; 4, 456°; 5, 419°; 6, 414°; 7, 340°; and 8, 265°. Introduction of air into the furnace was controlled by means of the doors

on the two bottom hearths; all the other doors were kept closed.

This method of operation immediately gave very good flotation results. The furnace was operated continuously in this manner for the duration of the campaign, about 1 month. The feed rate was varied at intervals to determine the optimum furnace capacity. Toward the end of the campaign, under-roasted calcine produced in the preliminary operations and the accumulated middling product from flotation were roasted. The flotation mill, which had a somewhat higher capacity than the furnace, was operated an average of about 12 hr per day. The data obtained are summarized in table I and are discussed below. The nine periods listed were consecutive and cover the entire continuous campaign.

### Treatment of Bulk Concentrate

The grade of concentrate was 23 pct cobalt at the minimum feed rate of 118 lb per hr (period 1), dropped to 21.7 pct when the feed rate was increased to 170 lb per hr (periods 2, 5, and 6), and dropped further to slightly less than 18 pct at a feed rate of about 190 lb per hr (periods 3 and 4). Sulphur elimination from the bulk concentrate averaged 45, 39, and 30 pct, respectively, for the three feed rates.

In general, 81 to 88 pct of the cobalt was recovered in the concentrate at a grade of 18 to 23 pct; 6 to 11 pct was recovered as a middling product assaying 3 to 5 pct; and 5 to 8 pct was lost in the tailing, which assayed 0.4 to 0.65 pct. In addition to the tailings loss, 0.6 to 2 pct of the cobalt was lost as dissolved cobalt, as determined by occasional analyses of the mill solutions.

At the beginning of the continuous campaign, the only spray nozzle available was one that produced a conical spray so that some of the water impinged directly on the charge. At the start of period 6, this nozzle was replaced by one that formed a fan-shaped mist so that much less water came in contact with the charge before vaporizing. An increase of 1 pct in grade in period 6 as compared to period 5 may have been caused by this change. The average quantity of water used throughout the continuous campaign was 220 lb per ton of feed.

### Flue Dust

Although most of the flue dust was recirculated to the furnace with incoming bulk concentrate, a considerable quantity was on hand at the end of period 6, when nearly all the bulk concentrate had been roasted. This dust was mixed with the remaining bulk concentrate, and the mixture, consisting of 62 pct flue dust and 38 pct bulk concentrate, constituted the feed for period 7. The data in table I show that this feed gave nearly the same results as had been obtained previously. However, 22 pct of the weight of the feed during this period was collected as flue dust in the cyclone.

### Treatment of Flotation Middling

By the end of period 8, nearly 5 tons of middling flotation product had been produced. This product had an average analysis of 4.2 pct Co, 40.9 pct Fe, and 31.5 pct S, and thus was similar in analysis to the original bulk concentrate except for its somewhat lower sulphur content. It constituted the feed for the last period of operation of the pilot plant and gave a very good grade of concentrate—21.9 pct. Recovery in the concentrate, 81 pct, was slightly

less than in the treatment of the bulk concentrate; loss in the tailing, 9.9 pct, was slightly higher.

### Conclusions

The following data for period 6 are probably representative of the results that would be obtained in plant practice. Taking into consideration the soluble cobalt loss (0.6 pct), about 84 pct of the cobalt was recovered at a grade of nearly 22 pct, 10 pct was recovered as a middling product, and 6 pct was lost in the tailing and in the mill solutions. On retreatment of the middling, about 9 pct of its total cobalt content, or slightly less than 1 pct of the cobalt in the original bulk concentrate, was lost; the remainder eventually would be recovered at a grade of approximately 22 pct. It is probable, therefore, that an over-all flotation recovery of 93 pct of the cobalt at a grade of about 22 pct could be expected in commercial operation.

The total feed rate to the furnace during period 6 was 3.5 lb per sq ft of hearth area per hour. On this basis, the capacity of a 21-ft, 8-hearth furnace would be approximately 100 tons of feed per 24 hr. Assuming a 20-pct circulating load of flue dust and middlings, the net capacity would be about 80 tons of bulk concentrate per day.

The flotation cell capacity (including cleaning) was 1.9 tons of calcine per cu ft per 24 hr. Allowing for a 15-pct loss in weight during roasting and a 10-pct circulating load of middlings, the net flotation capacity in terms of original bulk concentrate would be about 2 tons per cu ft per 24 hr.

The results of this investigation indicate that the roast-flotation process is technologically feasible for treatment of the Blackbird ore and that the principle of preferential oxidation induced by heat may warrant consideration for the preparation of other complex and refractory ores for beneficiation by flotation.

### Acknowledgments

The authors wish to express their appreciation to H. A. Pearse, Vice-President and Chief Metallurgist of the Howe Sound Co., and E. B. Douglas, Manager of the Calera Mining Co., for their co-operation. Mr. Pearse and Mr. Douglas spent considerable time at the Intermountain Experiment Station during the investigation and contributed in many ways to the success of the project. Thanks are due also to I. W. Nicholson and Russell Stewart, Metallurgists of the U. S. Bureau of Mines and Howe Sound Co., respectively, who jointly supervised the furnace operations, to C. H. Schack, Metallurgist of the U. S. Bureau of Mines, who supervised the flotation operations, and to the many other U. S. Bureau of Mines employees who participated in the investigation.

### References

- <sup>1</sup>G. C. Reed and J. A. Herdlick: Blackbird Cobalt Deposits, Lemhi County, Idaho. U. S. Bur. Mines R I 4012 (1947) 14 pp.
- <sup>2</sup>R. R. Wells, W. G. Sandell, H. D. Snedden, and T. F. Mitchell: Concentration of Copper-Cobalt Ores from the Blackbird District, Lemhi County, Idaho. U. S. Bur. Mines R I 4279 (1948) 21 pp.
- <sup>3</sup>F. K. Shelton, R. E. Churchward, J. C. Stahl, and G. F. Livingston: A Study of Certain Factors in the Hydrometallurgy and Electrodeposition of Cobalt. U. S. Bur. Mines R I 3832 (1945) 43 pp.
- <sup>4</sup>Horwood: U. S. Pat. 1,020,353 (1912). See A. F. Taggart: Handbook of Ore Dressing. First Edition. p. 875. 1927. New York. John Wiley and Sons, Inc.



# Kerosine Flotation of Bituminous Coal Fines

by L. E. Schiffman

This paper describes the operation of two kerosine flotation plants in Alabama for cleaning —10 mesh bituminous coal. One plant treats washer sludge, the other raw coal. Data on capacity efficiency and capital and operating costs is given. Methods for increasing cell capacity are discussed.

IN cleaning coal it has long been recognized that methods which give excellent results for the coarser sizes may give poor or even no cleaning for the finer sizes. Effective cleaning of the fines, therefore, is usually a separate and distinct problem. For many years the problem of fines was considered of relatively minor importance. Now there is growing interest in methods of cleaning the fine sizes of coal because:

1. There are more fines. Increasing mechanization and the adoption of full seam mining methods have resulted in the production of a greater percentage of smaller particles than were produced by hand loading methods.

2. The fines have deteriorated in quality. The same causes that have led to increased quantity have also operated to decrease quality through greater contamination with slate, rock, and other diluents.

3. The fines have become more valuable. When coal was comparatively cheap, loss of the finer sizes was tolerated but, with the rapid increase in the cost of mining coal, it has become imperative that the maximum amount of clean coal be recovered from the mixture of coal and rock delivered to the tipple.

4. In many cases, the fines have become a stream pollution problem. Increasing governmental activity and legislation to prevent stream pollution often have made it necessary to retain the fine material which formerly was permitted to escape with the washer effluent into flowing streams.

The Sloss-Sheffield Steel and Iron Co. operates four coal mines, all within a 20-mile radius of Birmingham, Ala., with a total annual production of 1,250,000 tons of washed coal. Three of these mines are in the Mary Lee seam and one in the Jefferson seam. Although some steam and domestic coals are produced, the major portion of the coal is used to make coke for blast furnace operation. Fine coal is desirable for coking, and all the coal so used by Sloss is crushed to  $\frac{1}{4}$  in. Therefore, there is no problem of an outlet for even the finest sizes of coal. Since the coal is used for metallurgical purposes, low ash is of greater importance than would be the case if it were used for steam generation or domestic heating. Based on present blast furnace practice at Sloss, it is calculated that each percentage of ash in the coal carries a penalty of \$0.22 per ton of coal. Thus there is a considerable margin for operating costs in a fine coal cleaning method that will result in materially lowering the ash content of the cleaned coal.

## Development of Fine Coal Cleaning Problem

In 1942 Sloss reopened its Bessie mine and constructed at this location a new preparation plant to clean  $1\frac{1}{4}$  in. x 0 coal from the Mary Lee seam.

*L. E. SCHIFFMAN is Electrical Engineer, Sloss-Sheffield Steel and Iron Co., Birmingham, Ala. AIME New York Meeting, February 1950.*

*TP 2918 F. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Jan. 13, 1950.*

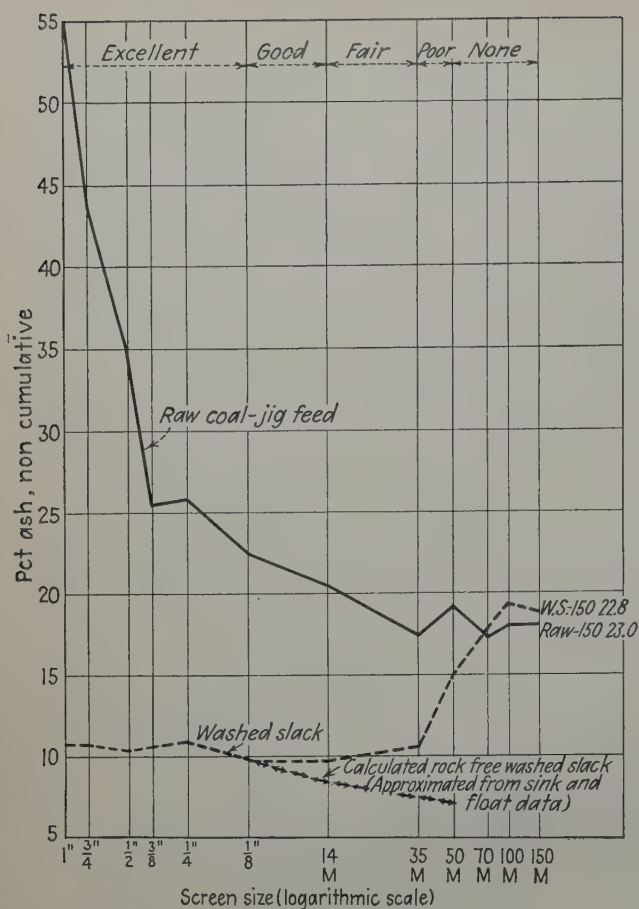


Fig. 1—Separation of coal from rock made by jig.

Ash content of sized coal. Screen analyses of Flat Top coal, average samples Dec. 1 to 8, 1944.

A McNally Norton (Baum type) five-compartment primary jig and a similar two-compartment secondary jig were installed. Contrary to the company's previous practice, the installation included 1-mm screens for dewatering the washed products and a sludge settling tank for recovering the solids in the screen underflow. With this plant in service, the —1-mm fines became separately available prior to mixing with the coarser coal. Samples analyzed for ash then revealed a fine coal cleaning problem because, with the coarser coal having an ash content of about 10 pct, the sludge ash was about 15 to 20 pct.

In 1943 Sloss installed new washing equipment at its Flat Top mine to clean 1 1/8 in. x 0 Mary Lee coal. Jigs duplicating those just installed at Bessie replaced six American (plunger type) jigs. Unlike Bessie, dewatering equipment for the primary coal was not included. Contrary to expectations, when the new equipment went into service, the ash in the primary coal was higher than it had been with the old equipment. Six months of investigation, testing, and adjustment followed. At the end of this period the ash was not appreciably lower, but the cause for the higher ash had been determined. In the old plunger jigs the fine coal, rock, and slate found their way into the hutch and, since the hutch discharge was wasted, did not appear in the finished product. In the new primary Baum type jig, the fines, including that portion of the rock and slate not removed by the jig, remained in the washed coal. Sink-and-float tests together with screen

analyses of the jig products indicated that down to 14-mesh the jig did a good job, down to 35-mesh, a fair job, and below 35-mesh, a poor job. The effectiveness of the jig in turning out a slate-free product is shown in fig. 1. The —35-mesh coal is about 10 pct of the total primary coal. Even with the increase in ash, it is obviously an advantage to retain this amount of coal that previously had been largely wasted, but, with the acquisition of these added coal fines, there also was acquired a fine coal cleaning problem.

### Experimental Cell Installations, Bessie

Thus it became established definitely that below a size of about 1 mm the jigs were not making a good separation and that some better method was needed for cleaning coal below this size. Meanwhile, B. W. Gandrud, of the Bureau of Mines, had discussed with Sloss some of the very interesting results he had secured in laboratory experiments using kerosine flotation for fine coal, and the Sloss-Sheffield Co. became interested in proving the feasibility of the method on a larger-than-laboratory scale.

In 1943, collaborating with Gandrud, Sloss built and installed at Bessie mine a homemade flocculation cell for the flotation of the sludge coal. This cell is shown in fig. 4a. Bessie was chosen as the place for the experimental work because the fine coal was already separately available in the form of sludge from the settling tank. The homemade cell clearly demonstrated that a good separation could be made. The results were sufficiently encouraging to warrant further experimentation. In November 1943, a 2-cell No. 24 (43x43) modified Denver Sub A flotation unit was installed. The modification consisted of a raking device to remove the floated product in place of the conventional paddle for froth removal. The 2-cell unit gave a suitable floated product of about 8 pct ash content, but the refuse leaving the cells was quite low in ash. It became apparent that more than two cells in series would be required if loss of coal in the refuse were to be held to a reasonable amount. Accordingly, an additional 2-cell unit was installed. With the four cells in series, the flotation equipment was put into continuous service, handling about 10 to 15 pct of the sludge, and became a part of the regular washer operation. Under this arrangement, extensive testing was possible. The results

Table I. Bessie Operating Statistics, Average of Daily Results

Seam, Mary Lee; output, 1,000 tons coking coal per day and 125 tons secondary coal per day.

	With Flotation 12 Months Ending Aug. 31, 1949	Prior to Flotation 6 Months Ending Oct. 31, 1947
<b>General Washer Results</b>		
Raw coal 1 1/4 x 0, ash	31.9	30.4
Washed coking coal 1 1/4 x 0, ash	10.1 <sup>a</sup>	11.4
Secondary coal, ash	17.3	13.3 <sup>b</sup>
Float in jig refuse at 1.45 sp gr, pct	3.0	3.2
<b>Flotation Details</b>		
Raw sludge, ash	20.0	
Floated coal, ash	8.2	
Secondary coal, 0' size from reject, ash	24.6	
Refuse from flotation, ash	55.5	
Reagent, 5 lb kerosine per ton of feed		

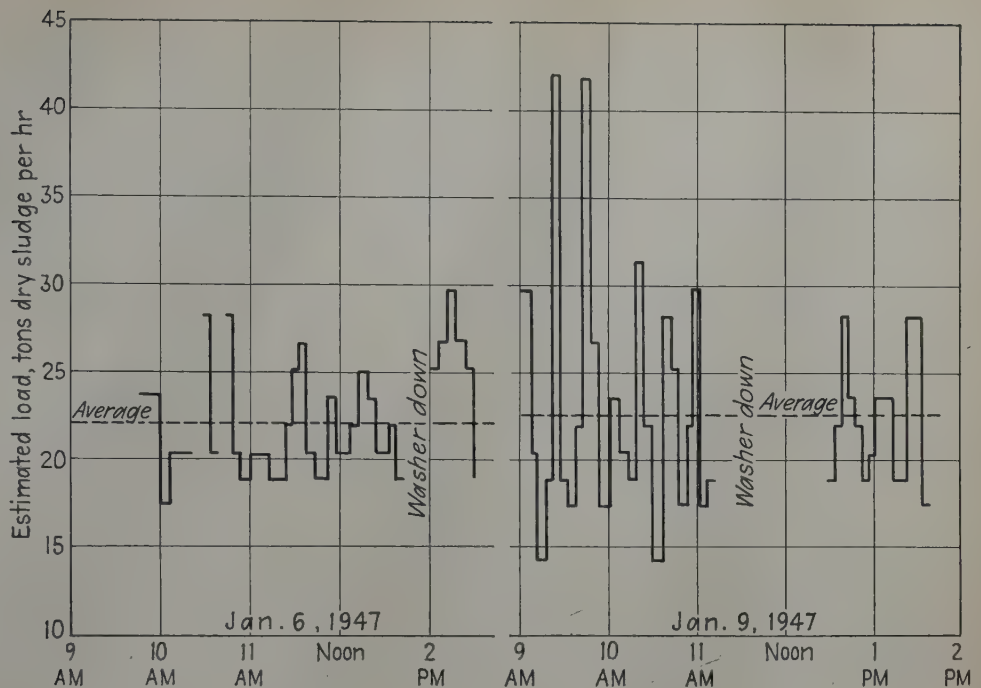
<sup>a</sup> Includes floated coal.

<sup>b</sup> Secondary coal 275 tons per day compared with 125 tons in first column.



**Fig. 2—Estimated loading of sludge recovery drag, Bessie mine.**

Estimate based on observation of volume at 5 min intervals. One cu ft of sludge on drag taken as 54 lb dry sludge. Drag speed was 14 flights per min.



secured from the 4-cell operation were good, with a refuse running 50 pct or higher in ash content.

#### Flotation of Sludge, Bessie Plant

A year of operation on a small portion of the sludge proved the practicability of the process. It then was decided to install equipment of sufficient capacity to process all of the sludge. As a preliminary step, a study was made of the rate at which the sludge was being removed from the settling tank by the drag conveyor used for this purpose.

It was found that the average quantity of sludge was about 22 tph, but the rate of withdrawal fluctuated widely, reaching a peak rate of about 45 tph. The fluctuations in rate are shown in fig. 2. Therefore, it was decided to make the installation of such capacity that it would handle 45 tph with a sacrifice of some coal in the refuse and 22 tph with but little loss of coal. Accordingly, two more 2-cell No. 24 (43x43) Denver "Sub-A" units and two 2-cell No. 30 (56x56) Denver "Sub-A" units were installed, making a total installation of 12 cells. This equipment was put in service in October 1947 and has been in continuous service since that time.

The Bessie washer flowsheet is shown in fig. 3 so that no attempt will be made to describe it here. Suffice it to say that the  $-1$  mm coal from both the primary and secondary circuits is delivered to the sludge settling tank from which it is removed by a slow-moving drag conveyor and then is discharged into a stationary splitter which divides it into two equal parts. The cells are installed in two parallel rows, each of which receives one half the sludge. Each row consists of two 100-cu ft cells (No. 30) followed by four 50-cu ft cells (No. 24) all in series. Floated coal is removed by a single overflow raking mechanism (replacing the usual froth paddle) in each cell and is discharged into two dewatering screws, each of which runs the length of a cell row with the discharge end of each screw projecting over the drag conveyor that carries the jig-plant washed coal to the loading bin. These screws serve the dual purpose of conveyors and dewatering de-

vices. The drainage from the dewatering screws is returned to the first cell in each row by gravity through a side entrance in the cell. The unfloated material, after having traveled through the six cells in series, is discharged into a 6-ft dewatering cone to allow the coarser portion to settle.

The cone underflow is pumped to a shaking screen equipped with  $\frac{1}{2}$ -mm screen cloth, and the  $+\frac{1}{2}$ -mm material is recovered and thrown into the secondary coal. The underflow from the screen joins the overflow from the cone, and these two products, constituting the refuse from the flotation plant, are sent to waste. There is no prior conditioning of pulp before entrance to the cells. The entering pulp averages about 24 pct solids by weight. Reagent flows by gravity from a 6000-gal storage tank through a cup feeder and a rotating distributor into the pulp streams at the splitter (75 pct) and into the inlets of the fourth cells (25 pct). The flotation plant receives an average of about 22 tph of sludge containing about 20 pct ash. In the plant this 22 tons is divided into approximately 14 tons of floated coal of 8 pct ash content, which goes into the primary coking coal, four tons of oversize reject of 24 pct ash content, which goes into the secondary coal, and 4 tons of refuse of 56 pct ash content, which goes to waste. Originally the reagent used was a mixture of kerosine and B23 frother, but at present only kerosine is used. Approximately 5 lb of reagent are used per ton of feed, which is greater than laboratory work by Gandrud has shown to be needed. However, considering the fluctuating character of the feed, the reagent requirements are not excessive. Table I gives a summary of operating results at Bessie mine over a period of 12 months ending August 31, 1949. Mine operation since then has been sporadic and later results are not included. Table II gives typical screen analyses of cell feed and resulting products.

Prior to the installation of the flotation plant, the Bessie washer was operated by two men, a jig operator and a helper. Because of the simplicity of the kerosine flotation process, the cell operation re-



When plans were made for the Bessie installation, it was thought that it would be necessary to eliminate fluctuations in the cell feed to secure satis-

### Flotation of Raw Coal, Kimberly Plant

In view of these possible advantages, it was decided the next installation should be arranged for raw coal feed. One point on which there was doubt was the matter of screening the raw coal to make a 10-mesh separation. As it arrives at the tipple, coal contains too much moisture for dry screening and, if wet screening were done in the usually accepted manner for wet screening, which calls for 5 to 10 gpm per cu yd per hr of screen feed, the resulting underflow pulp would be more dilute than desired. Screen manufacturers willingly furnished information on capacities for either wet or dry

	Pct Retained On,							
	6 Mesh	10 Mesh	14 Mesh	20 Mesh	35 Mesh	65 Mesh	100 Mesh	200 Mesh
Cell feed	6.1	18.2	29.1	45.2	72.8	86.5	91.3	95.5
Floated coal								
No. 1 cells	0	4.1	14.0	31.8	70.3	85.1	90.5	
No. 2 cells	1.1	8.1	20.9	41.8	81.4	91.1	91.7	
No. 3 cells	0.4	8.4	16.0	27.8	76.1	86.7	88.6	
No. 4 cells	1.1	13.8	29.9	50.5	87.6	93.3	94.1	
No. 5 cells	0.7	16.3	30.5	50.0	80.5	91.2	92.2	
No. 6 cells	2.1	20.1	35.6	55.2	77.7	82.5	84.6	
Total, all cells	0.3	6.2	17.2	36.9	71.4	86.2	91.4	95.9
Secondary coal (0' size)	22.6	50.4	67.9	85.0	98.7	99.6		
Refuse	0.5	2.1	4.3	10.8	40.5	63.8	73.5	84.9



screening, but there seemed to be a lack of data available for conditions intermediate between wet and dry. For lack of better information, it was decided to determine screen capacity on the basis that the amount of water required for the desired pulp density would give the coal sufficient mobility to make the screening capacity equivalent to that of dry screening. If it was found that more water was needed, part of it would be recirculated.

Kimberly mine in the Jefferson seam was selected as the logical place for experimenting with a full scale plant on raw coal feed for the following reasons:

1. Kimberly is a comparatively small mine having an output of 450 to 500 tons per day so that for an initial trial of this method, the installation could be made at the least expense.

2. Additional washing capacity may be needed at Kimberly because plans are being made to increase the tonnage. It was hoped the cell plant would increase washer capacity.

3. The jig at Kimberly is a Montgomery jig, a type of jig with a fixed screen under which a plunger with flap valves provides the jiggling strokes and pumps the water required. Tests of the washed product indicated that jig separation became poor at  $-\frac{1}{8}$  in. as compared with  $-35$ -mesh for the Baum type jig.

4. The fines in the Kimberly washed coal were found to be quite high in ash so that if raw coal flotation was successful, an overall reduction of 3 pct could be expected in ash content of the washed coal from this mine. Such reduction would give a greater return per dollar of invested capital than could be obtained at the other mines.

On the basis of experience at Bessie, it was decided a single row of six 100-cu ft No. 30 (56x56) Denver "Sub-A" cells would have a capacity of 20 tph, which was expected eventually to be the required capacity at Kimberly. The plant was laid out and built with this arrangement and went into service in April 1949.

To understand the operation at Kimberly, a brief flowsheet description is necessary. Coal crushed to  $1\frac{1}{2}$  in. at an underground screening and crushing station is brought from the mine by conveyor belt to a 300-ton storage bin. From this bin a feeder and 24-in. belt bring the coal to the washer where it is discharged on a single-deck, 5x14-ft vibrating screen equipped with a No. 5185 ton cap screen cloth having an equivalent opening of approximately 9-mesh. Three rows of sprays cover the deck with sprayed water at a rate of 250 gpm. The oversize from the screen discharges into a 50-tph Montgomery jig. The washed coal from the jig is conveyed by bucket elevator and drag to the loading bin, the refuse by bucket elevator to the rock bin from which it is hauled to waste by truck.

The underflow from the vibrating screen falls into a hopper from which an 8-in. pipe conveys it to the flotation cells. There are six cells in series, the pulp making a single pass through, with un-floated material being rejected at the last cell. Reagent flows by gravity from a 6000-gal storage tank through a cup feeder and distributor to the hopper under the screen (75 pct) and to the inlet of the fourth cell (25 pct). The floated coal is removed by a raking mechanism in each cell and discharged into a 14-in. dewatering screw. Near the discharge end of the screw are two sections of 1-mm wedgewire screen 60 in. long, 11 in. wide and curved

**Table III. Operating Costs of Flotation Plants Per Ton of Feed**

BESSIE	
Operating labor	\$0.00
Reagent, 0.75 gal kerosine at \$0.10 per gal	0.075
Power, 2.9 kw-hr at \$0.01 per kw-hr	0.029
Maintenance and supplies, (est.)	0.03
<b>Total cell plant proper</b>	<b>0.134</b>
Water supply, 250 gpm against 125 ft head	0.004
<b>Total operating cost per ton of feed</b>	<b>\$0.138</b>

KIMBERLY	
Operating labor	\$0.00
Reagent, 0.7 gal kerosine at \$0.10 per gal	0.07
Power, 4.5 kw-hr at \$0.015 per kw-hr	0.068
Maintenance and supplies, (est.)	0.04
<b>Total cell plant proper</b>	<b>0.178</b>
Power for water supply screening and material transport, 2.8 kw-hr at \$0.015 per kw-hr	0.042
Maintenance water supply, screening and material transport, (est.)	0.02
<b>Total operating cost per ton of feed</b>	<b>\$0.240</b>

**Table IV. Kimberly Operating Statistics, Average of Daily Results**

Seam, Jefferson; output 475 tons coking coal per day

	With Flotation Month of August 1949	Prior to Flotation 6 Months Ending March 31, 1949
<b>General Washer Results</b>		
Raw coal $1\frac{1}{2}$ x0, ash	22.4	22.5
Washed coal $\frac{1}{2}$ x0, ash	6.7 <sup>a</sup>	10.4
Jig refuse, ash	70.4	66.0
Float in jig refuse at 1.45 sp gr, pct	3.8	4.8
Hutch solids, ash	51.0 <sup>b</sup>	36.3 <sup>b</sup>
<b>Flotation Details</b>		
Raw coal to cells, 10 mesh x 0, ash	23.5 <sup>b</sup>	
Floated coal, ash	5.1	
Cell reject oversize to jig, ash	31.6 <sup>b</sup>	
Refuse from flotation, ash	58.2	
Reagent 4.5 lb kerosine per ton of feed		

<sup>a</sup> Includes floated coal.

<sup>b</sup> Average of a few snap samples. Routine samples not available.

to 7-in. radius through which water is squeezed by the screw action. The water so removed is returned by a return feed opening into the impeller zone in the first cell. The screw uses a 14-in. pipe as a trough, and this pipe projects 20 in. beyond the end of the screw, forming a nose through which the screw extrudes the coal, thus affording a squeezing action which aids in water removal. It is to the bottom of this pipe that the curved wedgewire screen is attached. The floated coal discharges from the screw into a drag conveyor which carries it up a 35° incline to the coarse coal drag where it mixes with the main body of coal and proceeds to the loading bin. The unfloated material leaving the last cell is discharged into a 5-ft diam x 5-ft high dewatering cone. The underflow from this cone containing the coarser part of the rejected material is pumped by a 2-in. sand pump to a 3x6-ft vibrating screen equipped with 10-mesh cloth. Screen oversize joins the coarser coal going to the jig. The screen undersize joins the overflow from the dewatering cone, the two together constituting the refuse from the flotation plant. It will be noted that this arrangement of dewatering cone and vibrating screen for unfloated material is similar to that at Bessie. This screening arrangement has a number

Table V. Typical Screen Analyses, Kimberly Mine Flotation Plant (Cumulative Pct)

	Pct Retained On							
	6 Mesh	10 Mesh	14 Mesh	20 Mesh	35 Mesh	48 Mesh	100 Mesh	200 Mesh
Cell feed	0.1	5.1	14.6	28.3	52.8	66.9	81.1	91.0
Floated coal								
No. 1 cell	0	1.0	5.4	15.8	44.4	58.9		
No. 2 cell	0	1.6	8.1	21.4	50.8	63.3		
No. 3 cell	0	2.4	9.8	25.8	54.7	66.8		
No. 4 cell	0	2.6	10.7	26.5	55.4	67.7		
No. 5 cell	0	3.1	13.2	29.2	56.0	66.9		
No. 6 cell	0	5.2	18.2	36.0	60.4	69.1		
Total, all cells	0	2.6	11.7	26.0	52.9	64.3	79.9	89.3
Oversize to jig	2.1	52.4	86.4	96.2	99.0			
Refuse	0.1	4.1	13.6	26.1	47.6	55.1		

of functions. First, and primarily, it serves as a guard for the circuit, protecting against loss of coal when oversize material too large for flotation appears in the cell circuit. Second, it prevents the loss of intermediate gravity coal near the upper limit of flotation size. While kerosine flotation will float particles of 10-mesh or larger of low gravity, intermediate gravity material does not float as readily and 10-mesh particles of coal of rather high ash, but too good to throw away, would otherwise be lost in the refuse. Third, the use of the guard screen permits using a mesh size on the main screen that is the top size for flotation, thus rendering the main screening job easier. Incidentally, this screen provides a very good method of checking the flotation circuit at a glance. Normally there is only a dribble of oversize on the screen. When the amount of oversize on the screen becomes appreciable, something is wrong with the flotation circuit or the feed to the circuit.

Under present operating conditions, the flotation plant receives a feed of about 12 tph of —10-mesh coal of about 23 pct ash at a pulp density of between 15 and 20 pct. It delivers about 7¾ tons of floated coal assaying 5 pct ash, returns ½ ton of unfloated material to the jig at about 30 pct ash, and sends 3¾ tons of refuse containing 58 pct ash to waste.

Table IV gives a summary of operating results at Kimberly mine for the month of August 1949, and table V gives typical screen analyses of cell feed and products.

Flotation of raw coal feed has proved quite successful and the advantages that were expected to result from this method have been realized.

1. Retention of fine coal. Wasting the hutch discharge is the method used at Kimberly to bleed water from the circulating system. Prior to the installation of the flotation units, the hutch discharge from the jig assayed 36 pct ash, indicating a loss of coal roughly estimated at 6 tons per day. The hutch material is now about one tenth the former quantity with an ash content of about 51 pct, representing a negligible loss.

2. Greater capacity on the jig. After installation of flotation, the elevator conveying the washed jig product was speeded up approximately 20 pct and a further increase in speed is contemplated.

3. Better jig performance. Prior to flotation the ash in the jig refuse averaged 66 pct and the float at 1.45 sp gr in the jig refuse, 4.8 pct. Now the jig refuse ash is 70 pct and the float in the refuse 3.8 pct.

The average ash of the Kimberly output for 6 months prior to cell installation was 10.4, the average ash for the first 5 months of cell operation was 7.0 pct.

The screening performance has exceeded expectations. Kimberly coal is extremely friable and efficiencies of screening are therefore hard to determine. If calculated on the fines available in the feed and the fines appearing in the undersize, the efficiency reported is too high. If calculated by combining the undersize and oversize and determining the percentage the undersize is of this total, the reported efficiency should be nearer its true value. The latter method is used in obtaining the figures given here. Two tests were made of screen efficiency. Both tests were made at rates in excess of the usual feeding rate of 1.12 tph per sq ft of screening surface because they were made in an effort to determine screen capacity. The usual spraying rate is 2.9 gpm per cu yd of feed per hr.

At a feed rate of 1.52 tons dry coal per hr per sq ft and a spraying rate of 3.0 gpm per cu yd per hr, 82.5 pct of the 10-mesh x 0 appeared in the undersize, 89 pct of the 14-mesh x 0 and 93 pct of the 20-mesh x 0.

At a feed rate of 1.80 tons per sq ft per hr and a spraying rate of 2.6 gpm per cu yd per hr, 77 pct of the 10-mesh x 0 appeared in the undersize, 83 pct of the 14-mesh x 0, and 87 pct of the 20-mesh x 0. Table VI gives screening data on these two tests.

As in the case of Bessie, no accurate means for regular determination of the quantity of feed to the flotation plant is available so that cost data is based on an occasional determination of feed rate and is therefore approximate. Maintenance is approximated on the basis that cell-wearing parts are replaced every two years, plus an amount for other maintenance based on Bessie experience. Cost of operation per ton of feed is given in table III. Prior to cell installation, one man operated the washer. With the addition of the cell units, one man still operates the washer, attending to both the jig and the flotation unit so that no labor cost is shown for flotation.

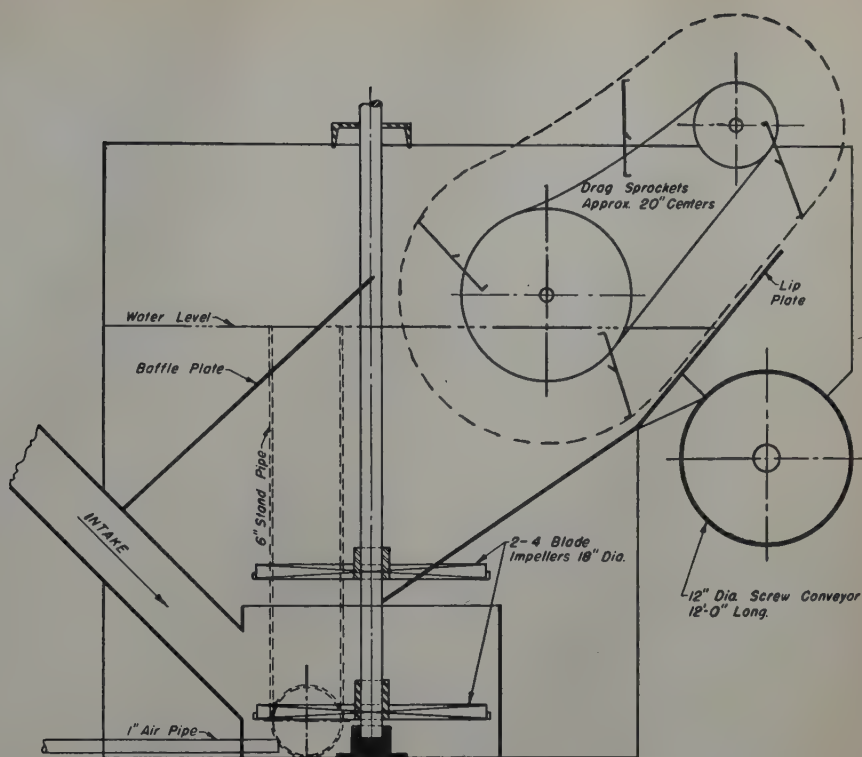
### Cell Capacity

The capacity of a flotation unit depends on many factors among which are pulp density and the permissible loss of values in the rejected material. In the Sloss units, the pulp density decreases from cell to cell as floated coal is removed, and the amount of coal floated per cell also decreases. To avoid appreciable loss of coal in the refuse, experience indicates that the amount of coal floated in the last cell should not exceed 1 tph for a 100 cu ft cell or ½ tph for a 50-cu ft cell.

The distribution of floated coal by cells varies with loading and other factors. Two typical examples of distribution with normal loading as determined by test are given here.



**Fig. 4a—Flocculation cell for the flotation of sludge coal, Bessie mine.**



At Bessie, with the float from the first cell taken as unity, the float from the second cell was found to be 0.80; from the third cell, 0.37; from the fourth cell, 0.33; from the fifth cell, 0.15; and from the sixth cell, 0.11. Likewise at Kimberly, with the float from the first cell as unity, the float from the second was found to be 0.98; from the third, 0.96; from the fourth, 0.78; from the fifth, 0.50; and from the sixth, 0.40.

The Bessie installation was intended to handle an average of 22 tons of dry feed per hour and is doing approximately that at present. Since a secondary coal product is being made from the cell reject, its capacity is probably greater than it would be without the secondary product unless a lower ash refuse were accepted.

The Kimberly installation was designed to handle 20 tons of dry feed per hour and is at present handling about 12 tons with very satisfactory results. Capacity tests of brief duration have been made at feed rates of approximately 20 tph with disappointing results. The floated product was satisfactory, but too much coal reported in the unfloat material, the refuse ash being about 45 pct. The top satisfactory capacity is therefore somewhere between 12

and 20 tph of feed. Until further information is available, a conservative capacity rating is 12 tph.

The approximate cost of the Bessie installation in 1947 was \$32,000, 87 pct of which can be apportioned to flotation equipment and auxiliaries and 13 pct to housing. No added facilities were required for water supply or transport of material to or from the flotation plant. At current costs, a similar plant now would be estimated at \$40,000, or \$1800 per ton-hour of capacity, when rated at 22 tph of feed.

The approximate cost of the Kimberly installation, which was completed in 1949, was \$35,000. This includes an added water supply, screening facilities, and transport of finished product from the plant. It can be apportioned 61 pct for flotation equipment and auxiliaries; 26 pct for water, screening and transport; and 13 pct for housing. At a conservative rating of 12 tph of feed, this is approximately \$2900 per ton-hour of capacity or about \$2200 for the cell plant alone.

#### Investigation of Methods for Increasing Capacity

While both the Bessie and Kimberly plants are of sufficient capacity to handle the available feed, other installations are contemplated, and therefore it has

**Table VI. Screen Analyses of Feed to and Products from 5x14 Robbins Vibrex Screen, Kimberly Mine (Cumulative Pct)**

	¼ In. Round	6 Mesh	10 Mesh	14 Mesh	20 Mesh	35 Mesh	65 Mesh	100 Mesh	200 Mesh
<b>Test No. 1</b>									
Feed	50.0	64.4	77.4	83.2	87.3	93.8	96.5	97.7	99.1
Oversize (to jig)	81.5	90.0	95.0	97.5	98.8	99.7			
Undersize (to cells)	0	0	8.0	20.5	36.5	61.3	77.6	84.8	92.4
<b>Test No. 2</b>									
Feed	52.3	67.1	79.4	84.8	88.8	94.2	96.6	97.8	99.2
Oversize (to jig)	78.3	87.1	93.7	96.2	97.6	99.3			
Undersize (to cells)	0	0	6.0	17.6	32.8	60.0	76.4	83.5	92.5

Screen cloth, ton cap No. 5185

Test No. 1 1.52 tons dry feed per sq ft screen area per hr  
3.0 gpm water sprayed per cu yd feed per hr

Feed 1½ in. x 0 raw coal. Weight, 60 lb per cu ft

Screening area, 62 sq ft

Test No. 2 1.80 tons dry feed per sq ft screen area per hr  
2.6 gpm water sprayed per cu yd feed per hr

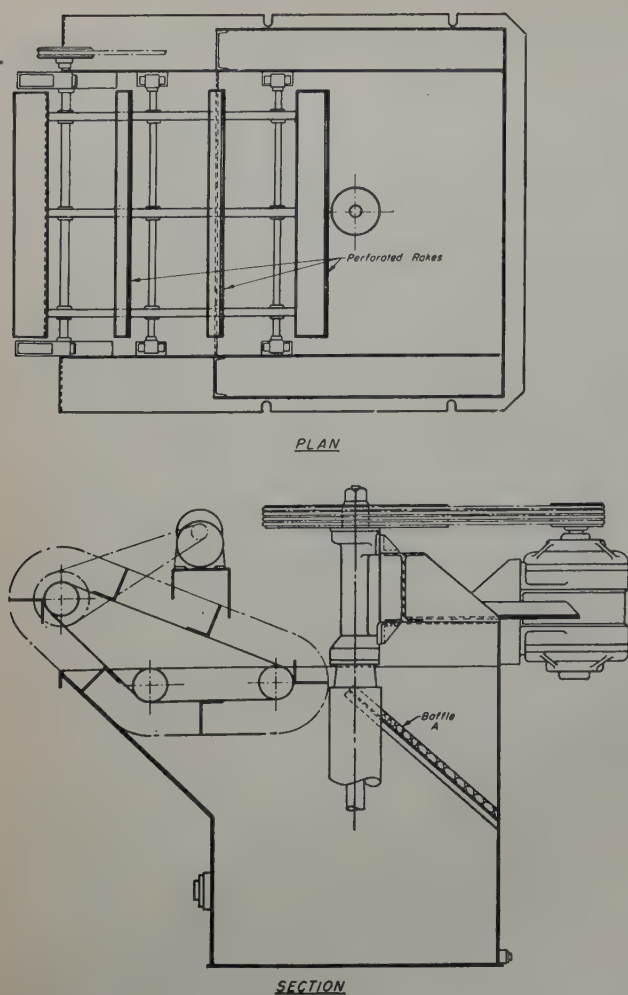


Fig. 4b—Standard raking mechanism.

been desirable to investigate means of increasing cell capacity for the benefit of these future installations. Investigations have been made on the following possible methods of securing such increase: supercharging, changing impeller speed, increasing area covered by raking, and conditioning.

**Super Charging:** When the first 100-cu ft cells were purchased, the Denver Equipment Co. which supplied them recommended supercharging be used to develop the full capacity of the larger cells. Accordingly, the Bessie installation was arranged for supercharging of both the 100 and 50-ft cells. No advantage was found in the use of supercharging and, after a few days of experimenting, supercharging was discontinued. After the Kimberly installation was made, observation of some turbulence breaking through the surface of the matte of floated coal led to the belief that a reduction in air might be advantageous. Experiments were made with the second cell by gradually closing the air intake. As a final result, the intake was completely closed with a resulting 7 pct increase in the amount of coal floated on this cell. No attempts have been made as yet to verify this on other cells, and the data given in this paper was secured with all intakes open.

**Changing Impeller Speed:** Observation of the same turbulence previously mentioned suggested a lesser impeller speed might be advantageous. Accordingly, arrangements were made at Kimberly to

slow the impellers in the first and second cells from a speed of 253 rpm to a speed of 207 rpm. Some quieting of turbulence was noted, the power input to these two cells was halved, but the first cell floated only about 80 pct of its normal amount and the second cell 90 pct of normal. Although the remaining cells, with more floatable coal available, floated somewhat more than their normal amount, the ash in the cell refuse averaged 48 pct compared to a normal 58 pct. Since a decrease in speed of 18 pct for the first two cells results in a decrease in float capacity averaging roughly 15 pct, conversely, an increase in speed can be expected to develop an increase in capacity, but such increase would be accompanied by a rapid rise in power consumption and probably by a rapid increase in wear of moving parts.

**Increasing Area Covered by Raking:** When the Kimberly installation was being planned, Denver Equipment Co. offered as an alternate arrangement a cell equipped with a spitzkasten and raking mechanism on each side instead of the cell with the single spitzkasten and raking mechanism as furnished for Bessie. No data was available, but such an arrangement might offer a considerable capacity increase. Installation of double raking was thoroughly discussed, but space limitations and the complication of handling material from two dewatering screws swung the decision in favor of the Bessie type of cell. After Kimberly went into service, discussion of the advantages of double raking over single raking continued. At length, the idea evolved that even with the single spitzkasten and single dewatering screw, the raking mechanism might be rearranged to reap the advantages of double raking. The second cell at Kimberly was picked to make the experiment. The standard raking mechanism is shown in fig. 4b. The mechanism as rearranged in the second cell is shown in fig. 4c. The change involved the removal of baffle A in fig. 4b. The Experimental arrangement rakes an active cell area 2.3 times as large as the standard mechanism does; or, if the spitzkasten area is included, 1.7 times as large. With standard rakes, the second cell usually discharges 0.98 tons of floated coal for each ton discharged by the first cell. With the experimental mechanism, the second cell discharges an average of 1.05 tons of floated coal for each ton by the first cell, indicating an increase in capacity of about 7 pct.

The experimental arrangement rakes as much active cell area as would the double raking mechanism and is believed to possess substantially the same advantages without the extra space required by the second spitzkasten or the need for a second dewatering screw.

**Conditioning:** The original experimental home-made flotation unit at Bessie was preceded by a rather crude homemade conditioning unit for agitation of the mixture of pulp and reagent. In the experimental work, no evidence was found of advantage resulting from the use of the conditioning unit, so conditioning was omitted from the Bessie installation and later from the Kimberly installation. When a search was begun for means of increasing cell capacity, the idea of conditioning was revived. It should be noted that, when testing slower impeller speed at Kimberly, the float capacity loss in the second cell was about one half that in the first, and it is therefore probable that conditioning ahead of flotation would permit slower impeller speeds with little or no loss in capacity. Slower



speeds would decrease both power and maintenance costs. It was not convenient to add experimental conditioning equipment to either the Bessie or the Kimberly installation so investigation was confined to a few laboratory tests. The tests were made with a 500-g Denver laboratory cell. Without conditioning, 90 pct of the float came off in 6 min. With 5-min conditioning, 90 pct came off in 5 min; and with 10-min conditioning, 90 pct came off in 4.2 min. Since in the Bessie installation the major portion of the coal is floated in the first two cells and since calculations show a retention time of roughly 5 min in the larger cells, making a conditioning period of about 10 min before entrance of the pulp to the third cell, the use of a conditioner has not seemed necessary at Bessie. Nevertheless, in view of the laboratory results, it is probable that conditioning would result in increased capacity. It is also probable that conditioning would decrease reagent consumption although no laboratory tests have been made to determine this.

Although they have not been investigated, two other methods of increasing capacity have been considered, namely, increasing initial pulp density to the cells and increasing pulp density somewhere along the row of cells by removal of water.

Gandrud has shown by laboratory work that the cell capacity increases as pulp density increases until a given density is reached after which capacity falls off rather abruptly. In the laboratory the optimum density found was 28 pct solids. The optimum for commercial cells might be decidedly different. In order to gain capacity by operating at as near optimum density as possible, it undoubtedly would be necessary to have some means of securing a uniform rate of feed, which is lacking at Bessie and Kimberly.

Increasing pulp density somewhere along the row of cells could be done by a number of methods, such as the use of a dewatering cone inserted between cells to eliminate part of the water or a classifier for the same purpose. In an installation of the size discussed in this paper, the methods so far proposed involve complications that make it cheaper to obtain capacity by the purchase of additional cells. For a large installation such methods may have merit.

### Reagent

The reagent used at both Bessie and Kimberly at the present time is kerosine with no additions. All of the original experimental work was done with a mixture of about 93 pct kerosine and 7 pct B23 frother as reagent. The Bessie installation was started using this same mixture. The B23 costs about \$1.05 per gal as compared with about \$0.10 for kerosine so that its use adds materially to the reagent cost unless accompanied by a marked reduction in reagent requirements. In order to determine the value of the B23 addition, the plant was operated for several alternate weeks with and without the addition. Although laboratory work has repeatedly shown the superior performance resulting from the use of the frother, no definite advantage could be proved by the actual plant performance. The use of B23 was discontinued.

After the Kimberly plant was in operation, several brief snap sample tests under the supervision of Gandrud and Riley<sup>2</sup> were made using both pine oil and B23 as an addition. These tests gave improved results when pine oil was added and still greater improvement when B23 was added. Except

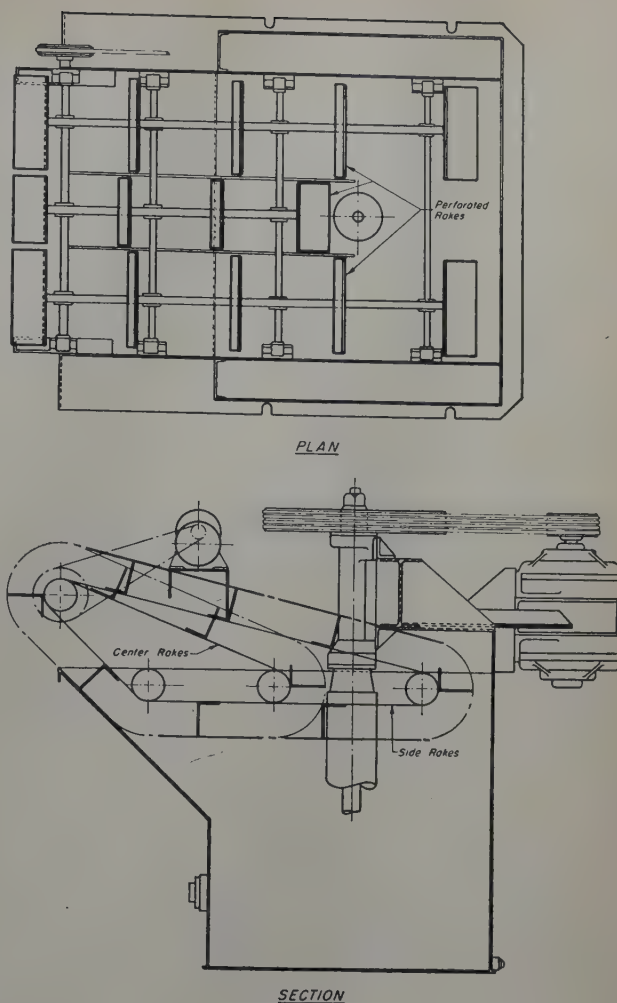


Fig. 4c—Raking mechanism as rearranged in the second cell, Kimberly.

for these brief test interludes, only straight kerosine without additions has been used as reagent at Kimberly. With the present rate of feed, the cell operating results have been considered satisfactory. Should an attempt be made to increase capacity, the use of additions to the kerosine probably would be considered again.

### Dewatering

One of the advantages of the kerosine flotation process is that the floated coal can be readily dewatered to about 25 pct moisture by the use of a very simple device, the dewatering screw. This device is described in detail by Gandrud and Riley.<sup>1</sup> In the Sloss installations, this advantage has not been fully developed because moisture reduction to this extent has not been needed. At Bessie the moisture content of the coal as discharged from the screw varies from 34 to 45 pct with an average of about 37 pct; at Kimberly, from 44 to 52 pct with an average of about 47 pct.

The floated coal drains readily without loss of coal because the process has put it in a flocculated or matte condition, and because most of the fireclay usually in the fines has been removed in the process. For instance, a sample of Kimberly floated coal that was discharged from the screw with 51 pct moisture

drained to 39 pct in  $\frac{1}{2}$  hr, 35 pct in 3 hr, and 32 pct in 22 hr. Thus the floated coal unmixed with coarser material will drain in a day to a moisture content approaching that of an effective dewatering screw. When mixed with coarser coal, as is the case at both Bessie and Kimberly, it drains readily in the loading bin and railroad cars so that upon arrival at the coking plant, 24 to 48 hr later, the moisture content has declined to a point where little would be gained by more effective initial dewatering. For example, Bessie coal, prior to the installation of flotation equipment, averaged 7.8 pct moisture as delivered to the coking plant and, since installation, averages 7.9 pct. Kimberly, prior to installation of flotation, averaged 6.5 pct and, since installation, averages 6.3 pct. Flat Top mine, which has no flotation units, averages 8.3 pct and Lewisburg mine, which also has no units, averages 8.9 pct.

At Bessie the floated coal mixed with coarser coal is conveyed by drag conveyor up a  $34^\circ$  incline, and at Kimberly the floated coal alone by drag conveyor up a  $35^\circ$  incline. As long as the coal is sufficiently dry to be conveyed successfully up these inclines, there is no need for further dewatering. Consequently, the screen area for draining purposes in the dewatering screw has been held to a minimum, and no attempt has been made to develop the full capability of the screw.

### Efficiency

The effectiveness of the process in making a proper separation during regular operation is judged by routine ash analyses of the floated coal and refuse and by observation of the amount of oversize reject going to the jig (Kimberly) or into secondary coal (Bessie). No tests of efficiency are available for Bessie. For Kimberly, an investigation has been made by Gandrud and Riley<sup>2</sup> when using kerosine and pine oil as reagents, and the results are given in a Bureau of Mines Report of Investigation about to be published from which the following excerpts are taken:

On the basis of the "error curve" method for evaluating performance, the plant in which the raw coal fines are treated compares favorably with plants using the more conventional type of processes such as jigging, launder washing and froth flotation.

The head sample for the float-and-sink tests . . . was obtained by mixing —10-mesh floated coal and —10-mesh refuse from the flotation unit in proportionate amounts on the basis of recovery weights. This had to be done in order to eliminate the conditions resulting from size degradation of the feed during treatment in the cells. This degradation on Kimberly coal is so severe that error curves . . . could not be plotted except on the basis of float-and-sink data representing feed after degradation had taken place.

. . . all material coarser than 10-mesh was screened out of the samples and excluded from the data. . . . The feed rate of the flotation unit was 11.5 tons per hour. The reagent feed was at the rate of 3.04 pounds of kerosene and 0.26 pounds of pine oil per ton of feed. Of this, about 65 pct of the kerosene was added to the feed in the intake pipe of No. 1 cell. The remainder of the kerosene and all of the pine oil were added to the weir compartment ahead of No. 4 cell, in other words, in the intake of No. 4 cell. The amount of water in proportion to solids could not be determined accurately but was probably about 5 to 1, resulting in about 17 pct solids in the feed.

The figures . . . show that the composite 10-mesh to 0 products analyzed 21.5, 6.3, and 63.6 pct ash for the

feed, cleaned coal and refuse, respectively, and that 73.5 pct of the feed was recovered as cleaned coal. According to the cumulative curve . . . the float-and-sink yield at 6.3 pct ash was 76.7 pct. Hence the efficiency of the flotation treatment was 73.5 divided by 76.7 or 95.8 pct. This efficiency may seem a little low in comparison with efficiencies normally attained in jig and table washing on the coarser sizes, but, as a matter of fact, the situation would have been considerably different on the basis of the 10-mesh to 0 feed prior to treatment in the cells. Float-and-sink tests made on a sample of 10-mesh to 0 feed taken ahead of the flotation cells showed only 74.3 pct yield at 6.3 pct ash instead of 76.7 pct shown . . . for the composite sample which had undergone size degradation in the cells as explained above. On this basis, the recovery efficiency would have been 73.5 divided by 74.3 or 98.9 pct.

In kerosine flotation, as in coarser coal cleaning methods, the effectiveness of separation becomes poorer below some limiting size. In the case of kerosine flotation, the size at which separation becomes considerably poorer appears to be 200-mesh or below. On a Kimberly floated coal sample of 6.1 pct ash content, the ash in the +48-mesh was 6.0 pct; in the 48x65-mesh, 5.0 pct; in the 65x100-mesh, 5.1 pct; in the 100x150-mesh, 5.7 pct; on the 150x200-mesh, 6.2 pct; and in the —200, 8.6 pct.

The probable cause of the increase in ash in the —200-mesh particles is entrapment of fine refuse and suspension of fireclay in the water carried with the floated coal from the cells.

### Summary

Seeking an answer to the problem of cleaning fine bituminous coal, the Sloss-Sheffield Steel and Iron Co. collaborating with B. W. Gandrud, U. S. Bureau of Mines, has experimented on a successively increasing scale with the kerosine flotation process for coal fines. The experimental work has led to the construction of two commercial flotation plants which are now in operation.

The first plant, with a feed of roughly 22 tph of 20 pct ash —10-mesh sludge from a jigging operation, is producing a floated product of about 8 pct ash content at an operating cost of \$0.14 per ton of feed with a capital investment of approximately \$1800 per ton-hour of capacity.

The second plant, with a feed of roughly 12 tph of 23 pct ash —10-mesh raw coal screened out ahead of the jigging operation, is producing a floated product of about 5 pct ash content at an operating cost of \$0.24 per ton of feed with a capital investment of approximately \$2300 per ton-hour of capacity or \$2900 per ton-hour for the complete installation.

Further investigation and experiment should lead to an increase in the floated coal obtained per cell unit with a resulting decrease in the capital investment required per ton-hour of capacity and also a resulting decrease in the operating cost per ton of feed.

### References

<sup>1</sup>B. W. Gandrud, and H. L. Riley: A Combination Cleaning and Dewatering Process for Treating Fine Sizes of Coal Preliminary Report. U. S. Bur. Mines. R. I. 4306.

<sup>2</sup>B. W. Gandrud, and H. L. Riley: Recent Developments in Connection with a Combination Cleaning and Dewatering Process for Treating Fine Sizes of Coal. U. S. Bur. Mines. R. I. In press.



# Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal

by M. R. Geer, H. F. Yancey, C. L. Allyn, and R. H. Eckhouse

Four coals were treated in the Humphreys spiral concentrator, and the products were examined by float-and-sink and screen-sizing tests to determine fundamental performance characteristics. The efficiency of the separation between coal and impurity was shown to be influenced greatly by particle size.

## Summary

FOUR coals exhibiting different washability characteristics were washed in the Humphreys spiral concentrator and the products examined by float-and-sink and screen-sizing tests to determine the mechanics of the separation between coal and impurity. With one coal, each  $\frac{1}{2}$ -in. increment of the full width of the stream in the spiral was examined separately to determine where each size and specific-gravity fraction of the feed was stratified.

The basic performance characteristics of the spiral were found to be the same for all of these coals. The coarsest fraction of heavy impurity stratified so far out in the stream that it could not be removed through the refuse ports and thus either entered a middling product or contaminated the washed coal in the case of a two-product separation. Impurity particles finer than about 100-mesh also were carried out in the main body of the stream and therefore were not removed in the refuse product. Little loss of clean coal in the refuse occurred in sizes coarser than 28-mesh or finer than 100-mesh, but considerable coal of intermediate size stratified in the stream that was drawn off through the refuse ports.

Because of this modifying influence of particle size, the spiral is unable to make an efficient two-product separation between coal and impurity without some retreatment. Recirculation of a middling product through the same spiral as a means of obtaining more efficient operation was not attempted in this investigation. The size and specific-gravity composition of the middling product are, however, such as to render it amenable to simple hydraulic classification for recovery of coal. Retreatment of a middling product would improve the efficiency of the spiral, but even when making a substantial amount of middling, considerable clean coal enters the refuse product. Consequently, retreatment of a combined refuse-middling product would appear to offer greater promise for providing maximum efficiency.

Because of the sizing characteristics of the spiral, a classified feed can be treated with higher efficiency

than is possible on a natural raw coal. For this reason, semiclassified feeds, such as silt-bank or classifier-underflow materials, can doubtless be treated with higher efficiencies than those shown for raw coals in this report.

Performance is, of course, only one of many factors that enter into the choice of cleaning equipment. The fact that the one company now using the spiral for cleaning coal has built a second spiral plant is ample evidence that the inherently low efficiency of this unit may be overshadowed by the low operating costs afforded by its extreme simplicity.

## Introduction

A number of factors enter into the selection of coal-cleaning equipment, and most of them bear directly on the final cost of prepared coal. One of these factors is performance of the cleaning unit in terms of the efficiency of the separation it effects between clean coal and refuse. The importance of performance in the selection of a coal-cleaning unit varies with the difficulty of the cleaning problem. Obviously, if the character of the coal is such that only certain types of cleaning units are capable of yielding a clean product meeting market requirements, performance is of prime importance. If, on the other hand, the cleaning problem is so simple that any type of cleaner will provide coal of suitable quality, performance is important only insofar as it affects costs by determining the amount of salable coal lost in the washery refuse.

With the objective of providing the coal industry with information that is helpful in the selection of

M. R. GEER and H. F. YANCEY, Members AIME, and C. L. ALLYN are Mining Engineer, Supervising Engineer, and Chemical Engineer, respectively, U. S. Bureau of Mines, Seattle, Washington. R. H. ECKHOUSE, Junior Member AIME, is Research Fellow, University of Washington, Seattle, Washington.

AIME New York Meeting, February 1950.

TP 2944 F. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Feb. 17, 1950.

Papers by authors of the U. S. Bureau of Mines staff are not subject to copyright.



Fig. 1—Full-size spiral installed in laboratory.

coal-cleaning equipment, the Bureau of Mines has published the results of detailed performance studies of the wet table,<sup>1</sup> pneumatic table,<sup>2</sup> Dutch cyclone,<sup>3</sup> and jigs of pneumatic,<sup>4</sup> plunger,<sup>5</sup> pulsator,<sup>6</sup> and Baum<sup>7</sup> types. The present report contains such information for the Humphreys spiral concentrator.

This spiral was developed during the recent war to treat the chrome-bearing sands of Coos Bay, Oregon. In the few years that have elapsed since its development, the spiral has been adopted for treating ores, nonmetallics, and finally coal. One coal-washing plant utilizing the spiral went into operation in the anthracite field in 1946,<sup>8</sup> and the same company recently has completed a second spiral plant. In addition, experiments on a pilot-plant scale were carried out with the spiral at a washery in Colorado.<sup>9</sup>

The spiral is an extremely simple device which involves no moving parts and is constructed almost entirely of unmachined castings. Since it is such an uncomplicated mechanism, operation is simple and virtually foolproof. These characteristics, which go far toward insuring low-cost operation, are attractive attributes in any coal-cleaning unit. Moreover, published information<sup>8</sup> indicates that the spiral can treat coal containing 30 to 40 pct ash to produce a washed product of 14 to 16 pct ash, with rejection of a refuse product containing up to 80 pct ash. Thus, the spiral was promising enough to warrant a detailed performance study.

### Object and Scope

The three published reports describing operation of the spiral contain some performance data; but, in the case of the anthracite plant, this information is not sufficiently complete to provide a clear picture of the mechanics of spiral performance. More detailed information was published for the pilot-plant operation in Colorado; but this unit was used exclusively for treating a table-middling product experimentally, and thus the data are not directly applicable to the treatment of raw coal.

The object of the present investigation was to provide detailed information on the influence of particle size and specific gravity in the separation between clean coal and refuse effected by the spiral. With any gravity concentrating device, except some heavy-medium processes, particle size modifies the influence of specific gravity. The magnitude and character of the influence of particle size determine the fundamental performance characteristics of any cleaning device.

A laboratory installation is ideally suited for determining basic performance characteristics because operating variables can be controlled. On the other hand, a laboratory installation provides little or no information on the other factors bearing on cost that enter into the selection of cleaning equipment. Consequently, the present report is limited in scope to performance data only and necessarily neglects the other factors that can be established best in plant operation.

Nevertheless, this investigation was intensive—over 60 test runs were made with the spiral, and the products obtained from most of them were examined by float-and-sink tests and screen analyses, rather than just ash analyses. Therefore, the conclusions reached regarding fundamental spiral performance characteristics are based on abundant detailed data for four different coals and hence are generally applicable.

### Description of Coals Tested

Coals from Alabama, Kentucky, West Virginia, and Washington were used in this investigation. Table I gives the source of these coals, and tables II, III, IV, and V give float-and-sink and screen-sizing data. All samples represent natural 8-mesh fines screened from run-of-mine coal, except the West Virginia, which was prepared by crushing 3-in. slack to pass 8-mesh.

Table I. Identification of Coals Tested

State	County	Bed	Mine
West Virginia	Wyoming	Pocahontas No. 6	Black Eagle
Alabama	Jefferson	Clements	Prospect
Kentucky	Union	No. 9	Poplar Ridge
Washington	Kittitas	Roslyn	Roslyn No. 3

The Black Eagle coal from West Virginia contains only 5.3 pct of impurity heavier than 1.60 sp gr. With this coal, the coarsest and finest sizes are about equally dirty, and the least impurity is found in the material of intermediate size.

The Clements coal from Alabama contains slightly more heavy impurity than that found in Black Eagle but has very little material of intermediate specific gravity, the presence of which is generally regarded as an index of how difficult a coal is to wash. The impurity in this coal tends to be concentrated in the finer sizes.

The Poplar Ridge coal from western Kentucky is much dirtier than either the Black Eagle or Clements coals, containing nearly 17 pct of impurity heavier than 1.60 sp gr. This coal contains the highest percentage of material of intermediate density of any of those tested. Also, the heavy impurity contaminating it is predominantly fine material; the coal finer than 200-mesh contains over 40 pct of heavy impurity.

The Roslyn coal from the State of Washington contains slightly more heavy impurity than that found in Poplar Ridge coal but somewhat less material of intermediate density. With Roslyn coal there



**Table II. Specific Gravity Analyses, by Size Fractions, of Black Eagle, W. Va., Coal**

Size, Mesh	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
8 to 14	Under 1.30	26.6	1.9	26.6	1.9
Weight, 26.9 pct	1.30 to 1.40	56.7	5.5	83.3	4.4
	1.40 to 1.60	10.6	17.7	93.9	5.9
	1.60 to 1.80	1.6	38.2	95.5	6.4
	Over 1.80	4.5	81.9	100.0	9.8
14 to 28	Under 1.30	38.0	1.5	38.0	1.5
Weight, 28.1 pct	1.30 to 1.40	48.5	5.3	86.5	3.6
	1.40 to 1.60	9.0	17.2	95.5	4.9
	1.60 to 1.80	1.2	36.3	96.7	5.3
	Over 1.80	3.3	80.0	100.0	7.8
28 to 48	Under 1.30	41.2	1.4	41.2	1.4
Weight, 19.6 pct	1.30 to 1.40	46.1	4.9	87.3	3.2
	1.40 to 1.60	8.6	16.7	95.9	4.5
	1.60 to 1.80	1.1	34.8	97.0	4.8
	Over 1.80	3.0	78.4	100.0	7.0
48 to 100	Under 1.30	35.3	1.3	35.3	1.3
Weight, 13.0 pct	1.30 to 1.40	50.3	4.6	85.6	3.2
	1.40 to 1.60	9.4	15.9	95.0	4.5
	1.60 to 1.80	1.3	33.5	96.3	4.9
	Over 1.80	3.7	76.5	100.0	7.5
100 to 200	Under 1.30	23.7	1.4	23.7	1.4
Weight, 5.8 pct	1.30 to 1.40	57.2	4.4	80.9	3.5
	1.40 to 1.60	12.4	14.7	93.3	5.0
	1.60 to 1.80	1.5	31.8	94.8	5.4
	Over 1.80	5.2	73.7	100.0	9.0
Under 200	Under 1.30	10.1	4.2	10.1	4.2
Weight, 6.6 pct	1.30 to 1.40	58.5	5.0	68.6	4.9
	1.40 to 1.60	23.6	12.0	92.2	6.7
	1.60 to 1.80	1.9	26.1	94.1	7.1
	Over 1.80	5.9	70.1	100.0	10.8
Composite	Under 1.30	32.6	1.6	32.6	1.6
Weight, 100.0 pct	1.30 to 1.40	51.5	5.1	84.1	3.7
	1.40 to 1.60	10.6	16.2	94.7	5.1
	1.60 to 1.80	1.4	35.0	96.1	5.6
	Over 1.80	3.9	78.4	100.0	8.4

<sup>a</sup> Moisture-free basis.

is a moderate increase in the percentage of impurity with decrease in particle size, the material finer than 200-mesh containing about twice as much impurity as the material coarser than 14-mesh.

None of these coals contains enough material of intermediate specific gravity to be considered particularly difficult to wash.

### Description of Spiral Installation

The laboratory installation of the spiral is shown in fig. 1. The unit itself is merely a spiral channel of modified semicircular cross-section. The type used for treating coal consists of six full turns in a vertical height of 5 ft with an outside diameter of 2 ft. A circular refuse port at the lowest point in the cross-section of the channel is provided at each 120° interval of spiral turn. A small launder paralleling the main channel carries wash water, which is introduced as a cross-flow at the inner edge of the stream below each refuse port.

In operation, coal and water at a solids concentration of 15 to 20 pct is introduced at the top of the spiral. As the coal flows downward, it is stratified in the stream according to particle size and specific gravity. The refuse product moves along the inner edge of the channel from which it is removed through the refuse ports. The proportion of refuse removed is controlled by the number of ports used and by adjustable splitters inserted in the ports. The remainder of the stream is discharged at the lower end of the spiral.

Where the spiral is used for treating ores, it is customary to draw a concentrate from the upper ports and a middling product from the lower ones. Optionally, a middling product can be cut from the inner portion of the stream discharge. Depending upon the character of the ore and the type of separation

**Table III. Specific Gravity Analyses, by Size Fractions, of Clements, Ala., Coal**

Size, Mesh	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
8 to 14	Under 1.30	87.5	1.4	87.5	1.4
Weight, 44.0 pct	1.30 to 1.40	6.4	6.6	93.9	1.8
	1.40 to 1.60	2.0	18.3	95.9	2.1
	1.60 to 1.80	0.8	42.2	96.7	2.4
	Over 1.80	3.3	74.6	100.0	4.8
14 to 28	Under 1.30	84.9	1.3	84.9	1.3
Weight, 28.3 pct	1.30 to 1.40	7.4	6.5	92.3	1.7
	1.40 to 1.60	2.2	17.5	94.5	2.1
	1.60 to 1.80	0.9	37.0	95.4	2.4
	Over 1.80	4.6	74.5	100.0	5.7
28 to 48	Under 1.30	77.5	1.3	77.5	1.3
Weight, 14.5 pct	1.30 to 1.40	10.7	5.8	88.2	1.8
	1.40 to 1.60	3.1	16.9	91.3	2.4
	1.60 to 1.80	1.3	34.2	92.6	2.8
	Over 1.80	7.4	76.9	100.0	8.3
48 to 100	Under 1.30	68.3	1.6	68.3	1.6
Weight, 7.8 pct	1.30 to 1.40	14.2	5.8	82.5	2.3
	1.40 to 1.60	4.0	16.5	86.5	3.0
	1.60 to 1.80	1.7	33.4	88.2	3.6
	Over 1.80	11.8	77.1	100.0	12.2
100 to 200	Under 1.30	57.0	2.1	57.0	2.1
Weight, 3.7 pct	1.30 to 1.40	20.7	5.7	77.7	3.1
	1.40 to 1.60	5.1	15.9	82.8	3.9
	1.60 to 1.80	2.1	31.9	84.9	4.5
	Over 1.80	15.1	74.1	100.0	15.0
Under 200	Under 1.30	29.4	5.9	29.4	5.9
Weight, 1.7 pct	1.30 to 1.40	36.4	6.5	65.8	6.2
	1.40 to 1.60	8.7	15.5	74.5	7.3
	1.60 to 1.80	3.2	30.9	77.7	8.3
	Over 1.80	22.3	71.9	100.0	22.5
Composite	Under 1.30	81.7	1.4	81.7	1.4
Weight, 100.0 pct	1.30 to 1.40	9.0	6.3	90.7	1.9
	1.40 to 1.60	2.6	17.3	93.3	2.3
	1.60 to 1.80	1.0	37.0	94.3	2.7
	Over 1.80	5.7	75.2	100.0	6.8

<sup>a</sup> Moisture-free basis.

desired, either the tailing or the concentrate may be retreated in a secondary spiral; the middling product often is recirculated through the primary spiral. In treating coal any of these possibilities for retreatment could be utilized. However, in the two commercial plants now employing the spiral for cleaning coal, no retreatment is practiced—the spiral makes a two-product separation in a single pass of the material.

The laboratory spiral, as indicated in fig. 1, is a full-size unit installed in closed circuit with a centrifugal pump and a pump sump to permit rapid testing of small samples. Part of the output of the pump is sent through a centrifuge to provide clarified wash water for this closed-circuit arrangement.

### Test Procedure

In making a test, the pump sump was filled to a fixed level with water, an 8-kg sample of coal was added to the sump, and the pulp was allowed to circulate through the spiral for 2 or 3 min to insure achieving equilibrium conditions. Simultaneous time samples were then collected from the individual refuse ports and from the end discharge of the spiral. In most of the work a splitter was employed to divide the end discharge stream into a washed coal and a middling product, the middling product comprising the inner portion of the stream.

Of the 18 refuse ports provided, only five—one for each of the first five turns of the spiral—were employed. In a typical series of tests on a particular coal, three or four trials were made with different settings of the splitters in the refuse ports in order to determine the optimum conditions required to give a refuse product of the desired quality. Then, with the refuse port openings fixed, several tests

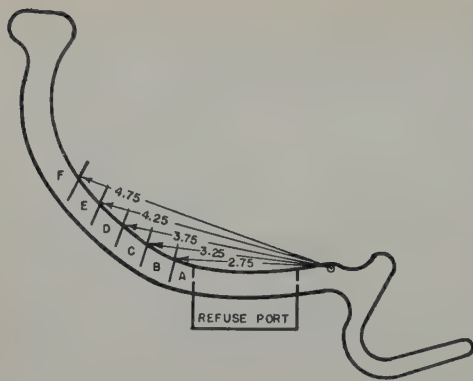


Fig. 2—Cross section of spiral showing zones in stream created by various positions of washed coal-middling splitter.

were made with different settings of the splitter that divided the end-discharge stream into washed coal and middling in order to establish the proper position for this division. With proper operating conditions thus established, a final test was made to obtain samples of all products for float-and-sink and screen-sizing examination.

A middling product was made in nearly all of the tests to obtain more complete information on spiral performance. It was realized that a three-product separation is not desirable in most coal-washing operations. However, it was thought that in this work a middling product should be produced and its character in terms of size and specific-gravity composition determined in order to show the possibilities inherent in a three-product separation. Little work was done towards retreating this middling material, and at the present it is not possible to state definitely whether the middling material should be

recirculated through the spiral or whether it should be retreated by a hydraulic classifier or other suitable means to recover its content of clean coal.

All spiral tests were made at a feed rate of 1 ton of coal per hr and at a concentration of solids in the feed of 18 to 20 pct. The influence of these two variables—solids concentration and feed rate—were not investigated thoroughly because previous published work on the spiral<sup>8, 9</sup> appeared to have demonstrated conclusively that the optimum concentration of solids in the feed was 15 to 20 pct and that feed rates in excess of about 1 ton per hr resulted in impaired efficiency.

#### Black Eagle, W. Va., Coal

The results of a series of tests made to determine the influence of refuse-port settings with Black Eagle coal are presented in table VI.

All were made with the use of one refuse port in each of the top five turns of the spiral; the ports are numbered from the top down. With these five ports three fourths open, as in test 26, the refuse amounted to 11 pct of the feed and contained 23.4 pct ash. Closing the top two ports to one half open and the bottom three ports to one fourth open, as in test 39, reduced the amount of refuse to 4.6 pct of the feed and increased the grade to 39.4 pct ash. Closing the ports still further, as was done in test 37, reduced the production of refuse to only 2.3 pct but increased the grade of refuse to only 43.3 pct ash. Thus, the port openings prevailing in test 39 were considered optimum, even though the ash content of the refuse product was less than 40 pct. The 4.6 pct yield of refuse in this test is a little less than the 5.3 pct of impurity heavier than 1.60 sp gr present in the raw coal. For comparison, the impurity heavier than 1.60 sp gr has an ash content of 66.9 pct.

Table IV. Specific Gravity Analyses, by Size Fractions, of Poplar Ridge, Ky., Coal

Size, Mesh	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative Weight, Pct	Cumulative Ash, <sup>a</sup> Pct
8 to 14 Weight, 21.6 pct	Under 1.30	42.2	3.9	42.2	3.9
	1.30 to 1.40	41.2	10.3	83.4	7.1
	1.40 to 1.60	9.8	21.1	93.2	8.5
	1.60 to 1.80	1.1	36.2	94.3	8.9
	Over 1.80	5.7	65.4	100.0	12.1
14 to 28 Weight, 32.3 pct	Under 1.30	35.1	3.2	35.1	3.2
	1.30 to 1.40	39.9	9.5	75.0	6.6
	1.40 to 1.60	11.8	19.7	86.8	8.3
	1.60 to 1.80	1.9	36.7	88.7	8.9
	Over 1.80	11.3	69.3	100.0	15.8
28 to 48 Weight, 19.3 pct	Under 1.30	33.3	3.0	33.3	3.0
	1.30 to 1.40	36.8	9.0	70.1	6.1
	1.40 to 1.60	13.2	19.2	83.3	8.2
	1.60 to 1.80	2.6	37.3	85.9	9.1
	Over 1.80	14.1	69.7	100.0	17.6
48 to 100 Weight, 12.0 pct	Under 1.30	25.0	3.1	25.0	3.1
	1.30 to 1.40	38.5	8.2	63.5	6.2
	1.40 to 1.60	16.5	18.7	80.0	8.8
	1.60 to 1.80	4.0	37.1	84.0	10.1
	Over 1.80	16.0	68.3	100.0	19.4
100 to 200 Weight, 5.3 pct	Under 1.30	14.7	3.5	14.7	3.5
	1.30 to 1.40	38.7	7.6	53.4	6.5
	1.40 to 1.60	21.3	17.5	74.7	9.6
	1.60 to 1.80	6.1	35.1	80.8	11.5
	Over 1.80	19.2	66.9	100.0	22.2
Under 200 Weight, 9.5 pct	Under 1.30	3.9	7.1	3.9	7.1
	1.30 to 1.40	23.2	7.5	27.1	7.4
	1.40 to 1.60	31.0	14.6	58.1	11.3
	1.60 to 1.80	16.7	27.7	74.8	14.9
	Over 1.80	25.2	63.2	100.0	27.1
Composite Weight, 100.0 pct	Under 1.30	31.0	3.4	31.0	3.4
	1.30 to 1.40	37.7	9.2	68.7	6.6
	1.40 to 1.60	14.6	18.5	83.3	8.7
	1.60 to 1.80	3.8	33.1	87.1	9.7
	Over 1.80	12.9	67.6	100.0	17.2

<sup>a</sup> Moisture-free basis.

Table V. Specific Gravity Analyses, by Size Fractions, of Roslyn, Wash., Coal

Size, Mesh	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative Weight, Pct	Cumulative Ash, <sup>a</sup> Pct
8 to 14 Weight, 31.3 pct	Under 1.30	39.6	5.9	39.6	5.9
	1.30 to 1.40	37.0	11.3	76.6	8.5
	1.40 to 1.60	7.6	26.6	84.2	10.1
	1.60 to 1.80	2.7	43.7	86.9	11.2
	Over 1.80	13.1	79.0	100.0	20.1
14 to 28 Weight, 31.4 pct	Under 1.30	37.2	5.5	37.2	5.5
	1.30 to 1.40	36.9	10.7	74.1	8.1
	1.40 to 1.60	9.3	24.9	83.4	10.1
	1.60 to 1.80	3.2	42.9	86.6	11.2
	Over 1.80	13.4	78.9	100.0	20.3
28 to 48 Weight, 13.2 pct	Under 1.30	28.9	5.2	28.9	5.2
	1.30 to 1.40	41.5	10.5	70.4	8.3
	1.40 to 1.60	10.2	25.0	80.6	10.4
	1.60 to 1.80	3.0	45.9	83.6	11.7
	Over 1.80	16.4	74.5	100.0	22.0
48 to 100 Weight, 11.8 pct	Under 1.30	23.2	4.8	23.2	4.8
	1.30 to 1.40	40.9	10.3	64.1	6.3
	1.40 to 1.60	13.3	23.9	77.4	11.0
	1.60 to 1.80	4.2	43.3	81.6	12.7
	Over 1.80	18.4	69.9	100.0	23.2
100 to 200 Weight, 7.7 pct	Under 1.30	18.4	4.5	18.4	4.5
	1.30 to 1.40	40.9	10.1	59.3	8.4
	1.40 to 1.60	15.5	22.4	74.8	11.3
	1.60 to 1.80	4.8	41.9	79.6	13.1
	Over 1.80	20.4	67.0	100.0	24.1
Under 200 Weight, 4.6 pct	Under 1.30	9.5	5.2	9.5	5.2
	1.30 to 1.40	35.1	9.7	44.6	8.7
	1.40 to 1.60	22.9	18.7	67.5	12.1
	1.60 to 1.80	5.5	36.7	73.0	14.0
	Over 1.80	27.0	65.1	100.0	27.8
Composite Weight, 100.0 pct	Under 1.30	32.5	5.5	32.5	5.5
	1.30 to 1.40	38.2	10.7	70.7	8.3
	1.40 to 1.60	10.5	24.2	81.2	10.4
	1.60 to 1.80	3.4	42.9	84.6	11.7
	Over 1.80	15.4	74.7	100.0	21.4

<sup>a</sup> Moisture-free basis.



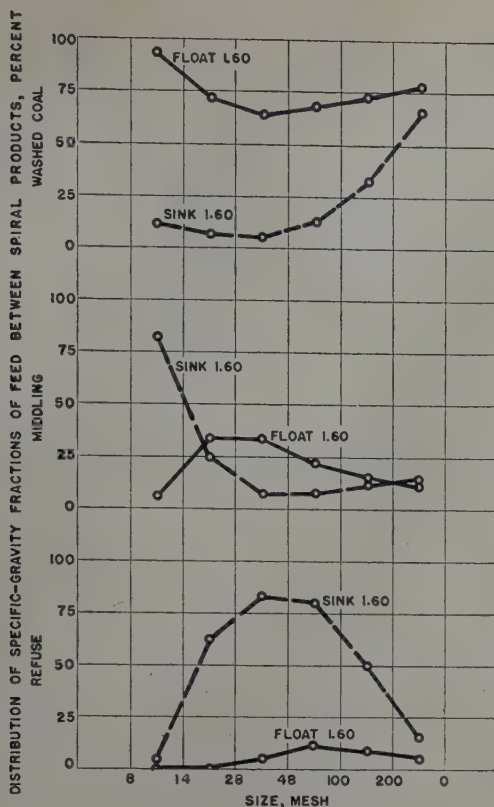


Fig. 3—Percentage distribution of the specific gravity fractions of Black Eagle coal between spiral products by size fractions.

The results presented in table VI demonstrate that with this coal the spiral was unable to effect a separation between coal and impurity accurately enough to give a refuse product of the quality generally considered acceptable.

Table VII shows the results of a series of five tests made with this coal to determine the influence of the split between washed coal and middling. For this series of tests, the washed coal-middling splitter was set according to the diagram shown in fig. 2. With the splitter set at 2.75 in., a washed coal amounting to 83 pct of the feed and containing 5.8 pct of ash was produced. With progressively increasing amounts of material directed into the middling product, as in tests 43, 44, and 45, the ash content of the washed coal was reduced steadily to a minimum of 4.9 pct. In test 46, however, in which the splitter was set to include in the middling all but the extreme outer portion of the stream, the ash content of the washed coal increased to 5.8 pct. As the increase in the proportion of middling was considered excessive for the accompanying reduction in the ash content of the washed coal, test 42 was considered to represent the optimum division between washed coal and middling.

Table VIII shows the results of float-and-sink tests made at 1.60 sp gr on the spiral products of test 42, the test in which the yield of washed coal was 83.0 pct and yield of middling 13.3 pct.

The washed coal contained 1.1 pct of impurity heavier than 1.60 sp gr, and the presence of this impurity increased its ash content by 0.5 pct. The refuse product contained over 50 pct of coal lighter than 1.60 sp gr. The middling product contained roughly three times the amount of heavy impurity present in the feed.

Screen analyses of these float-and-sink fractions

Table VI. Influence of Port Openings on Amount and Quality of Refuse, Black Eagle Coal

Product	Port Opening	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight Pct	Ash, <sup>a</sup> Pct
Test 26					
Refuse port 1	3/4	1.6	39.1	1.6	39.1
2	3/4	3.6	32.8	5.2	34.7
3	3/4	1.6	15.7	6.8	30.3
4	3/4	1.9	12.4	8.7	26.4
5	3/4	2.3	12.3	11.0	23.4
End discharge		89.0	6.7	100.0	8.5
Test 39					
Refuse port 1	1/2	1.1	42.4	1.1	42.4
2	1/2	2.0	48.3	3.1	46.2
3	1/4	0.4	37.8	3.5	45.2
4	1/4	0.4	22.1	3.9	42.9
5	1/4	0.7	20.0	4.6	39.4
End discharge		95.4	7.4	100.0	8.9
Test 37					
Refuse port 1	3/8	0.5	32.1	0.5	32.1
2	3/8	1.2	54.9	1.7	48.2
3	1/4	0.3	33.4	2.0	46.0
4	1/8	0.1	32.8	2.1	45.3
5	1/8	0.2	22.0	2.3	43.3
End discharge		97.7	7.2	100.0	8.0

<sup>a</sup> Moisture-free basis.

Table VII. Influence of Split Between Washed Coal and Middling, Black Eagle Coal

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
					Weight, Pct	Ash, <sup>a</sup> Pct
42	Washed coal	2.75	83.0	5.8	83.0	5.8
	Middling		13.3	16.0	96.3	7.2
	Refuse		3.7	40.3	100.0	8.4
43	Washed coal	3.25	67.5	5.2	67.5	5.2
	Middling		28.7	11.2	96.2	7.0
	Refuse		3.8	40.3	100.0	8.3
44	Washed coal	3.75	45.7	5.0	45.7	5.0
	Middling		50.3	10.1	96.0	7.7
	Refuse		4.0	40.3	100.0	9.0
45	Washed coal	4.25	23.7	4.9	23.7	4.9
	Middling		72.4	8.3	96.1	7.5
	Refuse		3.9	40.3	100.0	8.7
46	Washed coal	4.75	7.4	5.8	7.4	5.8
	Middling		88.6	7.1	96.0	7.0
	Refuse		4.0	40.3	100.0	8.3

<sup>a</sup> Moisture-free basis.

Table VIII. Float-and-Sink Separations at 1.60 Sp Gr on Products from Spiral Test of Black Eagle Coal

Product	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
Washed coal	Under 1.60	98.9	5.2	98.9	5.2
Weight, 83.0 pct	Over 1.60	1.1	53.4	100.0	5.7
Middling	Under 1.60	84.6	6.3	84.6	6.3
Weight, 13.3 pct	Over 1.60	15.4	69.3	100.0	16.0
Refuse	Under 1.60	52.6	7.7	52.6	7.7
Weight, 3.7 pct	Over 1.60	47.4	78.9	100.0	41.4

<sup>a</sup> Moisture-free basis.

Table IX. Screen Analyses of Specific Gravity Fractions of Spiral Products of Black Eagle Coal, Pct

Product	Weight, Pct	Screen Size, Mesh					
		8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	83.0						
Under 1.60	98.9	48.3	26.3	11.5	6.3	3.5	4.1
Over 1.60	1.1	55.8	9.7	4.4	5.3	8.8	16.0
Middling	13.3						
Under 1.60	84.6	6.5	42.5	31.1	12.6	3.9	3.4
Over 1.60	15.4	69.0	21.6	3.3	2.0	1.6	2.5
Refuse	3.7						
Under 1.60	52.6	0.0	4.2	32.5	35.3	14.8	13.2
Over 1.60	47.4	4.5	41.7	31.7	16.3	4.0	1.8

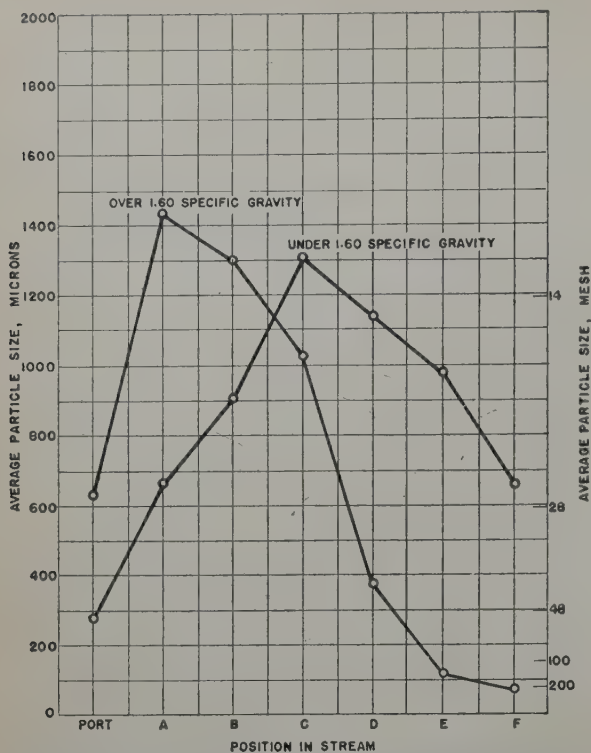


Fig. 4—Average particle size of specific gravity fractions of feed carried in various portions of stream.

reveal some interesting aspects of spiral performance, as indicated in table IX. The impurity in the washed coal was principally coarse material, 55.8 pct being in the 8 to 14-mesh size fraction. The coal contaminating the refuse product, on the other hand, contained no material coarser than 14-mesh and bulked largely between 28 and 100-mesh. The coal in the middling product was noticeably finer than that in the washed coal, concentrating largely in the range from 14 to 48-mesh. The 15 pct of heavy impurity associated with this coal in the middling product was distinctly coarse material; nearly 70 pct was in the 8 to 14-mesh size fraction.

These figures thus demonstrate that the spiral,

Table X. Screen Analyses of Specific Gravity Fractions Flowing in Various Zones of Spiral Stream, Pct.

Product and Specific Gravity	Weight, Pct	Screen Size, Mesh					
		8-14	14-28	28-48	48-100	100-200	Under 200
Refuse							
Under 1.60	3.9	0.0	4.2	32.5	35.3	14.8	13.2
Over 1.60	47.4	4.5	41.7	31.7	16.3	4.0	1.8
Zone A	13.2						
Under 1.60	84.6	6.6	42.4	30.9	12.8	3.9	3.4
Over 1.60	15.4	68.7	21.7	3.4	2.1	1.6	2.5
Zone B	15.5						
Under 1.60	97.2	23.3	42.4	21.4	7.2	3.1	2.6
Over 1.60	2.8	62.2	17.2	5.7	4.6	4.6	5.7
Zone C	16.9						
Under 1.60	98.3	55.2	32.0	8.7	3.3	0.7	0.1
Over 1.60	1.7	48.3	11.7	6.7	8.3	10.0	15.0
Zone D	26.7						
Under 1.60	99.4	46.1	28.4	11.0	6.2	1.8	6.5
Over 1.60	0.6	9.4	12.5	9.4	12.5	15.6	40.6
Zone E	16.3						
Under 1.60	99.7	38.4	20.5	16.3	12.1	7.9	4.8
Over 1.60	0.3	0.0	6.1	6.1	17.5	26.3	43.9
Zone F	7.5						
Under 1.60	98.9	26.2	7.4	11.4	16.8	14.8	23.5
Over 1.60	1.1	0.0	1.2	1.2	6.1	20.2	71.2

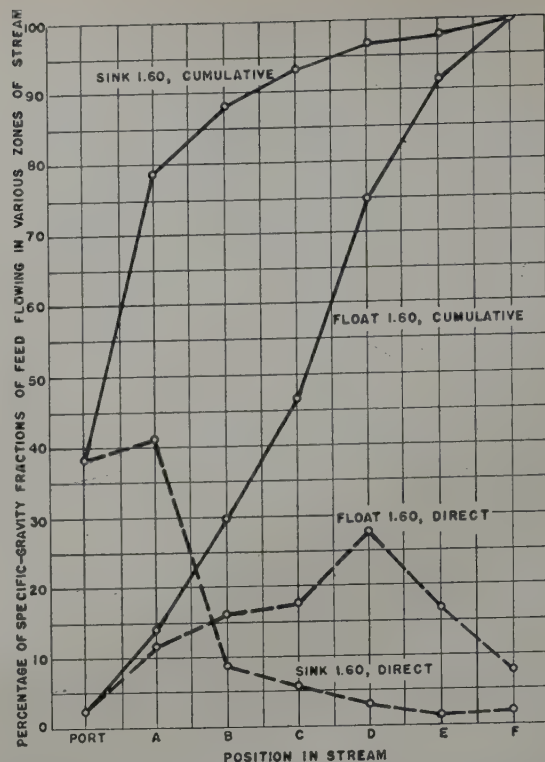


Fig. 5—Percentage of specific gravity fractions of feed carried in various portions of stream.

like most other gravity concentrating devices, effects a separation by particle size as well as by specific gravity.

The influence of particle size in determining the recovery of the specific gravity fractions of the feed in the various spiral products is illustrated clearly in fig. 3. These curves indicate, for example, that of the total 8 to 14-mesh coal lighter than 1.60 sp gr present in the feed, 94 pct was recovered in the washed coal. The corresponding recovery decreased to reach a minimum of 62 pct in the 28 to 48-mesh size, then increased again with further decrease in particle size. Similarly, of the 8 to 14-mesh impurity heavier than 1.60 sp gr present in the spiral feed, only 4 pct went into the refuse product, 84 pct was discharged as middling, and 12 pct entered the washed coal. The elimination of heavy impurity in the refuse product reached a maximum of 85 pct in the 28 to 48-mesh size and then decreased again with further decrease in particle size.

The influence of particle size on the performance of the spiral is illustrated even more clearly in the results of a series of tests on Black Eagle coal in which the character of the material flowing in each ½-in. increment of the stream was determined. Operating conditions for this series of tests were the same as those prevailing in test 42, which was described previously. The zones or increments of the spiral stream used in this work are illustrated in fig. 2. Zone A included all of the innermost portion of the stream except the material drawn off through the refuse ports. Zones B through E were approximately ½ in. wide, as measured along the cross-section of the spiral, and zone F included all of the stream beyond E.

Table X shows the proportion of the spiral feed flowing in each zone of the stream, the percentages of materials heavier and lighter than 1.60 sp gr in each zone, and the screen analyses of these specific



gravity fractions. As shown by the tabular data in table X and by the graphic presentation in fig. 4, the clean coal lighter than 1.60 sp gr becomes progressively coarser with greater distance from the inside of the spiral until maximum size is achieved in zone C. The clean coal flowing in the stream beyond this point becomes progressively finer.

The distribution of heavy impurity in the stream differs markedly from that of the clean coal. As shown by the data in table X and also by the graphic presentation in fig. 5, the material flowing in zone A just outside the refuse ports contains a high proportion of impurity. In fact, reference to fig. 4 indicates that, whereas 38 pct of the total impurity in the feed was stratified along the inner edge of the stream where it could be drawn through the refuse ports, 41 pct of the impurity was stratified just a little farther out in the stream where it could not be caught in the ports. The coarseness of the impurity flowing in zone A accounts for its presence that far out in the stream; the impurity in this zone contained 68.7 pct of material coarser than 14-mesh, while the impurity removed through the refuse ports contained only 4.5 pct in this size fraction. The proportion of the total impurity accounted for in the other zones of the stream decreases rather rapidly to reach a minimum of 1.0 pct in zone E. The particle size of the impurity also decreases rapidly in these zones.

To generalize on the performance of the spiral, the modifying influence of size on specific gravity is such that intermediate-size particles tend to stratify in the inner portion of the stream, coarser particles are stratified farther out, and the finest particles remain suspended in the water to assemble predominantly in the extreme outer portion of the stream. In consequence of this behavior, the washed coal is contaminated with impurity principally in the coarsest and finest sizes, while conversely, the major loss of clean coal in the refuse product occurs in the intermediate sizes.

The magnitude of this loss of coal to the refuse product in the intermediate sizes, in terms of the efficiency of a two-product separation of this coal, is indicated in table XI. Efficiency, as the term is used here, is the ratio of the yield of washed coal to the yield of float coal of the same ash content shown to be present in the feed by specific gravity analysis. As indicated in table XI, operation of the spiral to reduce the ash content of this coal from 8.4 to 7.2 pct by the removal of 3.7 pct of refuse resulted in an efficiency of 97.4 pct. If greater ash reduction is attempted in a two-product separation, efficiency decreases rapidly. Producing a washed product of 5.2 pct ash, corresponding to a separation at 1.60 sp gr, results in an efficiency of only 71 pct.

For the sake of comparison, this coal unquestionably could be washed at 1.60 sp gr on a table with an efficiency of 99 pct.

#### Clements, Ala., Coal

The Clements coal contains slightly more heavy impurity than that in the Black Eagle coal already described, and the impurity is considerably finer. From the standpoint of specific gravity composition, the Clements coal is very easy to wash since it contains little material of intermediate density. Inasmuch as the behavior of these two coals in the spiral was quite similar, less detailed data will be presented for the Clements coal.

Table XII indicates the results obtained in two trials made to determine the optimum settings of

**Table XI. Efficiency of Washing Black Eagle Coal to Various Ash Contents**

Test No.	Ash <sup>a</sup> in Washed Coal, Pct	Yield of Washed Coal, Pct	Yield of Float Coal, Pct	Efficiency, Pct
42	7.2	96.3	98.9	97.4
26	6.7	89.0	98.3	90.5
42	5.8	83.0	96.7	85.8
43	5.2	67.5	95.0	71.1

<sup>a</sup> Moisture-free basis.

**Table XII. Influence of Port Openings on Amount and Quality of Refuse, Clements Coal**

Product	Port Opening	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
Test 40					
Refuse port 1	½	1.2	50.4	1.2	50.4
2	½	1.5	59.5	2.7	55.5
3	¾	0.3	52.8	3.0	55.2
4	¾	0.3	30.6	3.3	53.0
5	¾	0.5	27.5	3.8	49.6
End discharge		96.2	4.2	100.0	5.9
Test 41					
Refuse port 1	¾	2.3	43.4	2.3	43.4
2	¾	2.4	43.5	4.7	43.5
3	½	0.6	32.4	5.3	42.2
4	½	0.7	20.7	6.0	39.7
5	½	1.1	15.5	7.1	35.9
End discharge		92.9	3.9	100.0	6.1

<sup>a</sup> Moisture-free basis.

**Table XIII. Influence of Split Between Washed Coal and Middling, Clements Coal**

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
					Weight, Pct	Ash, <sup>a</sup> Pct
47	Washed coal	2.75	86.4	3.4	86.4	3.4
	Middling		10.1	12.4	96.5	4.3
	Refuse		3.5	49.7	100.0	5.9
48	Washed coal	3.25	75.5	2.9	75.5	2.9
	Middling		20.9	10.8	96.4	4.6
	Refuse		3.6	49.5	100.0	6.2
49	Washed coal	3.75	56.9	3.1	56.9	3.1
	Middling		39.7	6.7	96.6	4.6
	Refuse		3.4	49.5	100.0	6.1
50	Washed coal	4.25	32.4	3.3	32.4	3.3
	Middling		64.2	5.5	96.6	4.8
	Refuse		3.4	49.5	100.0	6.3
51	Washed coal	4.75	21.5	2.6	21.5	2.6
	Middling		75.1	5.6	96.6	4.9
	Refuse		3.4	49.5	100.0	6.4

<sup>a</sup> Moisture-free basis.

the refuse ports. In test 40, made with the upper two ports one half open and the three lower ports one fourth open, a refuse product amounting to 3.8 pct and containing 49.6 pct ash was produced. Removing a greater percentage of refuse would have been desirable for, as shown by the float-and-sink data in table III, the raw coal contained 6.7 pct impurity heavier than 1.60 sp gr. However, when the ports were opened wider for test 41, the quality of the refuse was decreased from nearly 50 pct ash to 36 pct. In the light of these results, the port openings shown for test 40 were adopted as standard for all subsequent work.

The results obtained with various settings of the washed coal-middling splitter are presented in table XIII. A washed coal containing 3.4 pct ash was obtained at a yield of 86.4 pct in test 47, in which a middling product amounting to 10.1 pct of the feed was made. As with Black Eagle, moving the splitter farther out into the stream gave a washed product of somewhat lower ash content but only with drasti-

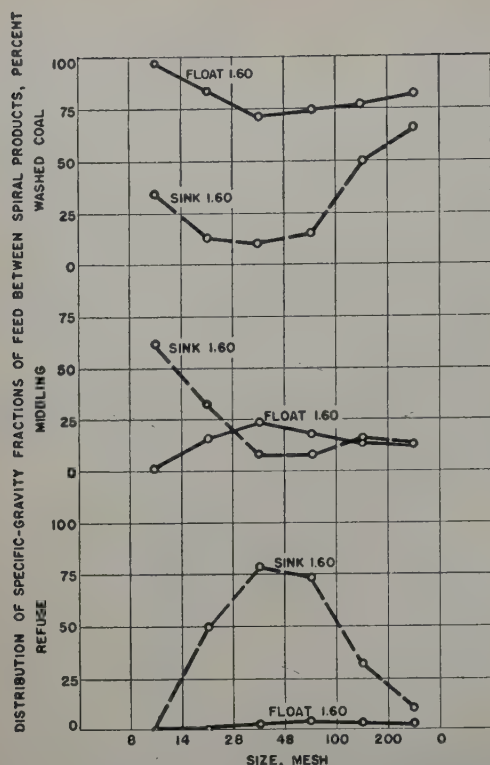


Fig. 6—Percentage distribution of the specific gravity fractions of Clements coal between spiral products by size fractions.

cally reduced yield. For example, comparing tests 47 and 48, to reduce the ash content of the washed coal from 3.4 to 2.9 pct, the percentage of the feed directed into the middling product had to be increased from 10 to 21 pct. Accordingly, the conditions prevailing in test 47 were considered optimum, and samples of all products made in this test were subjected to float-and-sink and screen-sizing examination.

The results of this examination are shown in table XIV.

The washed coal contained 1.9 pct of impurity heavier than 1.60 sp gr, and the presence of this impurity increased its ash content from 2.3 to 3.4 pct. The refuse contained 42.1 pct of coal lighter than 1.60 sp gr, and the middling product was composed of 85 pct coal and 15 pct impurity. Screen analyses of these specific gravity fractions are given in table XV.

The impurity contaminating the washed coal was substantially finer than that in the washed product from Black Eagle, not because of any difference in spiral operation but rather because of the fineness of the impurity in the raw coal. As with Black Eagle,

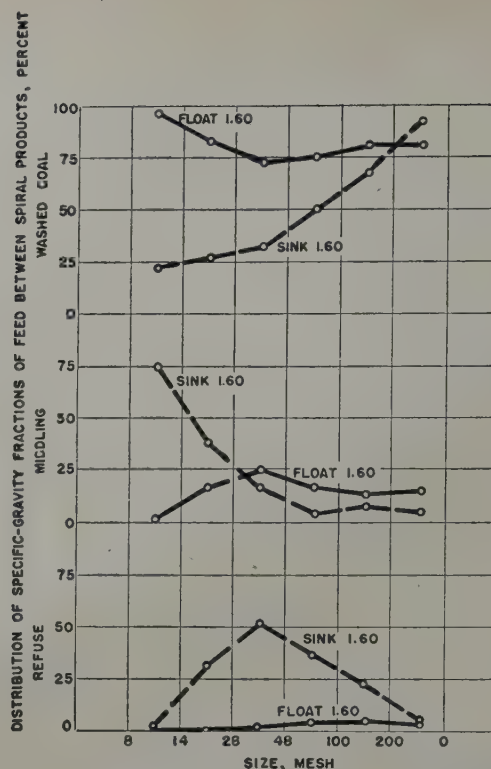


Fig. 7—Percentage distribution of the specific gravity fractions of Poplar Ridge coal between spiral products by size fractions.

the heavy impurity in the middling product was substantially coarser than that in the washed coal, and the impurity in both was much coarser than that drawn off through the refuse ports. The clean coal lost in the refuse product tended to be of intermediate size.

Fig. 6 illustrates the magnitude of the influence of particle size by showing the recovery in each spiral product of the specific gravity and size fractions present in the feed to the spiral. The curves in fig. 6 do not differ materially from those given in fig. 3 for Black Eagle coal. They show the same maximum elimination of impurity in the intermediate sizes and the same maximum loss of clean coal in the intermediate sizes.

On the whole, operation of the spiral on Clements coal was less satisfactory than on Black Eagle. The specific gravity analyses of Clements coal given in table III indicate that a reduction in ash content from 6.8 to 2.3 pct should be obtainable with high efficiency. The spiral was unable to produce a washed product containing 2.3 pct ash, and, even in making a washed product of substantially higher ash content, an appreciable loss of clean coal occurred.

#### Poplar Ridge, Ky., Coal

The Poplar Ridge coal is much more difficult to wash than either of the two coals described previously because it contains substantially more material of intermediate density and a great deal more heavy impurity. This coal might be described as moderately difficult to wash, while on the same basis the two previous coals would be described as very easy to wash. Considering all sizes together, this coal contains 16.7 pct of impurity heavier than 1.60 sp gr, and the proportion of impurity increases steadily with decrease in particle size. The sizes finer than 100-mesh contain over 30 pct of impurity.

The results of two trials made to determine the

Table XIV. Float-and-Sink Separations at 1.60 Sp Gr on Products from Spiral Test on Clements Coal

Product	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
Washed coal Weight, 86.4 pct	Under 1.60	98.1	2.3	98.1	2.3
	Over 1.60	1.9	58.3	100.0	3.4
Middling Weight, 10.1 pct	Under 1.60	85.5	3.0	85.5	3.0
	Over 1.60	14.5	68.1	100.0	12.4
Refuse Weight, 3.5 pct	Under 1.60	42.1	4.8	42.1	4.8
	Over 1.60	57.9	82.4	100.0	49.7

<sup>a</sup> Moisture-free basis.



**Table XV. Screen Analyses of Specific Gravity Fractions of Spiral Products of Clements Coal, Pct**

Product	Weight, Pct	Screen Size, Mesh					
		8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	86.4						
Under 1.60	98.1						
Over 1.60	1.9	57.5	23.5	9.6	5.4	2.4	1.6
		28.9	8.6	5.9	7.0	12.8	36.8
Middling	10.1						
Under 1.60	85.5	7.0	39.9	32.3	14.3	4.3	2.2
Over 1.60	14.5	53.9	25.7	5.2	4.5	5.0	5.7
Refuse	3.5						
Under 1.60	42.2	9.1	10.7	33.6	29.2	11.0	6.4
Over 1.60	57.8	1.3	28.1	35.9	24.3	6.5	3.9

**Table XVI. Influence of Port Openings on Amount and Quality of Refuse, Poplar Ridge Coal**

Product	Port Opening	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
Test 24					
Refuse port 1	3/4	3.0	47.6	3.0	47.6
2	3/4	4.2	45.2	7.2	46.2
3	1/2	1.0	40.5	8.2	45.5
4	1/2	1.2	32.8	9.4	43.9
5	1/2	1.6	25.0	11.0	41.1
End discharge		89.0	11.5	100.0	14.8
Test 25					
Refuse port 1	1/2	1.5	52.3	1.5	52.3
2	1/2	3.5	51.9	5.0	52.0
3	3/4	0.7	44.0	5.7	51.0
4	3/4	0.6	43.8	6.3	50.3
5	3/4	0.5	35.0	6.8	49.2
End discharge		93.1	11.9	100.0	14.5

<sup>a</sup> Moisture-free basis.

optimum opening of the refuse ports are shown in table XVI. In test 25, made with the top three refuse ports one half open and the lower two ports one fourth open, 6.8 pct of refuse containing about 50 pct ash was produced. This quantity of refuse is less than one half of the amount of impurity heavier than 1.60 sp gr shown to be present in the feed by the specific gravity analysis given in table IV. However, as demonstrated by the results of test 24, opening the refuse ports wider to extract a higher percentage of refuse impaired greatly the quality of this product. The port openings of test 25 were therefore adopted as optimum for subsequent work.

The results of five tests made to explore the influence of the split between washed coal and middling are given in table XVII. It will be observed that with this coal the production of a middling product was not an effective means of reducing the ash content of the washed coal. In test 33, for example, directing 14 pct of the feed into a middling product reduced the ash content of the washed coal by only 1.1 pct. Increasing the cut to middling, as was done in the other four tests of this series, had substantially no influence on the quality of the resulting washed coal. Consequently, the remainder of the performance data for this coal are presented in terms of a two-product separation.

Table XVIII shows the results of float-and-sink separations made at 1.60 sp gr on washed coal and refuse produced by the spiral when operated with the refuse port openings indicated for test 25. The washed coal contained nearly 15 pct of impurity heavier than 1.60 sp gr, and the presence of this impurity increased the ash content of the washed coal from 7.9 to 12.6 pct. The refuse had an ash content of about 54 pct and contained 21.2 pct of coal lighter than 1.60 sp gr.

Screen analyses of the specific gravity fractions of table XVIII are given in table XIX. The heavy im-

**Table XVII. Influence of Split Between Washed Coal and Middling, Poplar Ridge Coal**

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
					Weight, Pct	Ash, <sup>a</sup> Pct
33	Washed coal	2.75	79.4	11.8	79.4	11.8
	Middling		14.0	19.1	93.4	12.9
	Refuse		6.6	53.7	100.0	15.6
29	Washed coal	3.25	65.2	12.0	65.2	12.0
	Middling		28.7	14.0	93.9	12.6
	Refuse		6.1	51.6	100.0	15.0
30	Washed coal	3.75	44.7	11.9	44.7	11.9
	Middling		49.4	12.8	94.1	12.4
	Refuse		5.9	53.7	100.0	14.8
31	Washed coal	4.25	23.2	16.1	23.2	16.1
	Middling		70.9	11.9	94.1	12.9
	Refuse		5.9	52.6	100.0	15.3
32	Washed coal	4.75	10.8	16.4	10.8	16.4
	Middling		83.2	12.6	94.0	13.0
	Refuse		6.0	54.3	100.0	15.5

<sup>a</sup> Moisture-free basis.

**Table XVIII. Float-and-Sink Separations at 1.60 Sp Gr on Products from Spiral Test of Poplar Ridge Coal**

Product	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
Washed coal	Under 1.60	85.1	7.9	85.1	7.9
Weight, 94.3 pct	Over 1.60	14.9	39.6	100.0	12.6
Refuse	Under 1.60	21.2	13.0	21.2	13.0
Weight, 5.7 pct	Over 1.60	78.8	64.9	100.0	53.9

<sup>a</sup> Moisture-free basis.

**Table XIX. Screen Analyses of Specific Gravity Fractions of Spiral Products of Poplar Ridge Coal, Pct**

Product	Weight, Pct	Screen Size, Mesh					
		8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	94.3						
Under 1.60	85.1	28.9	38.0	17.4	9.4	4.0	2.3
Over 1.60	14.9	6.4	17.9	9.6	8.1	7.6	50.4
Refuse	5.7						
Under 1.60	21.2	0.0	4.1	37.9	35.2	15.2	7.6
Over 1.60	78.8	0.7	30.8	38.1	17.7	6.3	6.4

purity in the washed coal was predominantly fine material, 54 pct being finer than 200-mesh. The clean coal lost in the refuse product was largely of 28 to 100-mesh size, and, in this respect, performance of the spiral on this coal was the same as on the other two coals previously described.

Fig. 7, which shows the distribution made by the spiral of each size and specific gravity fraction of the feed between the various spiral products, differs from the corresponding curves given for Black Eagle and Clements coals in only one particular. With this coal the proportion of heavy impurity entering the washed product increased steadily with decrease in particle size instead of reaching a minimum in the intermediate sizes. This circumstance probably is attributable to the fineness of the impurity in the Poplar Ridge coal.

With a coal containing so much fine impurity, desliming of the washed coal from the spiral would have reduced its ash content substantially.

#### Roslyn, Wash., Coal

Roslyn coal, like Poplar Ridge, contains considerable heavy impurity and a moderate amount of material of intermediate density. Although the finer

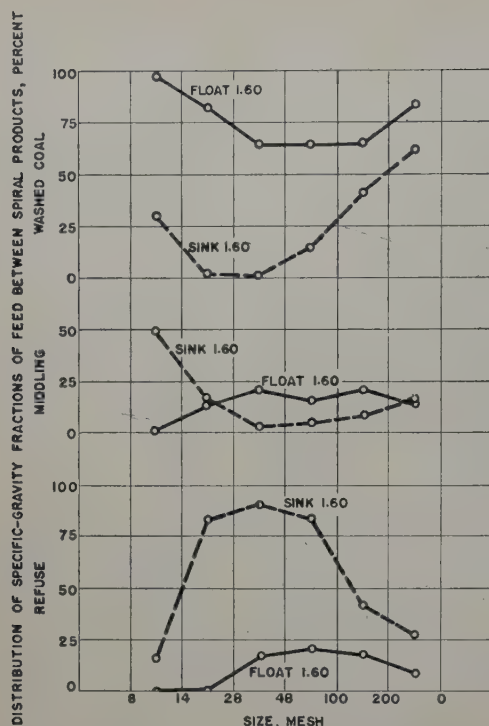


Fig. 8—Percentage distribution of the specific gravity fractions of Roslyn coal between spiral products by size fractions.

sizes are somewhat dirtier than the coarse material, this trend is not so pronounced as in the Poplar Ridge coal.

A great many spiral tests have been made with Roslyn coal, and no difficulty was experienced in setting the refuse ports to obtain a refuse product ranging from 56 to 59 pct ash. Refuse of better grade than this could not be made in appreciable quantity. Incidentally, with this coal, it was found that the same quantity and grade of refuse could be produced using only the top three ports as by using five ports, all only partly open. Thus, in making a two-product separation on Roslyn coal, the spiral could be shortened from six turns to three turns without any sacrifice in efficiency.

The results of a series of five trials made to determine the influence of the washed coal-middling splitter setting are given in table XX. The ash con-

Table XX. Influence of Split Between Washed Coal and Middling, Roslyn Coal

Test No.	Product	Splitter Setting, Inches	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
					Weight, Pct	Ash, <sup>a</sup> Pct
60	Washed coal	2.75	71.8	11.8	71.8	11.8
	Middling		12.2	29.0	84.0	14.3
	Refuse		16.0	58.8	100.0	21.4
61	Washed coal	3.25	60.3	11.2	60.3	11.2
	Middling		24.4	23.4	84.7	14.7
	Refuse		15.3	57.5	100.0	21.3
62	Washed coal	3.75	44.0	11.2	44.0	11.2
	Middling		39.3	17.8	83.3	14.3
	Refuse		16.7	59.3	100.0	21.8
63	Washed coal	4.25	22.0	12.9	22.0	12.9
	Middling		61.7	15.7	83.7	15.0
	Refuse		16.3	57.0	100.0	21.8
64	Washed coal	4.75	10.5	16.8	10.5	16.8
	Middling		74.0	15.1	84.5	15.3
	Refuse		15.5	56.5	100.0	21.7

<sup>a</sup> Moisture-free basis.

tent of the washed coal was reduced from 14.3 to 11.8 pct by making a middling product amounting to 12 pct of the feed. However, doubling the amount of material directed into the middling product only reduced the ash in the washed coal an additional 0.6 pct.

To obtain samples for float-and-sink tests, a trial was made with the spiral adjustments prevailing in test 60 of table XX. The spiral products from this test were deslimed at 200-mesh to eliminate a part of the clay slimes that form through the disintegration of the clay associated with Roslyn coal. The slimes removed amounted to 3.9 pct of the spiral feed and contained 50 pct ash. The deslimed spiral products were separated at 1.60 sp gr. The results are given in table XXI. The washed coal contained 3.7 pct of impurity heavier than 1.60 sp gr, and the presence of this material increased its ash content from 9.6 to 11.4 pct. The refuse contained about 28 pct of coal lighter than 1.60 sp gr, and the middling product was composed of 70 pct coal accompanied by 30 pct impurity.

Screen analyses of these float-and-sink fractions are given in table XXII. As with the other coals tested, the impurity contaminating the washed coal was principally material coarser than 14-mesh, together with slimes finer than 100-mesh. Conversely, the coal in the refuse product bulked largely between 28 and 100-mesh. The coal in the middling was substantially finer than the accompanying impurity.

The performance of the spiral on Roslyn coal is summarized best in fig. 8, which shows how the specific gravity fractions of the spiral feed were divided into washed coal, middling, and refuse in each size fraction. Just as with the other coals described previously, the recovery of clean coal was best in the coarsest and finest sizes, but the elimination of impurity was poorest in these sizes.

Table XXIII was prepared to show the type of two-product separation possible with the spiral on this coal. These data are from the test described previously in which the spiral products were deslimed; the two-product separation was created by adding the middling to the washed coal. In the 8 to 14-mesh size, little ash reduction was effected because this size of material reports almost wholly to the washed product. The spiral performed best on the 14 to 28-mesh size in which the ash was reduced from 20.4 to 11.1 pct with an efficiency of 97.6 pct. In the 28 to 48 and 48 to 100-mesh sizes the efficiency was lowest, for these are the sizes of coal that characteristically stratify with the refuse product and are thus lost. In the sizes under 100-mesh, efficiency improved, but ash reduction suffered because most of the fine material tended to accompany the washed coal.

#### Spiral Test on Classified Roslyn Coal

The performance data included in this report for four coals having materially different washability characteristics suggest that the ideal feed for the spiral—to permit it to make maximum ash reduction with minimum loss of coal—would be one in which the clean coal was of 8 to 28-mesh size and the accompanying impurity was of 28 to 100-mesh size. These ideal specifications could not be met by any natural raw coal, but they are approached by a hydraulically classified material. For example, the products from the intermediate spigots of a multiple-spigot classifier are comparatively free of all finer material, and the coal is substantially coarser than the accompanying heavy impurity.



**Table XXI. Float-and-Sink-Separations at 1.60 Sp Gr on Products from Spiral Test of Roslyn Coal**

Product	Specific Gravity	Weight, Pct	Ash, <sup>a</sup> Pct	Cumulative	
				Weight, Pct	Ash, <sup>a</sup> Pct
Washed coal	Under 1.60	96.4	9.6	96.4	9.6
Weight, 72.6 pct	Over 1.60	3.6	59.4	100.0	11.4
Middling	Under 1.60	69.2	11.7	69.2	11.7
Weight, 12.7 pct	Over 1.60	30.8	68.4	100.0	29.2
Refuse	Under 1.60	27.6	11.3	27.6	11.3
Weight, 14.7 pct	Over 1.60	72.4	76.7	100.0	58.6

<sup>a</sup> Moisture-free basis.

**Table XXII. Screen Analyses of Specific Gravity Fractions of Spiral Products from Roslyn Coal, Pct**

Product	Weight, Pct	Screen Size, Mesh					
		8-14	14-28	28-48	48-100	100-200	Under 200
Washed coal	72.6						
Under 1.60	96.4	52.5	26.5	10.6	6.2	2.8	1.4
Over 1.60	3.6	32.7	7.5	3.2	9.9	19.8	26.9
Middling	12.7						
Under 1.60	69.2	8.0	41.1	31.4	13.8	3.9	1.8
Over 1.60	30.8	72.1	13.1	3.3	4.0	3.7	3.8
Refuse	14.7						
Under 1.60	27.6	0.0	9.9	43.7	34.1	10.0	2.3
Over 1.60	72.4	13.9	35.4	26.1	16.7	5.0	2.9

In an effort to demonstrate that a hydraulically classified feed would permit better spiral operation, a sample of Roslyn coal was treated in a single-spirot classifier adjusted to remove as a spigot product only the coarse impurity. The overflow product from the classifier was then pumped through a cyclone to eliminate part of the material finer than 200-mesh. The resulting roughly classified product formed the feed for a spiral test in which a two-product separation was made.

The results of this test are compared with those for a test on a natural feed in table XXIV. With the spiral adjusted to give a refuse product of 60 pct ash, corresponding to the best refuse obtainable in tests on natural coal, it was possible to obtain a washed coal containing 11.7 pct ash instead of 14.3 pct. The washed coal from the classified feed contained only one half as much heavy impurity as that in the washed product obtained from a natural feed. The screen analyses of the heavy impurity contaminating the washed coal in these two tests demonstrate that classification of the feed minimized the amount of coarse impurity entering the washed coal. The presence of 26 pct of heavy impurity finer than 200-mesh in the washed coal from the classified feed is attributable largely to the fact that the clay associated with Roslyn coal slimes so readily. Had this test been made on a coal containing impurity that was more stable in water, the improvement possible through classification would have been more pronounced.

These rather fragmentary results appear to bear out the contention that preclassification of spiral feed would be beneficial. It must be realized also that silt bank material or the underflow product of a thickener operated as a hydroseparator such as currently are being treated in the spiral<sup>8</sup> constitute semiclassified feeds.

#### Acknowledgments

This investigation was conducted by the U. S. Bureau of Mines in cooperation with the School of Mineral Engineering of the University of Washington. The work was done under the general direction

**Table XXIII. Ash Content, Yield, and Efficiency by Particle Size for a Two-Product Separation of Roslyn Coal**

Size, Mesh	Weight, Pct	Ash <sup>a</sup> Content, Pct		Yield, Pct		Efficiency, Pct
		Feed	Washed Coal Refuse	Washed Coal	Float Coal	
8 to 14	42.5	17.7	15.4	80.4	96.7	98.6
14 to 28	27.0	20.4	11.1	70.7	84.4	97.6
28 to 48	15.0	22.2	9.9	50.7	69.3	85.1
48 to 100	9.1	24.2	11.9	47.2	65.2	77.3
100 to 200	3.9	27.2	20.7	46.9	76.2	80.1
Under 200	2.5	38.5	35.2	58.8	85.6	88.2
Composite	100.0	20.6	14.0	58.6	85.3	91.9

<sup>a</sup> Moisture-free basis.

**Table XXIV. Comparison of Spiral Tests Made on Natural and Classified Roslyn Coal**

	Type of Feed	
	Natural	Classified
Ash content of feed, pct	21.8	16.8
Ash content of washed coal, pct	14.3	11.7
Ash content of refuse, pct	59.3	60.2
Yield of washed coal, pct	83.3	89.4
Sink 1.60 specific gravity in washed coal, pct	7.7	3.5
Screen analysis of impurity in washed coal, pct		
8 to 14	64.0	12.4
14 to 28	9.0	22.9
28 to 48	2.7	9.6
48 to 100	4.2	11.6
100 to 200	7.5	17.5
Under 200	12.6	26.0

of Arno C. Fieldner, Chief, Fuels and Explosives Division, and Ralph L. Brown, Chief, Coal Branch, Bureau of Mines, and was supported by Drury A. Pifer, Director, School of Mineral Engineering, University of Washington. The authors wish to acknowledge the interest and wholehearted cooperation of Judson S. Hubbard, Vice President, Humphreys Investment Company, and members of his staff.

#### References

- <sup>1</sup>B. M. Bird: The Sizing Action of a Coal-washing Table. U. S. Bur. Mines R.I. 2755 (1926).
- <sup>2</sup>H. F. Yancey and C. B. Porter: Pneumatic Tabling of Coal; Effect of Specific Gravity, Size, and Shape. U. S. Bur. Mines Tech. Paper 536 (1932).
- <sup>3</sup>H. F. Yancey and M. R. Geer: Coal-cleaning Performance—Comparison of Pneumatic Jig, Pneumatic Table, and Baum-type Jig. *Trans. AIME* (1946) **168**, 208-223.
- <sup>4</sup>H. F. Yancey and T. Fraser: Analysis of Performance of Coal Jig. *Trans. AIME* (1925) **71**, 1079-1087.
- <sup>5</sup>H. F. Yancey, M. R. Geer, and R. E. Shinkoskey: Performance of a Pulsator-type Coal-washing Jig. U. S. Bur. Mines R.I. 3372 (1938).
- <sup>6</sup>H. F. Yancey and M. R. Geer: Performance of a Baum-type Coal-washing Jig. U. S. Bur. Mines R.I. 3371 (1938).
- <sup>7</sup>M. R. Geer and H. F. Yancey: Preliminary American Tests of a Cyclone Coal Washer Developed in the Netherlands. *Trans. AIME* (1948) **177**, 220-239; *Coal Tech.* (Feb. 1947) TP 2136.
- <sup>8</sup>W. L. Dennen and V. H. Wilson: Cleaning Anthracite Silt for Boiler Fuel with Humphreys Spiral Concentrator. *Trans. AIME* (1948) **177**, 357-371; *Coal Tech.* (Nov. 1948) TP 2479.
- H. H. Otto, V. H. Wilson, and William L. Dennen: Preparation of Anthracite Silt for Boiler Fuel in a Humphreys Spiral Test Plant. *Trans. Fifth Annual Anthracite Conference of Lehigh University* (1947) 269-291.
- J. D. Clendenin: Evaluating Performance of Anthracite Fine Coal Cleaning Plants. *Trans. Seventh Annual Anthracite Conference of Lehigh University*. (May 1949) TP 2016.
- <sup>9</sup>W. M. Bertholf: Cleaning Table Middling from a Coal Washery with the Humphreys Spiral Concentrator. *Trans. AIME* (1946) **169**, 232-239; *Min. Tech.* (May 1946) TP 2016.

# Petalite—A New Commercial Mineral

by John D. Clark

ONE'S lifetime does not usually offer the opportunity to observe and be a part of the commercial development of an entirely new material. Petalite has been known for over a century, but at first it was presumed rare. However, it was found to be so abundant that it has become a new and extremely useful industrial tool.

## Mineralogy

Petalite is a lithium-aluminum-silicate mineral. It was from petalite that J. A. Arfwedson first discovered lithium in 1818. The theoretical composition of  $\text{Li}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 8\text{SiO}_2$  is equivalent to 78.4 pct silica, 16.7 pct alumina and 4.9 pct lithia. The mineral in its known commercial occurrence is colorless to white and rarely gray, translucent and rarely transparent. It is brittle, has a vitreous luster, and occurs usually in cleavable masses (fig. 1). It is classified as a di-silicate, but is similar in composition to the metasilicate spodumene,  $\text{Li}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 4\text{SiO}_2$  (table I). However, their specific gravities are quite different (3.13 to 3.20 for spodumene, 2.39 to 2.46 for petalite).

## Occurrence

Petalite specimens have been reported in many localities; however, there are only three known countries having occurrences of commercial significance. The classic locality is Sweden's Utö Mines, which, although reportedly still containing considerable petalite, have been closed and are not expected to be operated in the near future. At

Varuträsk, in northern Sweden, petalite carrying 4 pct lithium oxide is the main ore. In 1946 there was reported about 1000 tons on hand. However, the iron content was 0.60 pct too high for other than chemical use.

Petalite has been reported in western Australia. At the Cottesloe mine, where small quantities have been reported available, it is produced in conjunction with feldspar. Petalite occurs in considerable quantity in the feldspar quarry at Londonderry, and it was estimated<sup>1</sup> in 1943 that over 3700 tons might be available.

The petalite in South West Africa was first discovered by F. J. Jooste, a lawyer by profession, in 1938. The following year he discovered a much larger deposit, and in 1948 another important discovery was made. These deposits occur near Kari-bib and constitute the only known commercial deposits. Their full extent has not yet been determined, but huge massive blocks of petalite of the purest grade, running into many thousands of tons, can be seen in all these deposits. To date, 60,000 tons of high grade, low iron containing ore has been blocked out in the major bodies. Mining for years to come will be confined to simple quarrying. Selection from accessory minerals has proved

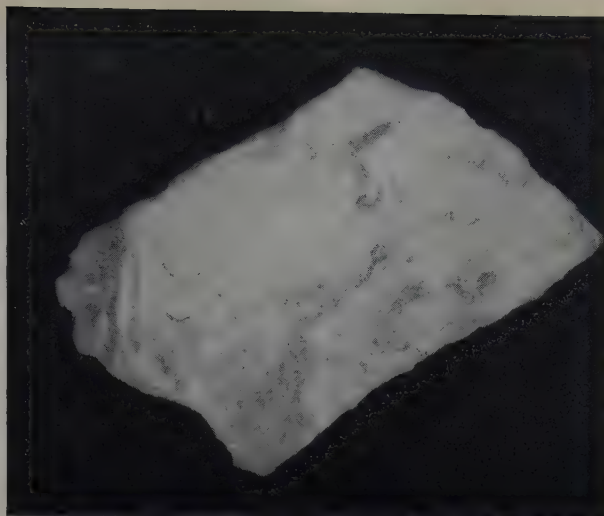
JOHN D. CLARK is Service Engineer, Foote Mineral Co., Philadelphia, Pa.

AIME Columbus Meeting, September 1949.

TP 2924 H. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Oct. 7, 1949.



Fig. 1—Specimen of translucent petalite from South West Africa. The evidence of cleavage shown is not generally so prevalent. Massive petalite is readily mineable free of all accessory minerals.



to be no problem. Transportation to dockside is more than adequate.

The mineral occurs in quartz blows, generally surrounded by granite and pegmatite bodies parallel to, or in line with, the lepidolite bodies, with amblygonite and beryl on a much smaller scale. Maps or charts of the deposits, unfortunately, are not yet available.

It is interesting to note that petalite has the appearance and characteristic mode of occurrence of pegmatitic quartz. It is not surprising, therefore, that for a great number of years many geologists of high repute, mining engineers and inspectors of mines, are reported to have walked over and even sat on these deposits without noticing the existence of this mineral. This is understandable because the cleavage of petalite, called "perfect" in handbooks, has been observed to be poor to fair in specimens examined.

In October 1945 at the AIME meeting at Franklin Institute, Philadelphia, I presented a paper<sup>2</sup> entitled "Lepidolite—Its Occurrences and Uses." Among the world's known lepidolite occurrences, I singled out the deposits of South West Africa with particular detail. I described how the ore bodies dipped with the pegmatites with which they are associated with white pegmatitic quartz. It is believed that this reported pegmatitic quartz may have been wholly petalite.

This paper also stated that "recent word from the mines is that petalite has been described as most abundant." No further details of origin were available at that time. The first shipment of several tons arrived in Philadelphia during the spring of 1945.

#### Research and Development

By early 1947 the advent of jet and rocket engines had stimulated great interest in the discovery and

application of heat-resisting ceramic materials. These were to withstand rapidly repeated heat shock up to and exceeding 1400°C (2550°F). Many trial bodies, which at first seemed promising, failed during endurance tests. However, these same bodies proved quite satisfactory for less technical applications, especially when operating temperatures were at 1300°C (2372°F) and lower, and when the rate of temperature change was measured in minutes instead of seconds. Thus, Floyd Hummel of Pennsylvania State College, who had been interested in the lithia-alumina-silica system, undertook further private investigations of this promising "tangent" early in 1947. Being first interested in the structural relationship between spodumene and cordierite, he extended his scope of interest to petalite. A study of the lithia-alumina-silica system was made by R. A. Hatch.<sup>3</sup>

A paper by R. Roy, D. M. Roy, and E. F. Osborn,<sup>4</sup> broadened the fundamental knowledge of the lithia-alumina-silica system. The molecular formula of petalite is in doubt according to Roy. Evidence indicates that the theoretical ratio of lithia-alumina-silica may be other than 1:1:8 as generally quoted in all handbooks.

The preliminary report of Hummel's<sup>5</sup> development of natural petalite ceramic bodies indicated that natural petalite undergoes a dissociation into beta spodumene and amorphous silica upon heating to 1100°C. This dissociation is irreversible, which is ideal to produce thermal shock-resisting ceramic bodies. Since this report, many investigations have been conducted portending a wide scope of commercial applications.

#### Thermal Expansion of Natural Petalite

Petalite begins to dissociate into beta spodumene and a siliceous glass around 1000°C, resulting in a linear coefficient of expansion of  $19 \times 10^{-7}$  cm per cm

per °C, with movement coming in the early stages of heating. At this point, the mineral is apparently undissociated. Firing to 1100°C for an hour brings about an unusual change in that the expansion is now a straight line type and about the same magnitude as that of fused silica. Heating to 1250°C lowers the linear expansion below that of fused silica.

### Theoretical Considerations

The fact that lithium resembles certain members of the alkaline earth group has been reported on numerous occasions, as has the anomalous behavior of silicate glasses containing the ions  $\text{Li}^+$ ,  $\text{Be}^{++}$ , and  $\text{Mg}^{++}$ . These ions, smaller than ions such as  $\text{Na}^+$ ,  $\text{K}^+$ ,  $\text{Ca}^{++}$ ,  $\text{Sr}^{++}$ ,  $\text{Ba}^{++}$ , and  $\text{Pb}^{++}$ , give rise to "contracted" glasses, that is, glasses that have a higher density than would be expected from theoretical calculations. They also have higher densities than the crystalline phases of the same compositions. As a glass cools from its molten condition, its density apparently depends on the size of the cations is has to accommodate. Evidence of this is indicated by studies of cordierite (a magnesium-aluminum-silicate) and spodumene as well as petalite.

Therefore, in practice, low expansion petalite ceramics are produced in a porous sintered form, taking full advantage of the minimum stress indicated by the combined low expansion of the silica residue and beta spodumene. Such bodies have a relatively long firing range.

To produce dense ceramics of low porosity, as well as expansion, advantage is taken of the admixtures, such as zircon, which act as inert low expansion fillers and provide means by which a maximum of sintered petalite can be retained out of solution. This also tends to lengthen the firing range of such bodies.

### Commercial Applications

Past applications of so-called "heat shock" ceramic bodies were limited to a few conventional types such as alumina, mullite, zircon, and cordierite. Under acute conditions it is necessary on occasion to resort to fused silica articles.

Commercial alumina or corundum bodies have linear coefficient of expansion in the range of 70 to 85 mullite and zircon from 40 to 45, and cordierite from 15 to 30 (all  $\times 10^{-7}$  cm per cm per °C). Fused silica is in the range of 5 to  $8 \times 10^{-7}$ . Porous or dense natural petalite bodies are adaptable to all ceramic forming practices. They can be produced having a low positive thermal expansion equivalent to fused silica, down to having to a low thermal contraction. Within this range, bodies can be formulated having zero expansion from 0 to 700°C. Such bodies require precise firing conditions and composition such that at least partial synthesis is necessary. Interferometer tests are used on all tests to eliminate the margin of error in conventional fused silica apparatus.

Table I Composition of Petalite

	1	2	3	4	5	6	7	8
$\text{SiO}_2$	78.68	76.19	76.16	76.50	76.99	75.30	62.91	51.74
$\text{Al}_2\text{O}_3$	16.62	16.48	17.24	17.50	16.25	17.20	28.42	27.50
$\text{Fe}_2\text{O}_3$	0.09	0.21	0.10	0.18	0.05	0.06	0.53	0.08
$\text{FeO}$	nil	nil	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>
$\text{Mn}_2\text{O}_3$	0.004	trace	trace	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	0.15
$\text{MgO}$	nil	0.54	0.24	<i>b</i>	<i>b</i>	<i>b</i>	0.13	<i>b</i>
$\text{CaO}$	nil	nil	0.21	0.10	<i>b</i>	<i>b</i>	0.11	<i>b</i>
$\text{Li}_2\text{O}$	4.13	3.72	3.62	4.50	4.41	3.94	6.78	4.30
$\text{Na}_2\text{O}$	0.08	0.36	<i>b</i>	0.16	<i>b</i>	<i>b</i>	0.46	0.94
$\text{K}_2\text{O}$	nil	0.18	0.32 <sup>a</sup>	0.39	0.48 <sup>a</sup>	0.20 <sup>a</sup>	0.69	7.40
$\text{Rb}_2\text{O}$	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	2.20
$\text{H}_2\text{O}^+$	0.01	1.04	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>	<i>b</i>
$\text{H}_2\text{O}^-$	nil	1.22	0.24	0.80	0.25	<i>b</i>	0.28	<i>b</i>
F	<i>b</i>	<i>b</i>	<i>b</i>	0.11	<i>b</i>	<i>b</i>	<i>b</i>	7.20

1. Petalite, colorless transparent, Cottesloe, Western Australia by Gov. Lab.
  2. Petalite, colorless milky, Cottesloe, Western Australia by Gov. Lab.
  3. Petalite, colorless milky, Karibib, S.W. Africa by Foote Mineral Co.
  4. Petalite, colorless milky, Karibib, S.W. Africa by Foote Mineral Co.
  5. Petalite, colorless milky, Karibib, S.W. Africa by Booth, Garrett and Blair.
  6. Petalite, colorless milky, Karibib, S.W. Africa by Booth, Garrett and Blair.
  7. Spodumene, Kings Mountain, N. C., by Foote Mineral Co.
  8. Lepidolite, Karibib, S.W. Africa, by Foote Mineral Co.
- <sup>a</sup> All "other alkalis" reported as  $\text{K}_2\text{O}$ .  
<sup>b</sup> Not determined.

The commercial range of applications extends up to 1200°C (2192°F), which is as high as cordierite and higher than the recommended maximum for fused silica. Advantage over translucent fused silica should be evident in the range 900°C to 1200°C, where the glass product is susceptible to more or less rapid devitrification. Thus, petalite ceramics constitute a bright, new tool on the industrial horizon. They have been successfully tested for many uses including pyrometric and combustion tubes and boats, refractory trays, saggars, batts and furniture, burner tips and blocks. They are even adaptable to oven and top-of-stove ware.

The electrical properties of such bodies are equivalent to L-3 type steatite porcelain, which has a notably poor resistance to heat shock. Spark plug insulators are being investigated.

The molecular formula of natural petalite closely approximates the oxide ratio of some ceramic glazes. Simple modification results in glazes suitable for hotel china, chemical and electrical porcelain, and sanitary ware.

Petalite has a lower alumina-lithia weight ratio than spodumene (slightly) or lepidolite, which makes it a preferred lithia source in low alumina-containing porcelain enamels and glasses.

### References

- <sup>1</sup> Private correspondence: Government Geologist, Western Australia.
- <sup>2</sup> J. D. Clark: Lepidolite—Its Occurrences and Uses. *Foote Prints*. (1946) **18** (1) 8-13.
- <sup>3</sup> R. A. Hatch: Phase Equilibrium in the System  $\text{Li}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2$ . *Amer. Mineral.* (1943) **28**, 471-496.
- <sup>4</sup> R. Roy, D. M. Roy, and E. F. Osborn: Composition and Stability Relationships Among the Lithium Aluminosilicates: Eucriptite, Spodumene and Petalite. *Jnl. Amer. Ceramic Soc.* (1950) **33** (5) 152-159.
- <sup>5</sup> F. A. Hummel: Thermal Expansion Properties of Natural Lithia Minerals. *Foote Prints*. (1949) **20** (2) 3-11.



# Tin Deposit of the Montserrat Mine, Bolivia

by Russell Gibson and F. S. Turneure

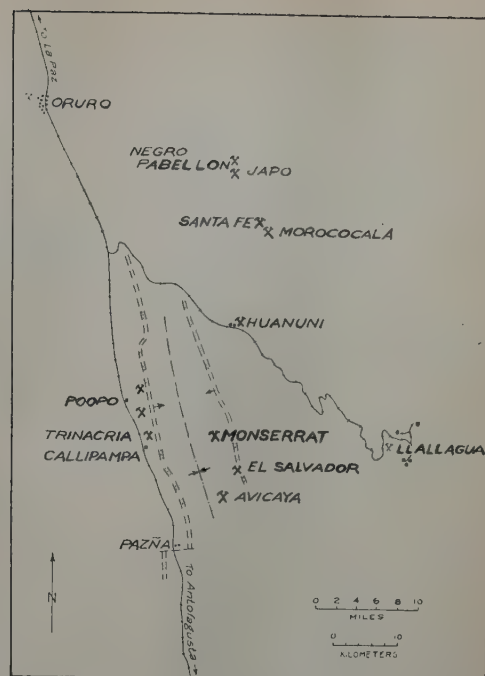


Fig. 1—Location of Monserrat mine.

The tin deposit of Monserrat, Bolivia, consists of one major vein 1600 m in length. The ore is unusual because of the notable quantity of teallite, even though cassiterite is the principal tin mineral. The deposit, formed at shallow depth under a wide range in temperature, may be classed as xenothermal. Polished sections reveal a complex history of replacement, with low-temperature minerals deposited before high-temperature minerals.

THE Monserrat mine<sup>1, 2</sup> of the Compañía Minera Monserrat is located 11 km east of Callipampa, a station on the Antofagasta and Bolivia Railway some 55 km south of Oruro. The mine lies in a group of low hills on the east slope of a broad valley of northerly trend. The valley is separated from the altiplano to the west by a prominent ridge that rises several hundred meters above the surrounding country (fig. 1). Callipampa is situated a short distance west of the ridge, near the east margin of the altiplano. The elevation at Callipampa is 3700 m and at the mine 4100 m.

Monserrat is near the northeast limit of what may be called the Poopo-Pazña district, an area of north-westerly trend about 25 km in length and 15 km in width. The district includes the tin-silver prospects of Poopo to the north, the zinc-tin deposits of Salvador to the southwest, and the tin-tourmaline veins of Avicaya to the south. Along the prominent ridge already mentioned, which forms the western limit of the district, there are several tin and antimony prospects including the Trinacria mine, which is famous for its fine specimens of cylindrite.

Although the principal ore mineral is cassiterite, the tin deposit of Monserrat contains an unusual amount of teallite. Cassiterite occurs in part as a finely granular mixture with sulphides of iron and zinc and in part as needle tin of late hypogene age.

The regional structure is dominated by broad folds typical of the Andes of Central Bolivia and similar to those of the Lallagua district. The area

adjacent to the Monserrat mine is underlain by thin-bedded shale somewhat variable in color but predominantly dark grey to black. The beds strike northwest and dip at low angles southwest, forming part of the northeast limb of a major syncline. A resistant sandstone formation, which underlies the black shale, crops out along a ridge east of the mine and also accounts for the high ridge to the west that was previously mentioned.

No igneous rock of any kind is found at Monserrat. The nearest intrusive bodies known are the irregular dikes and masses of quartz porphyry at Avicaya about 10 km to the south.

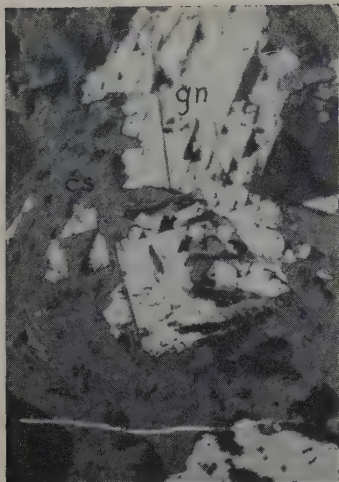
## Mine Workings

The Monserrat property includes one main vein, which can be followed on the surface for 1600 m, and two or three branch veins of distinctly minor importance. The principal haulage tunnel, referred to as the San Carlos adit or level III, extends northeasterly for 1700 m, following the main vein throughout most of this distance. Above the adit, there are

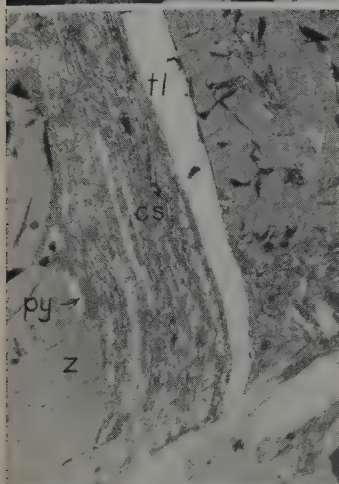
RUSSELL GIBSON, Member AIME, is Associate Professor of Economic Geology, Harvard University, Cambridge, Mass. and F. S. TURNEAURE, Member AIME, is Associate Professor, Department of Geology, University of Michigan, Ann Arbor, Mich.

AIME Columbus Meeting, September 1949.

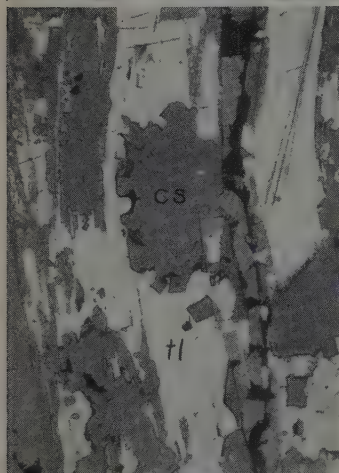
TP 2943 I. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Feb. 16, 1950.



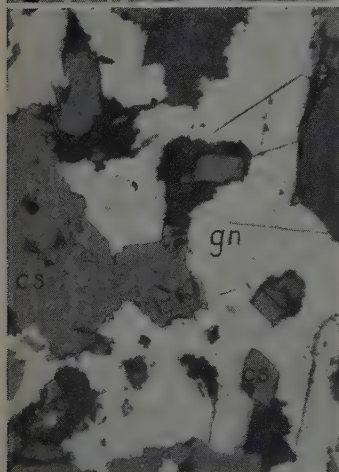
**Fig. 2—Galena (gn) replaced along cleavage by cassiterite (cs). X125.**



**Fig. 3—Bent grain of teallite (tl) replaced along cleavage by pyrite (py) and cassiterite (cs). X125.**



**Fig. 4—Curved teallite (tl) grains partly replaced by cassiterite (cs). X125.**



**Fig. 5—Galena (gn) partly replaced by cassiterite (cs). Shape of cassiterite is, in part, influenced by cleavage of galena. Some grains are sections of cassiterite "needles." X125.**

two principal horizons which are referred to as levels, although the drifts of each show minor differences in elevation. Because of the gentle slope of the surface, the two are considerably shorter than level III.

Several crosscuts have been extended from the drifts, especially in the hanging wall of the main vein. Development in depth has reached level VII, about 130 m below the adit level.

### The Monserrat Vein

In view of the fact that the country rock is a soft thin-bedded shale, the persistence of the Monserrat vein is remarkable. The strike of the vein changes gradually from N 22°E at the south end to N 45°E at the north end and thus cuts across the bedding of the shale. The predominant dip is 75°SE, but in places it reaches 86° and in the northern section flattens to 70°.

The vein lacks a persistent gouge, and it probably does not correspond to a major fault. Rather, it is a complex lode with a considerable variation in detailed structure. In places, the lode consists of a single vein with rather sharp walls and a crudely banded sulphide filling; elsewhere it splits into several closely spaced but independent stringers. To some extent, the branching structure corresponds to low-grade sections of the ore-shoot.

The development of the main vein on level III can be considered typical. In the central part of the ore-shoot, the vein splits into two distinct branches separated by as much as 15 m of shale, the two maintaining their identity for 150 m on the strike. Two or three additional branch veins of distinctly minor importance have been prospected by short drifts, and a number of narrow sulphide stringers have been exposed in the exploration crosscuts.

The vein filling generally consists of massive and crudely banded sulphide more or less frozen to the walls. Locally, an open crustified filling is pronounced, especially where the sulphides give way to quartz. A brecciated structure is evident in one or two places on level III. The filling averages in width from 15 to 25 cm except where the vein splits into a number of stringers. Here the lode structure attains a width of 50 cm or more.

From the surface to level III, a maximum vertical distance of 135 m, the main vein is ore-bearing almost continuously throughout the 1600 m of length, although in the south half of the shoot there are some low-grade areas. At greater depth, the ore shoots become smaller, but the vein continues strong to level VII, 265 m below the outcrop.

Wall-rock alteration along the main veins is scarcely noticeable; as a rule it consists of only a feeble silicification. In a few places, alteration is more intense, and a siliceous shale with finely disseminated sulphide extends a few centimeters from the vein. The fragments of shale within the vein and the narrow bands between stringers are silicified and sericitized to some degree, but no marked change is visible in many exposures.

### Vein Minerals

**General Description:** The description of the Monserrat ores is based upon an examination of levels II to V and on a study of representative specimens from levels II to VII.

The veins of Monserrat are of scientific interest because they contain teallite, an uncommon sulphide of tin and lead, abundant needle tin (acicular crystals of cassiterite), and wurtzite, the less common,



hexagonal form of zinc sulphide. In the tin veins of Carguaicollo,<sup>8</sup> described previously, teallite and wurtzite are abundant, and needle tin is present in small quantity, giving a mineral assemblage strikingly similar to that of Monserrat.

The dominant minerals of the veins are zinc sulphide (wurtzite and sphalerite), pyrite, cassiterite, teallite, and quartz. Less abundant are galena and carbonate. Arsenopyrite, marcasite, and bournonite are widespread but nowhere abundant. Stannite, chalcopyrite, and vivianite are of subordinate importance. Oxidized ore from upper workings collected from old dumps contains goethite, "limonite," jarosite, and kaolin minerals.

Banding is evident in many parts of the vein on all levels examined. The crude bands are commonly 1 to 5 cm wide, but examined closely these in turn are made up of bands from a fraction of a millimeter to 7 mm in width. The banding is emphasized in many places by the crustified filling and by the lines of vugs and druses. The vugs, rarely more than a few centimeters across, are lined by crystals of needle tin, pyrite, quartz, and sphalerite, and more rarely by wurtzite crystals and bladed vivianite. These features emphasize the fact that filling was the dominant method of mineralization at Monserrat.

Most of the ore is fine grained and some of it is exceedingly so, but the texture is erratic. The banding of the ore is due in part to variation in grain size. In some bands, the grains are 0.05 mm or less in greatest dimension, giving the ore a dense and massive appearance. In other adjacent or nearby bands, the grains are much larger, roughly a millimeter or two across; and lining the vugs, the grain size reaches 5 mm.

**Cassiterite:** Cassiterite is found in nearly every polished section examined. It occurs as shapeless grains, as veinlets in other minerals, and as prismatic grains of extremely small diameter (needle tin).

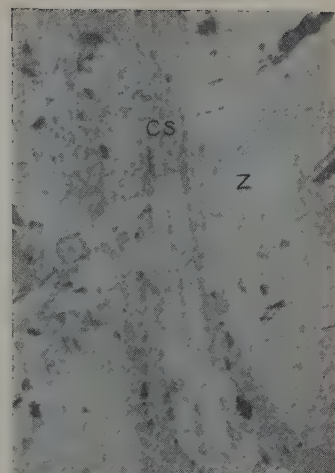
Granular cassiterite forms pseudomorphs after teallite and easily replaces galena, sphalerite, and wurtzite, especially along the cleavages of these minerals (fig. 2). Veinlets of cassiterite and galena cut sphalerite. Where large areas of teallite have been replaced wholesale by zinc sulphide, small curved residual shreds of teallite often remain in the field, and these remnants are often partly replaced by cassiterite (figs. 3 and 4).

Needle tin is widespread. It occurs as scattered, isolated euhedrons and in sheaf-like bundles, the latter especially well shown in association with late quartz in numerous vugs and druses. In hand specimen, the needles are straw-yellow; in thin section, colorless and pale yellow. Some grains are clear; others are turbid. The needles are commonly from 0.01 to 0.05 mm across and 0.20 to 0.50 mm long.

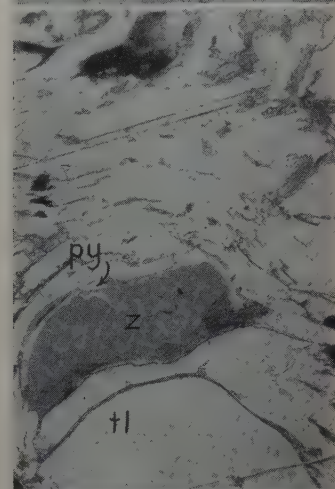
According to A. Hazard, the General Manager, needle tin is especially abundant in the northerly third of the vein above level III and also in the same general area between levels IV and V. But the needle tin is not confined to these places. It has been found in variable amounts from level I to level VII and through a lateral distance of 1300 m or more.

Like the granular cassiterite, needle tin is late in the sequence and replaces teallite, wurtzite, sphalerite, and galena (figs. 5 and 6). Needle tin is present in vugs perched on galena, teallite, and other minerals. Minerals associated with needle tin are the same as those associated with granular cassiterite, and there is no evidence that the needle tin belongs

**Fig. 6—Zinc sulphide, chiefly sphalerite (z) replaced by "needle tin" (cs). X125.**



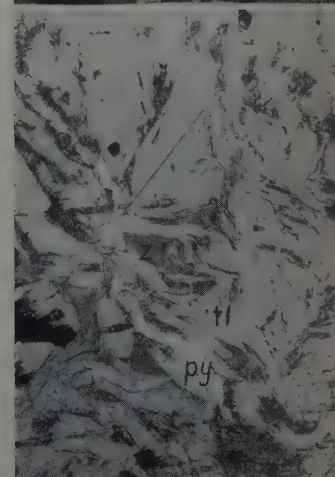
**Fig. 7—Curved grains of teallite (tl) partly replaced by zinc sulphide (z). Later pyrite (py) replaces teallite and follows fracture in zinc sulphide. X125.**



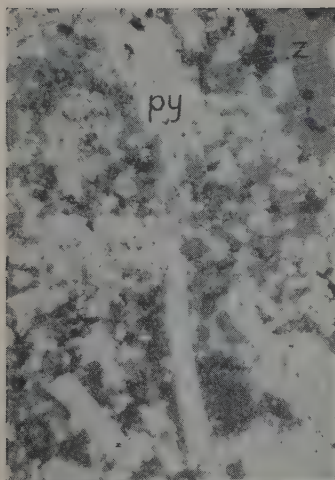
**Fig. 8—Residual teallite (tl) laths in a field of zinc sulphide (z). X240.**



**Fig. 9—Early stage replacement of teallite (tl) by zinc sulphide (z) and pyrite (py). X125.**



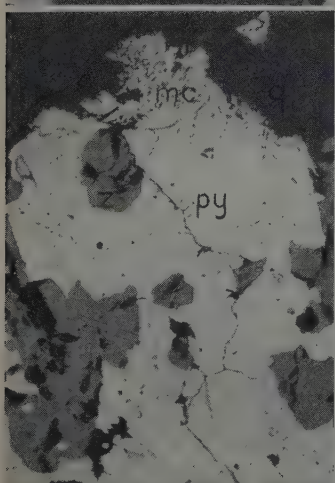




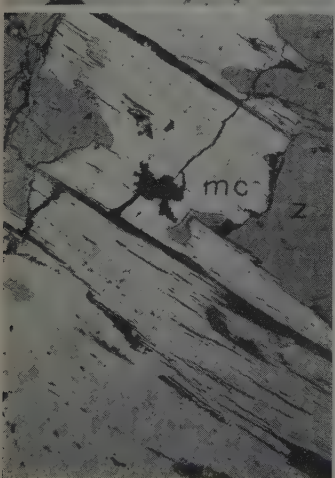
**Fig. 10—Pyrite (py) pseudomorphs after teallite (tl). X125.**



**Fig. 11—Pyrite (py) corroded along rims by marcasite (mc). Nearly every pyrite grain is rimmed by more or less marcasite. X125.**



**Fig. 12—Pyrite (py) partly replaced by marcasite (mc). X125.**



**Fig. 13—Marcasite (mc) possibly pseudomorphous after teallite. X40.**

to a different period of mineralization. There is no zone of supergene tin enrichment. Both the granular cassiterite and needle tin are late hypogene.

Teallite is clearly replaced by needle tin, granular cassiterite, and galena, but in sections showing this replacement there is no indication of supergene deposition of tin or other elements.

**Teallite ( $Pb\ SnS_2$ ):** Teallite is a widespread mineral in the veins but is not everywhere abundant. In hand specimens, it is black, platy and shows good metallic luster. Commonly, grains are 0.05 to 0.10 mm long, but many are only one tenth this length. Rarely, a lath 1 or 2 mm long is seen in coarse-grained ore. Because of its black color and excellent cleavage, fine-grained teallite is difficult to distinguish from fine-grained galena in hand specimen. Because of the softness and flexibility of teallite, it records with special readiness and clarity minor deforming stresses that operated subsequent to its deposition. The elongated teallite grains at Monserrat are commonly curved or bent and with this often goes an opening up of the (bent) cleavage plates; some are deformed only slightly, whereas others are bent through  $90^\circ$  (fig. 3). This deformation is largely prior to and independent of any post-mineral faulting or brecciation of the ore.

Teallite has been replaced chiefly by zinc sulphide, (wurtzite and sphalerite), cassiterite, pyrite, and galena. The deformation of many of the grains, as well as the excellent cleavage, may have facilitated replacement (fig. 4). Some former curved grains of teallite are replaced entirely by cassiterite and galena, the surviving curvature helping to certify the pseudomorphous nature of the alteration (fig. 3). The sphalerite and wurtzite of some polished sections also show curved and distorted grain boundaries, interpreted as a texture inherited from early teallite now entirely removed. If this is correct, then teallite in the early geologic history of the vein was much more abundant than now.

**Sphalerite and Wurtzite:** Sphalerite and wurtzite are the most abundant metalliferous minerals and are among the most widespread. Nearly every section contains one or the other. To determine which is present, immersion tests or thin sections are necessary because the two minerals look alike and are seemingly anisotropic in polished section. Powder X-ray photographs were also used to confirm the identity of some grains. In the following account, "zinc sulphide" is used in referring to instances where the precise identity of the zinc sulphide has not been ascertained. Where its identity has been established, however, by any of the means mentioned, the precise name will be used. In 19 thin sections containing zinc sulphide, the relative percentages of sphalerite and wurtzite are 43 and 57.

Zinc sulphide is the dominant mineral in many parts of the veins and, in some places, gives the appearance of being present in relatively pure bands. Numerous polished sections show, however, that it is almost invariably intergrown with other minerals. Zinc sulphide began depositing early in the sequence, although probably later than teallite, and continued to form throughout most of the mineralizing period.

Crystals of both wurtzite and sphalerite have been identified in vugs. Both are dark brown to black and are indistinguishable merely on the basis of color. Most of the zinc sulphide, however, occurs in cleavable masses and bands as a dominant part of the ore. In thin sections, zinc sulphide is red,



reddish brown or shades of yellow, and grains of any of these colors may show good birefringence. Moreover, one color fades into another. Some of the wurtzite that shows good birefringence in thin section is bladed or radially fibrous. Some of this wurtzite replaces sphalerite. Polished sections show tealite partly or completely replaced by zinc sulphide, pseudomorphously in numerous places (fig. 7). Some of these pseudomorphs are adjacent to laths of tealite replaced by cassiterite. Seemingly there has been wholesale replacement of the tin sulphide mineral by zinc sulphide, and many fields of sphalerite or wurtzite show residual shreds, wisps or blades of tealite (fig. 8). Zinc sulphide is replaced in many sections by galena, cassiterite, pyrite, and marcasite, but there are some reversals in the sequence with pyrite and a very few with the others.

**Pyrite:** Pyrite is present in nearly every specimen examined but is not conspicuous in the ore. About one half the polished sections that contain pyrite show more or less marcasite. A little disseminated pyrite is present in the wall rock near veins. Pyrite commonly occurs in scattered grains or small patches of grains and in tiny veinlets and occurs less commonly as thin bands. Individual grains are usually 0.01 to 0.10 mm and rarely 0.30 to 0.70 mm in diameter. In about one third of the polished sections, grains were large enough to permit positive determination of anisotropism, commonly weak, but occasionally strong. Polarization colors are shades of blue and brown.

Pyrite was deposited throughout most of the vein-forming period, although the greater part of it was deposited later than the abundant minerals, sphalerite, cassiterite, tealite, and quartz. It is found as veinlets along the cleavage of some minerals or cutting across aggregates of grains and as pseudomorphs after tealite (figs. 9 and 10). Residual patches of early pyrite are seen in zinc sulphide clearly not related to any cleavage. Some of the veinlets and residual grains are partly replaced by marcasite (figs. 11 and 12). Pyrite cannot be separated into early and late phases on the basis of size of grain or anisotropism.

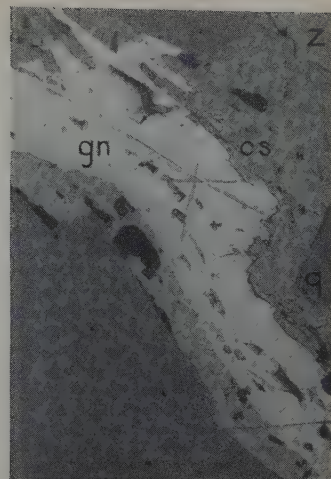
**Marcasite:** Marcasite is of minor importance quantitatively, but about two thirds of the specimens contain at least a little. Most of it was observed only in polished sections. Marcasite occurs commonly as veinlets, 0.005 to 0.01 mm wide, cutting and replacing other minerals and to a less extent as clumps of grains, the individuals of which range up to 0.15 mm in diameter.

Marcasite is later than pyrite with which it is most closely associated. It also veins zinc sulphide, cassiterite, galena, and carbonate and occasionally forms lath-like or lamellar pseudomorphs about 0.50 mm long after tealite (fig. 13). Marcasite is clearly late in the sequence.

**Galena:** Galena is present in about one third of the polished sections examined but is abundant in very few. It is not conspicuous in the hand specimens and would be overlooked in most of them, especially since it is the same color as tealite. All polished sections that show galena also contain the lead-bearing mineral tealite.

Galena occurs in irregular small grains, in veinlets in other minerals, and in pseudomorphs after tealite. Many curved or wisp-like grains originally composed of tealite are now converted wholly to galena or galena and cassiterite, although residuals of tealite remain in some (figs. 14 and 15). Veinlets

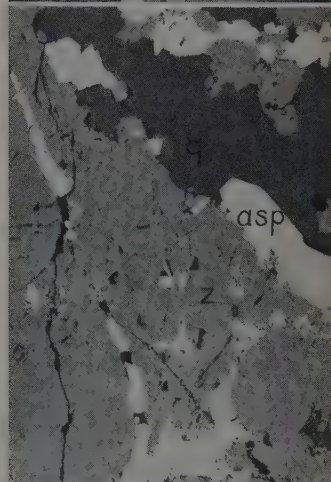
**Fig. 14—Curved tealite completely replaced by galena (gn) and cassiterite (cs). X125.**



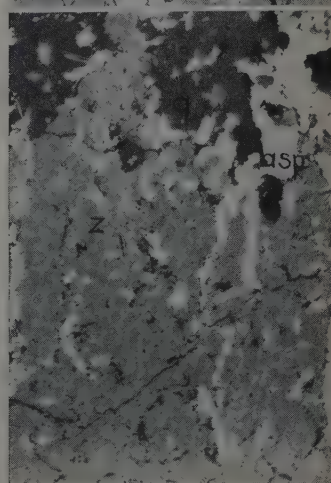
**Fig. 15—Curved tealite completely replaced by galena (gn) and cassiterite (cs). Note early sub-hedral quartz (q). X40.**



**Fig. 16—Curved grains of arsenopyrite (asp) suggesting control by early tealite. Arsenopyrite is corroded by zinc sulphide (z). X40.**



**Fig. 17—Strings of grains of arsenopyrite (asp) in zinc sulphide (z) and quartz (q). Arsenopyrite is partly replaced by zinc sulphide. X125.**





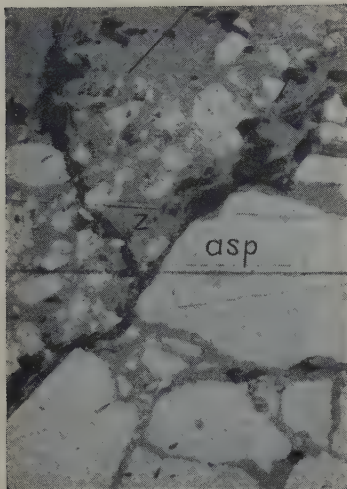


Fig. 18 (left)—Broken arsenopyrite (asp) partly replaced by zinc sulphide (z). X125.

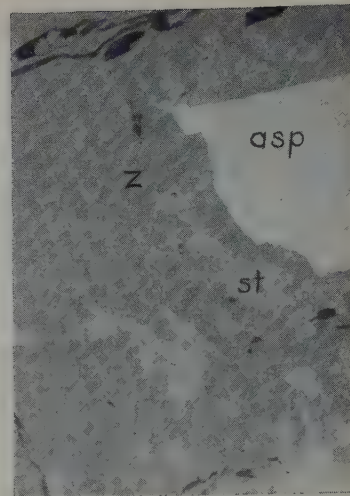


Fig. 19 (right)—Zinc sulphide (z) replaced along cleavage by stannite (st). (Exsolved stannite?) X125.

of galena cut sphalerite and wurtzite along cleavage. Residual grains of teallite in a field of later zinc sulphide are replaced by galena or galena and cassiterite. In some places galena and cassiterite are seemingly contemporaneous; elsewhere, cassiterite is slightly later than galena as shown by veinlets along galena cleavage.

When teallite became unstable in the later hypogene solutions, some grains were converted to galena and cassiterite, but many were replaced almost entirely by cassiterite. The alteration thus involves an introduction of tin by the hydrothermal solutions and does not represent simply the recombination of the tin present in the teallite. At this stage, the solutions may have been of higher intensity so that cassiterite was readily deposited both as a replacement of teallite and as a coating of needle tin, whereas much of the lead sulphide was swept upward to cooler regions.

**Arsenopyrite:** Arsenopyrite, a minor but widespread constituent of the ore, is present in about two thirds of the polished sections, but in only one does it amount to more than 5 pct. The grain size commonly ranges from 0.003 to 0.20 mm and exceedingly few grains are larger than 0.50 mm in diameter.

Arsenopyrite occurs in swarms and strings of small grains, in isolated euhedrons, and to a less extent as veins in other minerals. It is very commonly present at the grain boundaries of quartz with sphalerite or wurtzite (fig. 16) or of pyrite with zinc sulphide. It replaces bent teallite grains and galena that was probably derived from teallite; it also occupies intergranular spaces and forms veins in areas of zinc sulphide (fig. 17). Yet curved plates of arsenopyrite that were probably derived from teallite are corroded by zinc sulphide (fig. 16), and in one area broken arsenopyrite grains are surrounded by zinc sulphide (fig. 18). Although arsenopyrite is commonly later than most of the abundant minerals, it shows some overlap with zinc sulphide.

**Stannite and Chalcopyrite:** A few small grains of stannite, 0.01 to 0.05 mm in diameter, were seen in polished sections. The stannite is almost invariably in sphalerite or wurtzite and may represent exsolved material from the zinc sulphide similar to chalcopyrite in numerous other occurrences of sphalerite (fig. 19). However, Monserrat is notably lean in copper, and only two sections, both cut from the same hand specimen, show any chalcopyrite. In both sections the amount is exceedingly small. The chal-

copyrite occurs as veinlets in bournonite and stannite.

**Bournonite:** Bournonite is present in trivial amounts, usually a small fraction of 1 pct, in nearly one half the polished sections examined. The grains are irregular in shape, except where replacing teallite, galena, or sphalerite along cleavage, and are commonly 0.10 to 0.50 mm in size. They usually show multiple twinning and strong birefringence. Bournonite's place in the sequence is not certain, but it is probably late. It clearly replaces the above-mentioned minerals and is found at the borders of galena, sphalerite, quartz, and siderite. It contains residual wisps of teallite. It is replaced by chalcopyrite.

**Gangue:** The gangue minerals, chiefly quartz and carbonate with small amounts of other minerals, are subordinate to the metallic minerals and are not conspicuous.

**Quartz,** the most abundant and most widespread of the gangue minerals, is present in nearly every thin and polished section. It occurs in many textures and its deposition continued over a long period of time. In one half the polished sections, the quartz appears as somewhat corroded euhedrons that range from 0.02 to 0.50 mm in length. These are surrounded by teallite or by aggregates of later minerals formed largely by the replacement of teallite (fig. 14). In places, the ragged residual grains are almost completely replaced by metallic minerals.

Quartz occurs less commonly as mosaic patches of ordinary vein quartz with wavy extinction. In some of these patches, hundreds of tiny grains of arsenopyrite, pyrite, and sphalerite are present in a field only 2 mm in diameter. Quartz is present also in vugs with needle tin and pyrite crystals and in late veinlets accompanied by marcasite, one of the last minerals to be deposited.

Carbonate is present in nearly one half the thin and polished sections, but invariably in small amounts. It rarely can be seen in large enough grains to be identified in hand specimen. When tested by immersion or microchemistry, several of these proved to be siderite. Under the microscope carbonate is seen in veinlets cutting zinc sulphide, galena, cassiterite and other minerals, and in irregular grains interstitial to quartz, pyrite, and wurtzite. It is present in vugs, also, as a late mineral. In places, carbonate is replaced by late pyrite.

In several specimens a little vivianite was observed in clusters of radiated or divergent blue



crystals in vugs. Commonly, they are perched on siderite, but to a minor extent the siderite is later than the vivianite. The crystals range from 4 to 12 mm in length. A little sericite is present in the vein quartz as well as in the wall rock near the veins.

**Summary of Sequence:** Although some overlap in deposition is indicated by the textures, four principal episodes may be distinguished in the formation of the ore: (1) Early deposition of vein quartz, (2) formation of teallite, (3) widespread replacement of teallite by abundant sphalerite and wurtzite, and by arsenopyrite, galena, cassiterite, pyrite, and marcasite, and (4) continued deposition of late gangue, including both quartz and siderite (fig. 20).

### Origin

A number of features show that the ores of Monserrat were formed at moderate to shallow depth. Most of the ore is fine grained and some of it is exceedingly fine grained. Crustified structure with drusy cavities and fine banding are common. The view that the deposit is of shallow type also receives support in its geological setting, that is, in the narrow but persistent vein enclosed in soft, unaltered shale.

The mineralogy of the ore points to deposition over an unusual range in temperature. Many of the minerals, such as pyrite, sphalerite, galena, chalcopyrite, quartz, and sericite, are stable over a wide range of conditions and give no clue to the temperature at which the deposit was formed. Cassiterite and arsenopyrite, however, normally are associated with high intensity minerals and point to high temperature. According to Gordon Smith's<sup>4</sup> recent experimental work, cassiterite can be crystallized from aqueous solutions of sodium stannate and in the laboratory from room temperature up to at least 450°C. Marcasite and wurtzite, on the other hand, are normally regarded as low-temperature minerals. The stability relations of teallite are not accurately known, but its associations in many deposits in Bolivia indicate that it is probably deposited under medium to low intensity conditions. The presence of all these minerals of extreme temperature range at Monserrat is an expression of telescoping.

In mineralogy, sequence of deposition, and origin, Monserrat offers interesting comparisons with Llallagua<sup>5</sup> and particularly with Carguaicollo.<sup>3</sup>

At Carguaicollo, teallite and franckeite are succeeded by cassiterite. At Llallagua, franckeite is followed by wolframite, pyrrhotite, and arsenopyrite. At Monserrat, teallite is followed by sphalerite and this in turn by arsenopyrite and cassiterite. In all three localities the tin-bearing sulphides are relatively early in the sequence, and these minerals, probably to be regarded as indicative of moderate to low temperature, are followed by minerals commonly regarded as minerals of high temperature significance. The sequence is thus opposed to the conventional view that deposition proceeds on a descending temperature scale. But it supports Graton's<sup>6, 7</sup> hypothesis that, at least during certain periods of the mineralizing process, low-temperature minerals are followed by those of higher temperature. Graton concludes that this must be true for at least the early stages of the mineral-depositing process, and we think that this is confirmed at Carguaicollo and Monserrat. At Llallagua, however, the temperature changes were more complex; early tourmaline, cassiterite and bismuthinite (high temperature) are followed by franckeite (low temperature) and this in turn by wolframite, pyrrhotite and arsenopyrite (high temperature). Sales and Meyer,<sup>8</sup> in their recent paper on the vein formation at Butte, Montana, suggested that mineralization took place there, also, on a rising intensity scale. "The zonal growth of the Butte district to its present dimensions strongly urges the proposition that at a given reference point in the vein system the intensity factor increased, on the average, during the growth of the veins." . . . "We must entertain the possibility of a slow average rise in temperature and chemical potential, two important components of the composite intensity factor, until the end of effective hydrothermal supply." Hart,<sup>9</sup> also pointed out at Butte that deposition may occur on a rising intensity scale. He mentions specifically sphalerite followed by copper-iron minerals.

Lacy<sup>10</sup> noted at one place at Cerro de Pasco that high-temperature minerals followed low-temperature minerals. "Along the solution channelway, heat was introduced faster than it could be dissipated by conduction, so, at one place along this channel, there were rising thermal conditions. Accompanying this rise in thermal gradient, the nature of the iron mineralogy changed from pyrite to pyrrhotite and magnetite, i.e., minerals usually assumed to indicate higher temperature were formed."

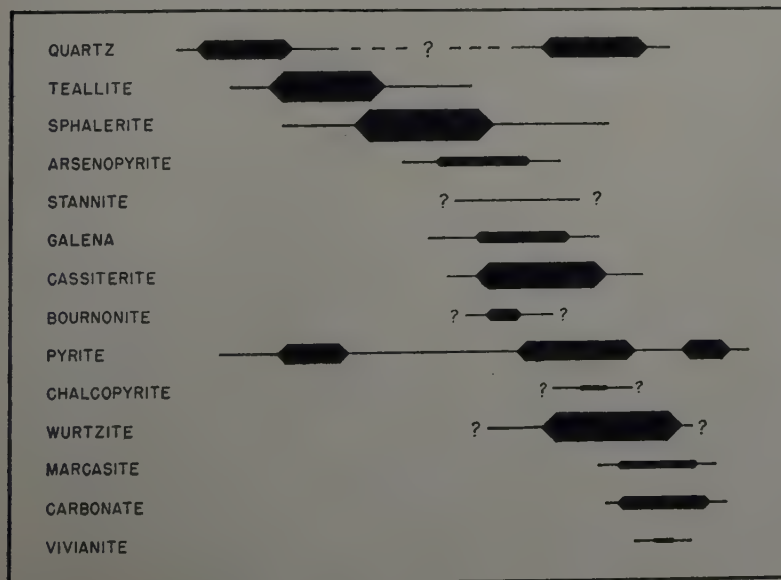


Fig. 20—Graphical representation of mineral sequence.

It is recognized that this argument is based heavily on the influence of temperature as the effective cause of deposition of ore minerals. It is granted that this is only one of several factors that operate to cause deposition. As pointed out in the next paragraph, a change in the alkaline-acid character of the solutions may have been an important contributing cause in the deposition of cassiterite.

The assemblage of minerals at Monserrat is, in the main, similar to that in other districts where an alkaline character of the ore-depositing fluid has been assumed. Smith reports that cassiterite is stable in hot alkaline solutions and can be crystallized from such solutions. He concluded that "in nature, if tin is transported as alkali stannate dissolved in aqueous solutions, a decrease in alkalinity of the solutions may precipitate cassiterite (and) if tin is transported from magmas as alkali thiostannate dissolved in aqueous solutions, a decrease in alkalinity and a decrease in the sulphide ion concentration due to reaction with vein wall rocks would both favor precipitation of cassiterite."<sup>11</sup> The abundance of wurtzite and the widespread development of marcasite at Monserrat, minerals known to crystallize in the laboratory<sup>12</sup> from acid solutions, suggest that the solutions may have decreased in alkalinity as deposition went on. This may have been a contributing cause in the deposition of cassiterite at Monserrat and perhaps at Carguaicollo. Anderson,<sup>13</sup> in his work at the Last Chance and Hornsilver mines in Idaho, notes that wurtzite is a late mineral in his "early assemblage mineral succession" and concludes that the deposition of wurtzite was favored by a decline in the temperature of the solution or by an increase in the acidity.

Decrease in alkalinity may not be the cause of deposition of cassiterite at Llallagua and Oruro. At these places cassiterite was deposited long before the minerals that suggest an approach toward acid solutions, i.e., marcasite, alunite, dickite, and kaolinite. Moreover, the deposition of sulphide in solution can scarcely be a major factor at Llallagua and Oruro, for at these localities the bulk of the sulphides follows the cassiterite.

Chace<sup>14</sup> concluded in his study of the Oruro tin-silver veins that the solutions that deposited those ores were probably alkaline at the beginning of the mineralization period but gradually became neutral, then slightly acid, and finally distinctly acid. His conclusion is strengthened by the fact that kaolinite, dickite, and alunite are present in the Oruro veins. Alunite was reported in the Monserrat ores by Ahlfeld,<sup>15</sup> but none was observed in the present study.

Monserrat, like Carguaicollo and Llallagua, does not fit well into Lindgren's genetic classification of ore deposits. In texture the deposits resemble epithermal ores and were probably formed in a shallow environment at low pressure. They correspond in these respects to epithermal deposits. But the temperature, in the middle period of mineralization at Monserrat and Carguaicollo at least, became abnormally high for this environment, and high-temperature minerals not characteristic of epithermal ores were deposited following low-temperature minerals.

The combination of high temperature and shallow depth would promote rapid deposition accompanied by telescoping. In a nonvolcanic environment such as the shale terrane at Monserrat, heating of the vein

walls by the solutions themselves is of major importance, and this would favor deposition of minerals on an ascending temperature scale. Moreover, the change in the character of the solutions from alkaline to acid, as postulated at Monserrat, is most likely to develop in a shallow environment at low pressure.<sup>16, 17</sup> Thus the complex textures and widespread replacement so characteristic of the ore are readily explained by the rapid shifts in temperature, the low pressure, and the change in the character of the solution from alkaline to acid.

We have used Buddington's term, xenothermal, to classify the great tin deposit at Llallagua, which has an abundance of high and low-temperature minerals. The same term can be extended to Carguaicollo and Monserrat, even though these ores exhibit a milder degree of telescoping than do those of Llallagua.

### Acknowledgments

The writers desire to thank the Compañía Minera Monserrat for permission to publish the results of the investigation. Fernando de las Casas, Peter Joralemon and A. D. Wandke, graduate students at Harvard University, rendered effective assistance in the laboratory. To Professor L. C. Graton we are especially grateful for his helpful assistance and constructive criticism in the preparation of the manuscript. Part of the cost of the research was financed by a grant from the Milton Fund of Harvard University.

### References

- <sup>1</sup> F. Ahlfeld: Die Zinnerzlagertstätte Monserrat, Bolivia. *Ztsch. Prak. Geol.* (1938) **46**, 71-75.
- <sup>2</sup> F. Ahlfeld: Die Bodenschätze Boliviens. 67-68 (1939) Berlin.
- <sup>3</sup> F. S. Turneaure and R. Gibson: Tin Deposits of Carguaicollo, Bolivia. *Amer. Jnl. Sci.* (1945) **243A**, 523-541.
- <sup>4</sup> F. Gordon Smith: Transport and Deposition of the Non-sulphide Vein Minerals. *Econ. Geol.* (1947) **42** (No. 3) 251-264.
- <sup>5</sup> F. S. Turneaure: Tin Deposits of Llallagua, Bolivia. *Econ. Geol.* (1935) **30**, 14-60, 170-190.
- <sup>6</sup> L. C. Graton: Depth Zones in Ore Deposition. *Econ. Geol.* (1933) **28**, 521.
- <sup>7</sup> L. C. Graton and S. I. Bowditch: Alkaline and Acid Solutions in Hypogene Zoning at Cerro de Pasco. *Econ. Geol.* (1936) **31**, 688.
- <sup>8</sup> Reno Sales and Charles Meyer: Results from Preliminary Studies of Vein Formation at Butte, Mont. *Econ. Geol.* (1949) **44**, 465-484.
- <sup>9</sup> L. H. Hart: A Theory of Mineral Sequence in Hypogene Ore Deposits. *Econ. Geol.* (1940) **35**, 1014-1018.
- <sup>10</sup> W. C. Lacy: Doctoral thesis, Harvard University, 1949.
- <sup>11</sup> Pp. 262-263 of ref. 4.
- <sup>12</sup> E. T. Allen, J. L. Crenshaw, and H. E. Merwin: Effect of Temperature and Acidity in the Formation of Marcasite and Wurtzite; a Contribution to the Genesis of Unstable Forms. *Amer. Jnl. Sci.* (1914) **38**, 372-431.
- <sup>13</sup> A. L. Anderson: Epithermal Mineralization at the Last Chance and Hornsilver Mines, Lava Creek District, Butte County, Idaho. *Bull. Geol. Soc. Amer.* (1947) **58**, 451-482.
- <sup>14</sup> F. M. Chace: Tin-silver Veins of Oruro, Bolivia. *Econ. Geol.* (1948) **43**, 467.
- <sup>15</sup> Pp. of ref. 2.
- <sup>16</sup> Pp. 687 of ref. 7.
- <sup>17</sup> A. L. Day and E. T. Allen: Volcanic Activity and Hot Springs of Lassen Peak. (1925) p. 139.



# Some Applications of Millisecond Delay

## Electric Blasting Caps in Mining

by D. M. McFarland

A FEW years ago a novel electric detonator known as the split-second or millisecond delay electric blasting cap was introduced for use in quarry blasting. Regular electric blasting caps fired in series may be depended upon to fire within a millisecond or so from the first to the last in a series. Regular delay electric blasting caps are provided that fire one period after the other period in intervals of  $\frac{1}{2}$  to possibly  $1\frac{1}{2}$  sec. Most split-second or millisecond delays are designed to fire one period after the other period in possibly 25 to 50 millisecond intervals. The ear is not capable of detecting time intervals of this magnitude.

The primary thought at the time millisecond delays were introduced was to investigate the results on rock breakage by firing a line of holes in a quarry face so that charges in adjacent holes would not be detonated simultaneously. This could not be accomplished satisfactorily with regular delays. The time interval between successive periods of  $\frac{1}{2}$  to 1 sec was sufficient to permit considerable movement of the burden. If the burden of one hole was reduced to a great extent by the firing of an adjacent hole, the firing of the hole with the reduced burden would likely reveal this lack of confinement by a terrific report and wild throw of rock.

In the early blasts with millisecond delays it was observed that instead of the usual sharp report, the blast had a muffled sound and vibration was not as perceptible as when simultaneous firing was used. Because many quarry operators were being threatened with injunctions or suits for damages by neighbors who claimed structural damage to their buildings, millisecond delays were tried extensively in quarries. In the majority of these trials, the results were very satisfactory. The seismologists recorded the ground movement created by many blasts and verified the initial observations that millisecond delays could be used to reduce vibrations appreciably. In the past few years the advantages of this principle of nonsimultaneous firing of the charges in blasts has become generally accepted. Today the quarry operator who has vibration troubles, inadequate breakage, and excessive backbreak and has not investigated the possibilities of millisecond delay blasting is ignoring a remedy that has proved satisfactory for many. His complacency may be costing him money.

Because of the results attained in quarry blasting, it was logical that millisecond delays should be tried in construction work such as in road cuts. As formations in this type of work are likely to change

rapidly with advance of the cut, it is more difficult to evaluate results than in quarry blasting. However, this improved control over timing has been beneficial in limiting throw, promoting fragmentation, and reducing overbreak. In blasting near buildings the reduction in vibration and in throw has been especially helpful. As blasters employed in construction work learn what may be accomplished by closer control over the time of firing of explosives charges, more and more millisecond delays are being used to supplant instantaneous electric blasting caps.

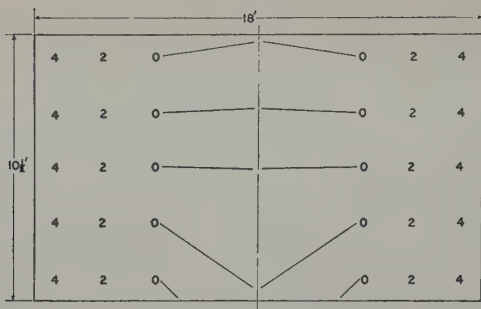
### Improved Fragmentation Underground

With this background of promising results, it was not surprising that millisecond delays should go underground. In limestone mining use of millisecond delays as compared with use of cap and fuse or electric blasting caps showed improved fragmentation in stopes and in slabbing operations. Then an opportunity developed to use millisecond delays in some tunnels being driven in a limestone mine (fig. 1). Using the normal charge employed and merely substituting three millisecond delay periods for three regular delay periods, there was a noticeable difference in the appearance and the position of the pile of rock after a blast. A greater portion of the face was exposed, the crest of the pile was farther from the face, and the pile was heaped high along the center line of the tunnel leaving room to walk along the ribs to the face. Fragmentation was appreciably increased. It gave the impression that the slabs had been thrown against each other with tremendous force, promoting the movement of the broken rock along the center line of the tunnel away from the face. Because the drilling and the charge weights were unchanged, the evidence was convincing that the difference in timing was responsible for the difference in results. Probably a greater portion of the energy from the explosives had been expended in doing useful work on the rock. Zeros followed by two periods of millisecond delays were used in the V cut and in two slabs to either side of the cut in this simple round.

When millisecond delays, substituted period for period for regular delays, are first tried in a drift round in a mine, and the usual charge of explosives

D. M. McFARLAND, Member AIME, is Manager, Technical Division, Atlas Powder Co., Wilmington, Del. AIME New York Meeting, February 1950.

TP 2962 A. Discussion. (2 copies) may be sent to Transactions AIME before Dec. 29, 1950. Manuscript received Jan. 1, 1950.



**Fig. 1—Tunnel round in limestone mine. Numbers indicate millisecond delay period.**

loaded in each hole, the user is likely to receive a shock upon returning to the face to observe results. The usual report is that the round pulled well, but the throw of the rock was excessive, thus creating a pile too shallow and too extended for efficient loading with a mucking machine. To avoid occurrences such as this that displease the miners and result in their opposition to any further trials, it is the practice of many mining companies to limit their experimental work to some unused section of their operations. When they have developed the new procedure to the extent that reasonably consistent and favorable results are obtained, trials in active operations follow.

If there is no opportunity to substitute millisecond delays for regular delays except in actual operations, in drifts as well as in shafts, experience has shown that it is advisable to reduce the usual charge of explosives for the first trial by at least one fourth. This will likely arouse considerable opposition from the miners, but it may avoid excess throw that might damage supply lines and timbers.

### New Control of Timing

Although a review of the literature will give the impression that almost every conceivable type of drift round has been tried, it must be kept in mind that either cap and fuse or regular delay electric blasting caps were used to fire the charges in the holes. Except for pyramid cuts where the holes first to fire met at a common point or a V cut where two holes met at the apex, it was impossible to expect simultaneous firing of the cut holes when primed with cap and fuse. With the introduction of electric blasting caps it was possible to fire cut holes almost simultaneously and thus depend upon concerted action from them. However, after the cut holes were fired with instantaneous electric blasting caps, each of the successive delays that followed were expected to fire individual holes that were designed to break to a newly created free face. This holds true for delay electric blasting caps as well as cap and fuse detonation because the delays of any specific period cannot be depended upon to fire simultaneously. With millisecond delays those of any specific period do not fire simultaneously, but they do fire with much less variation in timing, one from the other, than do regular delay electric blasting caps. To utilize this newly acquired control over timing to the greatest advantage is the problem that concerns us. It is somewhat comparable to the problem which occurred when simultaneous electric blasting caps were first introduced on construction work and blasters started to use them in place of caps and fuse. It is likely that this change, which took place many years before my experience with explosives,

presented many problems at the time. Now blasts numbering several hundred holes fired instantaneously are an accepted practice.

### Control of Throw and Breakage

From experience with electric caps in shooting angles in limestone mines (fig. 2), the principle of accurate timing has been investigated. Assuming that all factors other than timing of the electric blasting caps are held constant, it may be shown that the less the variation in time of firing of the holes, the greater the throw of the angle. However, to obtain this great throw, fragmentation of the angle is decreased. Some systems of primary blasting in limestone mines require throw and tolerate poor fragmentation that must be corrected by secondary blasting.

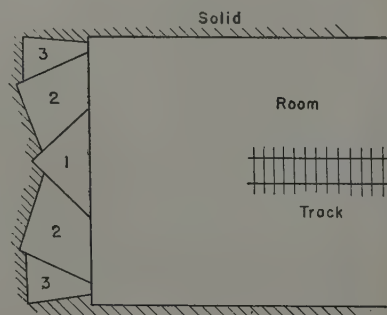
When tunnel men of long experience insist upon using first period delay electric blasting caps in cut holes and not zero delays, or instantaneous electric blasting caps, they are following a procedure that our experience indicates was correct to reduce throw and to promote fragmentation of the cut.

One of our competent technical representatives had the opportunity to substitute millisecond delays in a tunnel where regular delay electric blasting caps were being used. Regular first delays were being placed in the cut holes to reduce throw and thus protect the timbering. When millisecond first delays were used in the four cut holes, a noticeable increase in throw was observed. This can be expected because four millisecond delays of the first period will fire much closer together than four regular delays of the first period. The next trial was with two first period millisecond delays in one pair of cut holes and two second period millisecond delays in the other pair of cut holes. This selection of delays in the cut holes reduced throw appreciably and was used in a number of rounds that followed.

In blasting a number of experimental rounds with millisecond delays, it was found possible to reduce the drilling by several holes per round and to reduce the total explosive charge appreciably. Fragmentation was reported to be improved and overbreak to be decreased. These two factors were responsible for a reduction in the cars required per unit of advance to transport the muck from the tunnel. A very apparent reduction in smoke and fumes after a blast was noticed. Although part of this may be attributed to the reduced charge, it is likely that a greater portion of the energy of the explosives was expended in doing useful work on the rock as indicated by less severe air blast as the rounds were fired.

In a potash mine where the face is undercut, there was an opportunity to study the effect of the selection of millisecond delay periods upon the movement of the burden. Essentially, past practice had been to shoot the line of holes adjacent to the undercut with

**Fig. 2—Angle shooting in limestone mine. Numbers indicate sequence each angle is shot.**





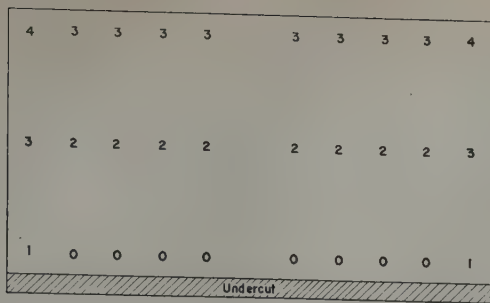


Fig. 3—Former type of round in potash mine, using regular delays.

zero delays with the exception of those at either end, called the rib holes, in which first period delays were used (fig. 3). The second row from the undercut was primed with second delays except for the holes at either end in which third delays were used. The third row from the undercut was primed with third delays with fourth delays at either end. Although millisecond delays had shown marked improvement when substituted for regular delays, the study was made to ascertain if breakage and digging might be improved further.

The preliminary results that were reported indicated that by using a delay pattern to cut slabs diagonally across the face rather than parallel with the undercut, slabs were reduced in size and breakage improved. For example, assume that there are five holes to either side of center in each of the three rows parallel with the undercut (fig. 4). The bottom row starting from center and progressing to the right or to the left would have 0, 1, 2, 3, 4; the second row, 1, 2, 3, 4, 5; and the third row, 2, 3, 4, 5, 6. This arrangement resulted in the center of the face being moved out farthest and provided a loose pile for the loading machines. When a period was skipped using 0, 2, 3, 4, 5 to either side in the bottom row; 2, 3, 4, 5, 6 in the second row; and 3, 4, 5, 6, 7 in the top row, movement of the face was less and a tighter pile resulted (fig. 5). This of course was not desirable. However, it gives us a clue on how throw of a cut might be retarded. This problem is far more complicated when millisecond delays are used than when regular delays are used. Although insufficient evidence is at hand to formulate any definite rules, from results to date we can infer that the smaller the time interval between the firing of the cut holes and those first to follow, the greater the throw of the cut is likely to be. Possibly this effect might be influenced to a lesser degree by the holes firing two periods after the firing of the first cut holes.

#### Test Procedure

Unfortunately, from this point on we must follow an uncharted course and by trial and error develop the basic information about how increasing or decreasing time intervals between millisecond delay periods in developing a cut in any specific formation affects results.

The logical procedure would be to drill and blast the cut holes alone when primed with zero delays, which are instantaneous electric blasting caps. If the throw is excessive, substitute first period millisecond delays for zeros in the cut holes. If throw needs to be reduced farther, then try zeros in the first pair of cut holes, first period millisecond delays in the second pair of cut holes, and second period millisecond delays in the third pair of cut holes where a six-hole V cut is used. By starting with a pair of

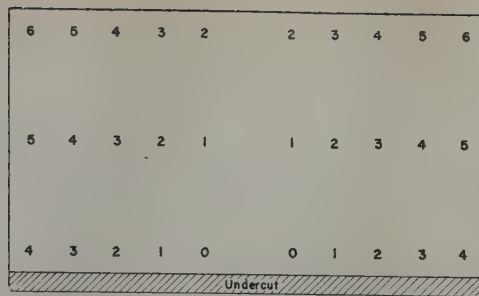


Fig. 4—Millisecond delay round in potash mine resulting in loose muck pile loading.

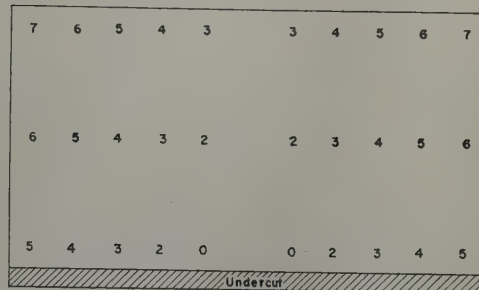


Fig. 5—Millisecond delay round resulting in tight, high muck pile.

first period millisecond delays, then a pair of second period millisecond delays followed by a pair of third period millisecond delays in a six-hole V cut, the use of any of the simultaneous firing zeros will be avoided. Extreme throw should not be expected.

After some satisfactory combination of delay periods has been decided upon for the cut, it might be well to investigate the effect of increasing time intervals when firing the cut holes plus the first holes to follow the cut holes. It is considered advisable to skip a delay period between the cut holes and the first ones to follow the cut. If this combination increases throw appreciably over the cut fired alone, then a longer interval between the cut holes and those first to follow might be tried. When a reasonably satisfactory combination has been found, it should be possible to complete the round by using alternate numbered millisecond delay periods. If for example, a six-hole cut is being primed with two zeros, two first delays, and two second delays, then 4, 6, 8, 10, 12, 14, and 16 delay periods may be used to complete the round. This will give a total of nine delay periods in addition to the zeros. As more information is developed on the use of millisecond delays in drift rounds, it is likely that simplified drill patterns will be developed that will enable the user to minimize the number of delay periods that are required. Some investigators have been exploring these possibilities and have published findings surprising to those well acquainted with theories that have been accepted in the past.

To make any prediction at this time as to the extent that millisecond delays will replace regular delay electric blasting caps one or two years from now in mining operations is merely a guess. However, the use of any product that at no additional cost will enable mining men to convert a greater portion of the energy from explosives into useful work is likely to increase rapidly. This has taken place in quarries and will likely follow in mines as the rules governing the effective use of fast delay electric blasting caps are developed.

# The Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate

by Maurice Rey, Paul Chataignon, and Victor Formanek

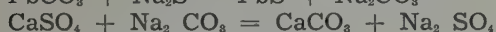
IT is found when floating oxidized lead ores by sulphidization, that the presence of calcium salts in the water, is usually detrimental and lowers the recovery.

This effect is particularly marked in dry countries such as North Africa, where the waters often carry large amounts of calcium sulphate and where the ore may even contain gypsum.

The effect of calcium salts is readily visible. Whereas in their absence cerussite is quickly stained brown and then black by sodium sulphide, in their presence the mineral remains very light in color. A similar effect is produced when barium sulphide is used as a sulphidizing agent instead of sodium sulphide. Magnesium salts have little or no effect and even tend to reduce the detrimental effect of calcium salts.

A study of this phenomenon indicates that it is due to the precipitation of calcium or barium carbonate in contact with the mineral simultaneously with the formation of lead sulphide.

The chemical reactions can be interpreted as:



They might also be written:



The precipitation of calcium carbonate can be followed by the lowering of the pH with which it is accompanied. Magnesium carbonate is more soluble than calcium carbonate and usually does not precipitate under the conditions prevailing.

It is interesting to note that calcium salts have no effect on anglesite (lead sulphate) because calcium sulphate is soluble, but barium salts hinder the sulphidization of anglesite because of the precipitation of barium sulphate.

## Remedies

When calcium sulphate is present in large amounts, the softening of the water with soda ash is usually too expensive to be considered, but the precipitation of the objectionable calcium carbonate can be prevented in two different ways.

One is the use of sodium hydrosulphide instead of sodium sulphide. This salt gives a lower pH than sodium sulphide and does not bring about the immediate precipitation of the calcium which remains in solution as calcium bicarbonate.

The other procedure is to add ammonium salts such as the sulphate or chloride which have the property of increasing markedly the solubility of calcium carbonate. Ammonium salts have other

effects such as cutting down conditioning time and accelerating flotation. They should be added to the flotation cells rather than to the ball mills.

**Table I. Effect of Sodium Hydrosulphide and/or Ammonium Sulphate on the Flotation of Two Oxidized Lead Ores**

Reagents	Mibladen Ore <sup>a</sup>		La Plagne Ore <sup>b</sup>	
	Con- centrate Pb, Pct	Tail- ing Pb, Pct	Con- centrate Pb, Pct	Tail- ing Pb, Pct
Without addition of CaSO <sub>4</sub>				
Na <sub>2</sub> S	56.0	0.68	43.4	0.65
NaSH	56.2	0.60	40.4	0.50
Na <sub>2</sub> S + (NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub> <sup>c</sup>	54.6	0.56		
NaSH + (NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub> <sup>d</sup>			43.0	0.66
With addition of 40 lb/ton CaSO <sub>4</sub>				
Na <sub>2</sub> S	39.4	3.85	34.0	3.21
NaSH	49.1	0.70	49.6	0.88
Na <sub>2</sub> S + (NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub> <sup>c</sup>	53.9	0.85		
NaSH + (NH <sub>4</sub> ) <sub>2</sub> SO <sub>4</sub> <sup>e</sup>	51.0	0.45	41.1	0.78

<sup>a</sup> Mibladen ore, 7 pct Pb, 80 pct of which is oxidized.

Flotation of galena with 0.1 lb per ton amylxanthate.

Flotation of cerussite with 7 lb per ton Na<sub>2</sub>S; 60 pct or the equivalent amount of NaSH. Four additions of 3.0, 2.0, 1.0, 1.0 lb per ton; and 0.5 lb per ton amylxanthate in two stages.

<sup>b</sup> La Plagne ore, 6 pct Pb, mainly oxidized.

Flotation of galena with 0.08 lb per ton amylxanthate.

Flotation of cerussite with 7 lb per ton Na<sub>2</sub>S; and 60 pct or the equivalent amount of NaSH; 2 lb per ton sodium silicate; 0.32 lb per ton amylxanthate.

<sup>c</sup> 12 lb per ton. <sup>d</sup> 4 lb per ton. <sup>e</sup> 8 lb per ton.

In table I are given the results of representative tests on two different oxidized ores. They show that the strongly detrimental effects of calcium sulphate can be offset by the two procedures outlined above.

Sodium hydrosulphide is now used regularly on a mill scale on certain ores.

Tests are being carried out with ammonium salts.

It should be noted that malachite is subject to influences similar to cerussite.

One final word of caution—when the ore is rich in primary slime, which is flocculated by the calcium salts, it may be indispensable to remove these by washing or precipitation with sodium carbonate instead of keeping them in solution by the above methods.

## Acknowledgment

We wish to thank Société Minière & Métallurgique de Penarroya and Minerais et Métaux, for permission to publish these results.

MAURICE REY, Member AIME, is Professor of Non-Ferrous Metallurgy, School of Mines, Paris, France, PAUL CHATAIGNON is Director of Ore-Dressing Laboratory, Minerais et Métaux, Paris, and VICTOR FORMANEK is Research Engineer.

TN 44 B. Manuscript received July 25, 1950.



# The Effect of Mill Speeds on Grinding Costs

by Harlowe Hardinge and R. C. Ferguson

Laboratory and plant data covering 12 different operations show that lower than "standard" ball mill speeds increase grinding efficiency. In the case of high pulp-level mills, the gain is so great that the increase in capital cost of the larger lower speed mill will pay for itself in less than a year's time.

THE object of this paper is to show the economic advantage of operating ball mills at relatively slow speeds. Although many operators know that grinding costs are reduced by operating mills at low speeds, they seldom do so for a variety of reasons. When capital is limited, first cost is of primary importance, and installation of the less costly, high-speed mill is favored. When materials are in short supply, as during a war period, capacity rather than economy is the primary consideration. Existing mills are then speeded up to provide the desired increase in capacity, and new mills are purchased without consideration of maximum grinding efficiency. When the mill speed is increased to secure more production, the increase in grinding costs is often obscured by the total benefit derived from the additional output. This increase in grinding cost is seldom recognized as being sufficient to be worth correcting. For these reasons one may be lead to the erroneous conclusion that slow-speed operation is only occasionally applicable.

The effect of mill speed on grinding efficiency has been studied in laboratory tests.<sup>1,2</sup> The results of some of these tests are given in tables I and II and fig. 1. In these tabulations, batch tests using ore charges of 125 to 200 lb were considered comparable to high pulp-level operation and charges of 50 to 100 lb comparable to low pulp-level operation. For the high pulp-level operation, these tests indicate the existence of a maximum in efficiency

for operation at a speed of 50 pct of critical, both for the soft (dolomite) and hard (chert) ore. In low pulp-level operation, a decrease in speed increases the efficiency of grinding dolomite; no maximum of efficiency appears in the range of speeds tested. The results with chert were inconclusive.

If the performance of large mills is indicated

MILL SPEED VERSUS GRINDING EFFICIENCY

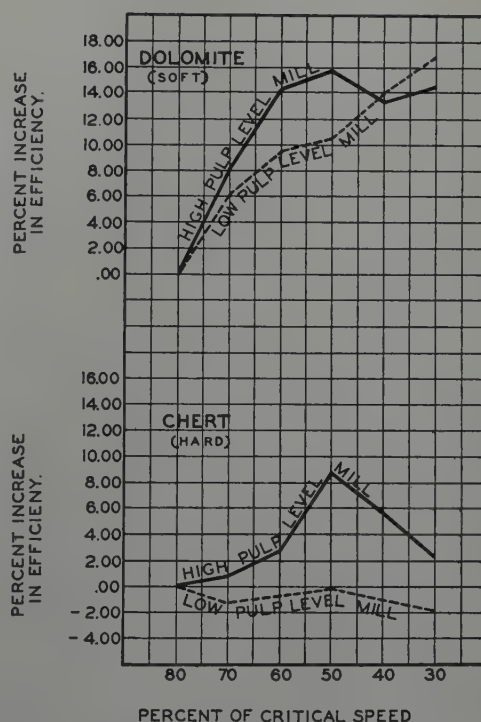


Fig. 1—Data taken from tables I and II, with 80 pct of critical speed as the base.

HARLOWE HARDINGE, Member AIME, is President and General Manager, Hardinge Co., York, Pa. and R. C. FERGUSON, Member AIME, is District Manager, Hardinge Co., Hibbing, Minn.

AIME El Paso Meeting, October 1948 and San Francisco Meeting, February 1949.

TP 2925 B. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Feb. 24, 1950.

**Table I. Speed vs. Efficiency in a High Pulp-level Laboratory Mill<sup>a</sup>**

Speed, Pct Critical	Chert				Dolomite			
	Ore Charge, Lb <sup>b</sup>	Surface Tons per hp-hr		Efficiency Gain, Pct <sup>c</sup>	Ore Charge, Lb <sup>b</sup>	Surface Tons per hp-hr		Efficiency Gain, Pct <sup>c</sup>
		Indiv.	Avg			Indiv.	Avg	
30	200 150 125	22.7 22.4 22.1	22.4	2.75	200 150 125	66.8 68.2 67.2	67.5	14.20
40	200 150 125	23.5 23.2 22.6	23.1	5.95	200 150 125	64.5 67.3 68.5	66.8	13.05
50	200 150 125	24.2 24.1 23.5	23.7	8.72	200 150 125	68.0 69.5 68.0	68.5	15.90
60	200 150 125	22.5 22.5 22.2	22.4	2.75	200 150 125	64.7 68.5 69.0	67.4	14.05
70	200 150 125	21.8 22.3 21.9	22.0	0.92	200 150 125	61.2 64.8 65.4	63.8	7.95
80	200 150 125	21.3 22.1 22.0	21.8	0.00	200 150 125	53.6 61.9 61.7	59.1	0.00

Data from TP 581, U.S. Bur. of Mines.

<sup>a</sup> Batch ball mill, 19 in. ID x 36 in. length; ball charge, 796 lb, 2½ in. maximum ranging down to 1 in.; ball volume, 45 pct; pulp density, 60 pct.

<sup>b</sup> Feed, 1.7 pct +8-mesh, 0.4 pct —200-mesh; product, the —200-mesh fraction ranged from a minimum of 13.1 pct to a maximum of 18.0 pct.

<sup>c</sup> Percentage gain in efficiency referred to the efficiency of the 80 pct critical speed run.

**Table II. Speed vs. Efficiency in a Low Pulp-level Laboratory Mill<sup>a</sup>**

Speed, Pct Critical	Chert				Dolomite			
	Ore Charge, Lb <sup>b</sup>	Surface Tons per hp-hr		Efficiency Loss, Pct <sup>c</sup>	Ore Charge, Lb <sup>b</sup>	Surface Tons per hp-hr		Efficiency Gain, Pct <sup>c</sup>
		Indiv.	Avg			Indiv.	Avg	
30	100 75 50	21.8 21.4 21.4	21.5	1.86	100 75 50	66.9 66.6 66.5	66.7	17.0
40	100 75 50	22.0 21.6 21.4	21.7	0.91	100 75 50	65.8 65.8 63.5	65.0	14.05
50	100 75 50	22.4 21.8 21.5	21.9	0.00	100 75 50	65.6 62.8 60.0	62.8	10.20
60	100 75 50	22.0 21.8 21.6	21.8	0.45	100 75 50	67.3 62.0 58.1	62.5	9.65
70	100 75 50	21.7 21.7 21.7	21.7	0.90	100 75 50	64.1 60.2 57.5	60.6	6.33
80	100 75 50	21.9 21.9 21.9	21.9	0.00	100 75 50	60.1 57.1 53.8	57.0	0.00

For source of data and meaning of <sup>a</sup>, <sup>b</sup>, and <sup>c</sup>, see footnotes to table I.

**Table III—Field Performance of Conical Mills.**

Company Name	Cons. M & S Co. Kimberley, B.C.		A.S. and R. Co. Parral, Mex.		Hudson Bay M & S Co.		Nevada Mines Div. McGill-Kennecott Copper Co.	
Mill size	10x48	10x48	8x36	8x36	10x66	10x66	8x30	8x30
Capacity, tons per day	1,085	1,085	260	260	550	450	408	451
Installed hp	300	300	150	150	400	400	150	119.8
Hp to operate	242	210	155	150	436	295	28	24.5
Mill speed, rpm	19	16.1	23	22.5	21.4	16	100	88
Pct critical	76	64	82.7	80.9	86	63	4 in.	3 in.
Ball load, lb	50,000	46,000	30,000	30,000	76,000	65,000	25,000	25,000
Ball sizes	3 in. C.1	3 in. C.1	3 in. C.1	3 in. C.1	3 in. 3½ in.	3 in. 3½ in.	4 in.	3 in.
Lining con., #/ton	0.04	0.05	0.239	0.231	0.168	0.32 <sup>a</sup>	1.55	1.34
Ball con., #/T	0.85	0.55	3.45	3.44	3.06	2.3	1 in.	1 in.
Feed size	4-mesh	4-mesh	4-mesh	4-mesh	½ in.	½ in.	9	9
Pct plus	17.8	16.7	7.1	7.1	2.4	13.5	9	9
Product pct +48-mesh	2.0	4.6	3.7	3.4				
Product pct —200-mesh	53.5	55.2	60.2	60.9	84.8	90.0	45.7	47.8
Circulating load	5.065	5.915	920	1,200	3,000	4,000	0.038	0.067
Tons —200-mesh/hp-hr	0.077	0.087	0.0336	0.0354	0.039	0.053		
Remarks and Source	From File No. 289C. Also Taggart		Data from AS&R El Paso Office to Hardinge Co., 1937		<sup>a</sup> High wear reduced later by high lifters. Data H.Co. O.R. 533 and 583.		Data from Taggart	
Company Name	Kerr-Addison Gold Mines		Cons. Arizona Smelting Co.		Mufulira Copper Mines		Cananea Con. Copper Co.	
Mill size	10x72	10x72	8x36	8x36	10x72	10x72	8x28	8x28
Capacity, tons per day	697	724	132	140	854	830	300	300
Installed hp	450	450			450	450		
Hp to operate	492	462	114	101	390	385	110	92.6
Mill speed, rpm	21.8	19.8	18	16	19.5	18	25	17
Pct critical	88	79	65	59	78	71	87	59
Ball load, lb	76,000	76,000	28,000	28,000	75,000		16,000	16,000
Ball sizes	3½	3½			4½			
Lining con., #/ton	0.129		0.3	0.3	0.3			
Ball con., #/ton	1.75		2.41	2.41	2.5	2.0	6.3	4.6
Feed size	¾ in.	¾ in.	1 in.	1 in.	¾ in.	¾ in.	C.1	C.1
Pct plus	11	12	28	28	0.4		4-mesh	4-mesh
Product pct +48-mesh							3.4	3.4
Product pct —200-mesh	60.8	64.0	52	52	57	65	26.8	27.8
Circulating load	2,450				3,620	8,000	Open Cir.	Open Cir.
Tons —200-mesh/hp-hr	0.031	0.036	0.023	0.027	0.046	0.049	0.0291	0.036
Remarks and Source	File 432K and H. Co. O.R. 693		H.Co. O.R. 62, 1920		H.Co. O.R. 617 3-10-39 5-16-34		H.C. O.R. 377 5-1-22	



Table IV. Effect of Mill Speed on Grinding Cost at Lake Shore Mines

5x16 Ft Tube Mill Dischg. Arrange- ment	Speed		HP		Ball Wear Lb per Day	Ball Cost Dollars per Day	Days for Liners	Cost of Liners Dollars per Day	Cost of Grates Dollars per Day	Total Cost Dollars per Day	Pet Cost per Day	Grind Cap. Rating, Pct	Cost ÷ Ca- pacity
	RPM	Pct Crit.	Out- put	Cost Dollars per Day									
Trunnion Overflow	30	84.6	182.5	18.25	421	15.55	1,300	0.77		34.57	107.1	123.5	86.8
	27	76.0	164.0	16.40	408	15.10	1,350	0.74		32.24	100.	122.0	82.0
Grate Discharge	30	84.6	204.5	20.45	590	21.80	1,200	0.71	1.50	44.46	138.8	147.0	93.7
	27	76.0	184.0	18.40	572	21.10	1,250	0.68	1.50	41.68	129.2	145.0	89.1

Data from *Trans. C.I.M.M.* (1940) 43, 427.

Tube mills in third stage of fine-grinding circuit. Classifier, overflow: 5 to 8 pct +40 microns (90 to 93 pct —325-mesh). Cast balls: ¾ in., 477 to 514 Brinell hardness, and \$0.037 per lb in 1939. Liners grooved. Feed: sands from secondary tube-mill classifier, 200 to 300 tons per 24 hr (including circulating load), size approximately 5.4 pct +65-mesh. Mill load: 45 pct of mill volume for trunnion-overflow mill and 50 pct for grate-discharge mill.

Table V. Speed vs. Efficiency from Field Performance Given in Table III

Company Name	High Speed		Low Speed		Rate of Change of Efficiency, Tons —200-Mesh per Hp-hr per Pct Critical Speed (6)	Rate of Change of Energy Re- quired, Hp-hr per Ton —200-Mesh per Pct Critical Speed (7)
	Speed Pct Critical	Efficiency, Tons —200- Mesh per Hp-hr (3)	Speed Pct Critical	Efficiency, Tons —200- Mesh per Hp-hr (5)		
(1)	(2)	(3)	(4)	(5)	(6)	(7)
C.M. & S. Co.	76	0.0770	64	0.0870	0.000833	0.1244
A.S. & R. Co.	82.7	0.0336	80.9	0.0354	0.001000	0.8407
Hudson Bay Co.	86	0.0390	63	0.0530	0.000609	0.2945
Nevada Con.	100	0.0380	88	0.0670	0.001417	0.9492
Kerr-Addison	88	0.0310	79	0.0360	0.000555	0.4978
Con. Ariz.	65	0.0230	59	0.0270	0.000667	1.0735
Mufulira	78	0.0460	71	0.0490	0.000428	0.1901
Cananea	87	0.0291	59	0.0360	0.000246	0.2352
Avg	82.84	0.0396			0.000719	0.5257

by the performance of laboratory mills, it may be assumed that for high pulp-level operation efficiency is at a maximum in the vicinity of 50 pct of critical speed irrespective of the hardness of the ore. For low pulp-level operation, increased efficiency of grinding with reduced speed may be assumed for soft ores and an indeterminate change in efficiency for hard ores. The effect of speed on the efficiency of large mills is shown in table III and summarized in table VIII. Table III lists only conical mills because data for these mills were available to the authors. These tabulations also show that circulating loads tend to increase as mill speeds decrease. Table V lists speed vs. efficiency from field performance given in table III. Data on cylindrical mills are not as readily available. It is reported<sup>2</sup> that in the operation of 9x9 cylindrical ball mills at Climax Molybdenum Corp. a speed reduction from 76 to 66 pct of critical was accompanied by an increase in efficiency of between 6 and 7 pct when the mills were operated without grates. When operated with grates, the same speed reduction showed only a 3 pct increase in efficiency. This tends to verify the laboratory results indicated in tables I and II. Gow et al<sup>2</sup> found an increase of from 19.1 to 27.3 surface tons per hp-hr (i.e., a 42.93 pct increase) when the speed of a 6x4 trunnion overflow cylindrical mill grinding dolomite was reduced from 80 to 60 pct of critical. The Lake Shore Mines, table IV, increased their 5x16 tube-mill efficiency 9.03 pct when the speed was decreased from 84.6 to 76 pct of critical.

Table VI. Speed vs. Ball Consumption From Field Performance Given in Table III

Company Name	High Speed		Low Speed		Change in Ball Wear with Change in Critical Speed, Lb per Ton per Pct Critical Speed (6)
	Speed Pct Criti- cal (2)	Ball Wear Lb per Ton (3)	Speed Pct Criti- cal (4)	Ball Wear Lb per Ton (5)	
(1)	(2)	(3)	(4)	(5)	(6)
C.M.&S. Co.	76	0.85	64	0.55	0.0250
A.S. & R. Co.	82.7	3.45	80.9	3.44	0.0056
Hudson Bay Co.	86	3.06	63	2.03	0.0330
Nevada Con.	100	1.55	88	1.34	0.0175
Con. Ariz.	65	2.41	59	2.41	0.0000
Mufulira	78	2.50	71	2.00	0.0714
Cananea	87	6.30	59	4.60	0.0607
Avg			69.27	2.338	0.0304

To evaluate the effect of a reduction in speed upon the overall grinding economy, grinding costs and overall capital costs must be combined. Grinding costs are composed of the expenses for power, balls, liners, maintenance, and labor. Capital costs consist of the costs of the mill, motor, classifier, freight, and auxiliaries.

In the cost calculations shown in table IX, power to operate a mill at different speeds is assumed to be proportional to the rpm within the range of interest. Ball consumption is estimated as follows: the change in ball consumption per unit change

Table VII. Computed Power Requirements and Ball Consumption for Various Pct Critical Speeds

(1) Speed Pct Critical	(2) Tons —200- Mesh per Hp-hr	(3) Pct Increase in Tons —200-Mesh per Hp-hr Below 80 Pct Critical	(4) Pct Decrease in Tons —200- Mesh per Hp-hr Above 50 Pct Critical	(5) Hp-hr per Ton —200- Mesh	(6) Pct Decrease in Hp-hr per Ton, Below 80 Pct Critical	(7) Pct Increase in Hp-hr per Ton Above 50 Pct Critical	(8) Ball Con- sump- tion Lb per Ton	(9) Pct Decrease in Ball Wear Below 80 Pct Critical	(10) Pct Increase in Ball Wear Above 50 Pct Critical
82.84	0.0396			25.25					
80.	0.0416	0.00	34.18	24.04	0.00	51.96	2.664	0.00	52.05
70.	0.0488	17.31	22.78	20.49	14.77	35.84	2.360	11.41	34.70
69.27							2.338		
60.	0.0560	34.61	11.39	17.86	25.71	19.22	2.056	22.82	17.35
50.	0.0632	51.92	0.00	15.82	34.19	0.00	1.752	34.23	0.00

Table VIII. Summary of Mill Speed Data

(1) Mill Size	(2) Range of Speeds Pct Critical	(3) Pct Increase in Effi- ciency Tons per Hp-hr	(4) Avg Pct Increase in Efficiency Per Pct De- crease in Critical Speed	(5) Avg Pct Decrease in Ball Consumption per Pct Decrease in Critical Speed
19x36 in. cyl. mills	80-50	15.90	0.53	
5x16 ft cyl. mills	84.6-76	9.03	1.05	0.359
6x4 ft cyl. mills	80-60	42.93	2.15	
9x9 ft cyl. mills	76-66	6.50	0.65	0.000
Conical mills*	80-50	51.92	1.73	1.141

\* Conical mill data taken from table VII.

Table IX. Comparative Costs at Different Pct Critical Speeds

(1)	(2)	(3)	(4)
1 Mill size	10x10	10x11	10x12
2 Pct critical	80	60	50
3 Speed rpm	20.1	15.1	12.5
4 Ball charge lb	100,000	108,000	117,000
5 Pct of mill volume	41	40	47.5
6 Tph new feed	47.5	47.5	47.5
7 Ton—200-mesh per hp-hr	0.0416	0.0560	0.0632
8 Hp-hr per ton —200-mesh	24.04	17.86	15.82
9 Hp consumed	580	431	384
10 Tph —200-mesh produced	24.128	24.128	24.128
11 Hp installed	600	450	400
12 Ball wear lb per ton	2.664	2.056	1.752
13 Annual grind cost: power, in dollars	41,760.	31,032.	27,648.
14 Annual grind cost: balls, in dollars	59,221.	45,705.	38,947.
15 Grinding costs: power and balls, \$	100,981.	76,737.	66,595.
16 Cost of mill and motor, \$			
17 Mill cost plus 1 yr cost of power and balls	58,000.	63,000.	68,000.
18 Reduction in power and ball cost per year over 80 pct critical speed mill	0.	24,244.	34,386.
19 Saving in ball and power costs, cents per ton over 80 pct critical speed mill	0.	7.08c	10.05c
20 Reduction in ball and power costs per year using 9x9 mill data only	0.	9,109.	12,920.
21 Saving in ball and power costs for 9x9 mill in cents per ton	0.	2.66	3.77

Line 1	Mill sizes determined by hp requirements, line 10, and pct critical speeds, line 2.
Line 6	Determined by the average of 12.2 hp per ton of original feed for the mills shown in table III.
Line 7	From table VII, col. 2.
Line 8	From table VII, col. 5.
Line 9	580 hp assumed for illustration. 431 and 384 hp determined by multiplying line 7 by line 9.
Line 10	Line 9 x line 7.
Line 12	From table VII, col. 8.
Line 13	Based on a 7200 hr year power at \$0.01 per hp-hr.
Line 14	Based on a 7200 hr year forged steel balls at \$0.065 per lb delivered (1948 prices).
Line 20	Derived from table VIII, col. 4, by direct ratio. This assumes ball consumption is also altered proportionately $\frac{0.65}{1.73} \times 24,244 = \$9,109.00$ .

in percentage of critical speed is computed in col. 6, table VI for each of the mills cited and the average of 0.0304 then calculated. By averaging cols. 4 and 5 of table VI, an average ball consumption of 2.338 lb per ton is computed for an average speed of 69.27 pct of critical. This is used as the basis for the calculation of col. 8 of table VII, assuming a linear relation between speed and ball wear with a slope of 0.0304. Liner costs were not evaluated because of the lack of quantitative information.

Table IX shows the comparative costs involved in grinding an ore at an hourly rate of 47.5 tons to produce 24.128 tons of —200-mesh per hr when mills operated at 80, 60, and 50 pct of critical speed are used. The final cost figures are arrived at without consideration of costs of liner wear, maintenance, labor, classifier, and auxiliaries. Table IX indicates a substantial saving in overall cost when low-speed operation is practiced. The gain is still substantial even if the 9x9 mill data are used. This mill, lines 20 and 21, shows the least gain of all mills tabulated except the laboratory unit.

### Conclusions

1. Grinding efficiency increases as mill speed decreases within the range of practical operation.

2. Both power and ball cost per ton of —200-mesh produced decreases with a decrease of mill speed.

3. A slow speed, high pulp-level mill with sufficient additional volume to equal the capacity of an equivalent higher speed mill will make up the difference in capital cost between the two mills in well under a year's operating time through the saving in power and ball cost alone.

4. If an existing high pulp-level mill operating at "normal" speed is replaced by a new, but larger, lower speed mill of the same capacity, the new mill can pay for itself in less than two years' time out of the savings in power and ball cost alone. Only a very moderate resale value for the old mill is assumed in this case.

5. If an existing high pulp-level mill is replaced by a slow speed larger mill, an increase of 20 to 25 pct in capacity is possible without increasing the power consumption or cost of operation. If the decrease in operating cost per ton is also capitalized, this decrease alone will pay for the new mill in less than three years' time.

### References

- <sup>1</sup> TP 581, U. S. Bur. of Mines.
- <sup>2</sup> A. M. Gow, M. Guggenheim, A. B. Campbell, and W. H. Coghill: Ball Milling. *Transactions AIME* (1934) 112, 24.
- <sup>3</sup> Private communication from E. J. Duggan, Mill Superintendent of Climax Molybdenum Corp., March 19, 1948.



# Separation of Precious Metals from Anode Slimes by Flotation

by R. T. Hukki and U. Runolinna

Preliminary separation of precious metals by flotation can offer a simplification of the conventional method of treatment of anode slimes. Laboratory flotation experiments show that rich gold and silver concentrate can be obtained with excellent recoveries.

THE purpose of this paper is to present the results of an investigation into the possibilities of separating precious metals from anode slimes by selective flotation. The work was carried out at the State Research Institute in Helsinki, Finland. The sample of anode slimes tested was submitted by the Outokumpu Co. in Finland.

In 1947 the average assay of filtered and dried anode slimes in the Outokumpu refinery in Pori was as follows: Au, 0.50 pct; Ag, 9.38 pct; Cu, 11.02 pct; Ni, 45.21 pct; Pb, 2.62 pct; Fe, 0.60 pct; Sn, 1.00 pct; Sb, 0.04 pct; As, 0.70 pct; Se, 4.33 pct; S, 2.32 pct;  $SO_4$ , 2.17 pct;  $SiO_2$ , 2.25 pct. The daily assay naturally varies considerably from the figures presented above.

The present conventional method of treatment of anode slimes consists of the steps shown in the flowsheet in fig. 1. As seen from this simplified flowsheet, the present method of treatment involves many steps by which the components of the anode slimes are removed one after another until the main products, gold and silver, may be separated. Due to the small scale of operations, the above procedure is carried out in small batches requiring considerable amount of manual labor.

If, however, the anode slimes could be treated in

R. T. HUKKI, Member AIME, is Professor of Mineral Dressing, Finland Institute of Technology, Helsinki, Finland, and U. RUNOLINNA is Research Engineer, The State Research Institute, Helsinki, Finland.

AIME St. Louis Meeting, February 1951.

TP 2966 B. Discussion (2 copies) may be sent to Transactions AIME before April 1, 1951. Manuscript received Jan. 16, 1950.

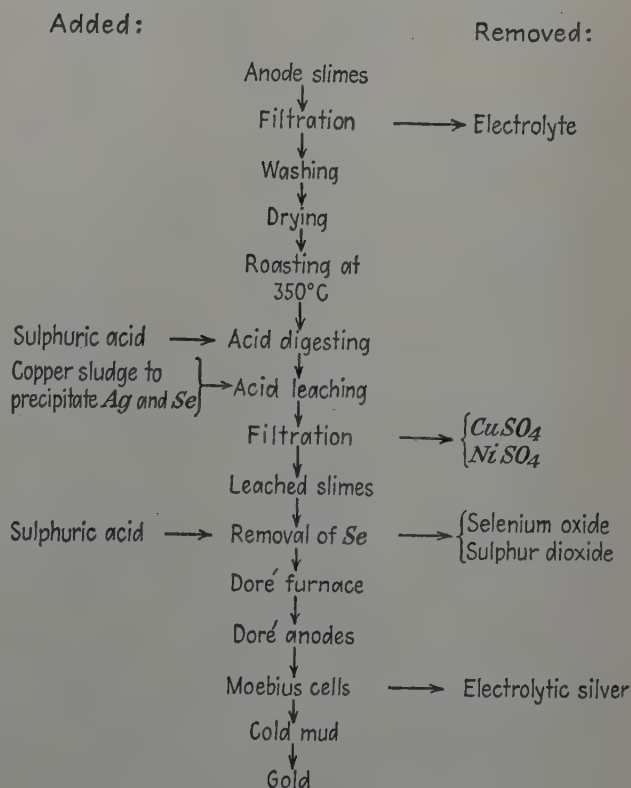


Fig. 1—Flowsheet of conventional method of treatment of anode slimes.

such a way that the precious metals could be separated in the first step in the form of high grade concentrate and with 99 pct or better extraction, then the entire treatment flowsheet could possibly be simplified and the further refinement of the precious metals concentrate reduced to bare essentials.

A "mineralogical" study of the anode slimes suggests that both gold and silver are in the form of selenides. A small amount of gold may exist as telluride. All particles are extremely fine but very strongly flocculated. The color is pitch black. This gold and silver-bearing part amounts to 15 to 25 pct of the slimes. One might say that this fraction corresponds to the sulphide minerals of a sulphide ore.

The second distinct fraction of slimes is the extremely fine, silky and brown colored nickel oxide amounting to 45 to 60 pct and representing oxide minerals of a sulphide ore.

The third fraction is metallic copper, which amounts to 5 to 15 pct and exists in coarser particles. Some of the copper may be in the form of sulphide and/or selenide.

The fourth fraction is the "gangue" of the slimes consisting mainly of quartz particles with some silicates and amounting to 15 to 35 pct.

**Table I. Results of a Flotation Experiment**

	Weight,		Assay, Pct			Distribution		
	G	Pct	Au	Ag	Ni	Au	Ag	Ni
Final concentrate	48	16.7	2.04	40.03	12.10	99.96	99.41	5.1
Recleaner tailing	4	1.4	0.007	0.18	53.78	0.03	0.04	1.9
Cleaner tailing	14	4.9	0.001	0.13	59.04	0.01	0.09	7.3
Final tailing	221	77.0	Trace	0.04	44.34	Trace	0.46	85.7
Head	287	100.0	0.335	6.71	39.81	100.00	100.00	100.0
<b>Reagents Added per 2000 cc Pulp Volume</b>								
			Rougher Flotation		Cleaner Flotation	Recleaner Flotation		
H <sub>2</sub> SO <sub>4</sub> conc. cc			400		200	200		
Reagent 208, mg			200		50	50		
B-23, drops			2		1	1		
Conditioning time, min			15		10	10		
Flotation time, min			3		3	3		

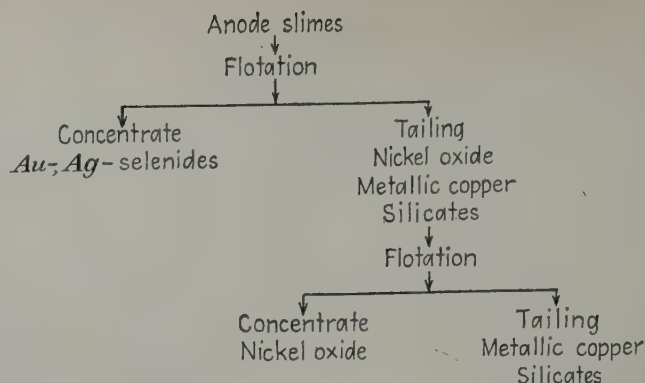
A comparison of anode slimes with a sulphide-oxide-silicate ore suggests similarity of the methods of treatment. The obvious flotation flowsheet is shown in fig. 2.

A number of experiments were run in a Fagergren flotation cell. The following observations were made:

1. A remarkable extraction of precious metals into the flotation concentrate was obtained as indicated in table I. Even after two steps of cleaning, the recovery was well over 99 pct for both gold and silver.

2. The refinery electrolyte is a good flotation medium. It is highly acid containing 200 g per liter free sulphuric acid. Selectivity of operation can be still improved by addition of fresh sulphuric acid. If, however, the primary electrolyte is removed prior to flotation by filtration and washing, the selectivity of separation in a neutral pulp is lost.

In the test shown in table I, substantial amounts of fresh sulphuric acid were used. In continuous practice the amount of acid needed could be reduced. Our experience indicates that a reduction of acid from the given figure of 400 cc per 2000 cc pulp volume to 10 cc resulted in the same grade of final



**Fig. 2—Flowsheet of anode slimes.**

concentrate and in extractions of 99.1 and 99.3 pct for gold and silver, respectively.

3. The best collector found for precious metals so far has been American Cyanamid Company's Reagent 208. All tests with xanthates have been inferior due to the high acidity of the pulp. DuPont B-23 was used as frother.

4. The physical appearance of separation, the ease of its performance, and the sharpness of its end point are most unusual and seldom, if ever, enjoyed in the flotation of common ore minerals.

5. The only disappointment with the flotation concentrate has been its high nickel content. From 5 to 10 pct of the original nickel was carried over with the concentrate. Repeated cleanings reduced the amount of nickel only slightly.

6. The separation of nickel oxide from the tailing of gold and silver float was carried out with conventional reagents for oxide minerals. Before flotation the electrolyte was removed by filtration and washing. As a collector, oleic acid or tall oils may be used. A concentrate assaying 67.4 pct Ni and representing 80 pct extraction of the head was obtained in a preliminary test.

7. The presence in excessive quantities of certain organic chemicals such as glue, oils, etc. commonly used in connection with electrolysis might in adverse cases partly or totally prevent the separation of precious metals by flotation.

The economical value of flotation in the treatment of anode slimes not by any means limited to copper depends on each individual case. If the machinery and satisfactory practice for a conventional method already exist, as is the case with Outokumpu Co., the application of flotation would result in a substantial reduction of the bulk of the material and probably the simplification and/or elimination of some of the steps in the present method of treatment. The financial returns may or may not be of great interest. If, however, a company is thinking of an entirely new electrolytic plant or if the volume of operation and/or existing difficulties warrant major changes, then a preliminary treatment of the slimes by flotation might prove to be also an economical success.

#### Acknowledgment

This investigation was carried out on the initiative of the Outokumpu Co., especially on that of J. Kinnunen, chief chemist of Outokumpu Co. in Pori. All assaying was done in his laboratory. Thanks are also due to Mr. Eero Mäkinen, President of Outokumpu Co., for permission to publish the results obtained.



# Progress Report on Grinding

## at Tennessee Copper Company

by  
J. F. Myers  
and  
F. M. Lewis

**This second progress report of grinding presents comments regarding ball consumption and data pertaining to the hydroscillator, which is closed circuited with the tricone mill. A study and postulate of how balls function is presented. The paper summarizes cost reduction obtained by the method of "grinding for the tail race" rather than for "mesh tons."**

**A**T the Regional meeting in Columbus, Ohio, in September 1949, the authors presented a progress report of the first year's operation with a Hardinge tricone mill in closed circuit with a Dorr hydroscillator. The present report covers our findings on this grinding circuit to January, 1950.

In order to clarify our position, the authors wish to state that no invention or discovery is claimed. The Tennessee grinding circuit is simply an engineered arrangement of known equipment arranged to incorporate all of the recognized efficiency factors of comminution as they are known today.

As we concluded our first report, it was not clear how effective the Tennessee mode of operation would be on harder ores. The only information we had at that time was that the Bond grindability test at 48-mesh gave 6.04 g of undersize per revolution. A later test on the same sample at 100-mesh reduced this to 2.04 g, which is definitely in the harder ore class, and which makes the reported 8.80 kw-hr per ton of —200-mesh produced a very creditable figure.

### Hydroscillator Operation

The machine as now developed has proved dependable and easy to operate. It starts easily under full load and responds quickly to operating adjustments. The hydraulic holes in the oscillating

plate do not plug when shut down under full load. Absolutely clean hydraulic water is essential. An Elliott water screen or similar device should be an essential part of the machine to keep all scale and trash of all kinds from the hydraulic compartment. The shift operators have expressed approval of the machine.

In table I we show a typical screen analysis of the hydroscillator rake sand as compared to conventional classifier rake sand on our ore.

We have previously reported that of the 28.7 pct reduction in power, the hydroscillator contributed 6 pct, on a —200-mesh basis. This is a conservative figure. Some of our test runs indicated that the saving was as high as 10 pct. Laboratory studies by the Dorr Co. engineers indicate that still greater efficiency is possible. Nevertheless, the 10-ft plate is overflowing 2100 tons per day at 3 to 4 pct +65-mesh, which is a very creditable performance by

---

*J. F. MYERS and F. M. LEWIS, Members AIME, are Superintendent of Concentration, and Assistant Mill Superintendent, respectively, Tennessee Copper Co., Copperhill, Tenn.*

*AIME New York Meeting, February 1950.*

*TP 2967 B. Discussion (2 copies) may be sent to Transactions AIME before Dec. 29, 1950. Manuscript received Feb. 10, 1950.*

---

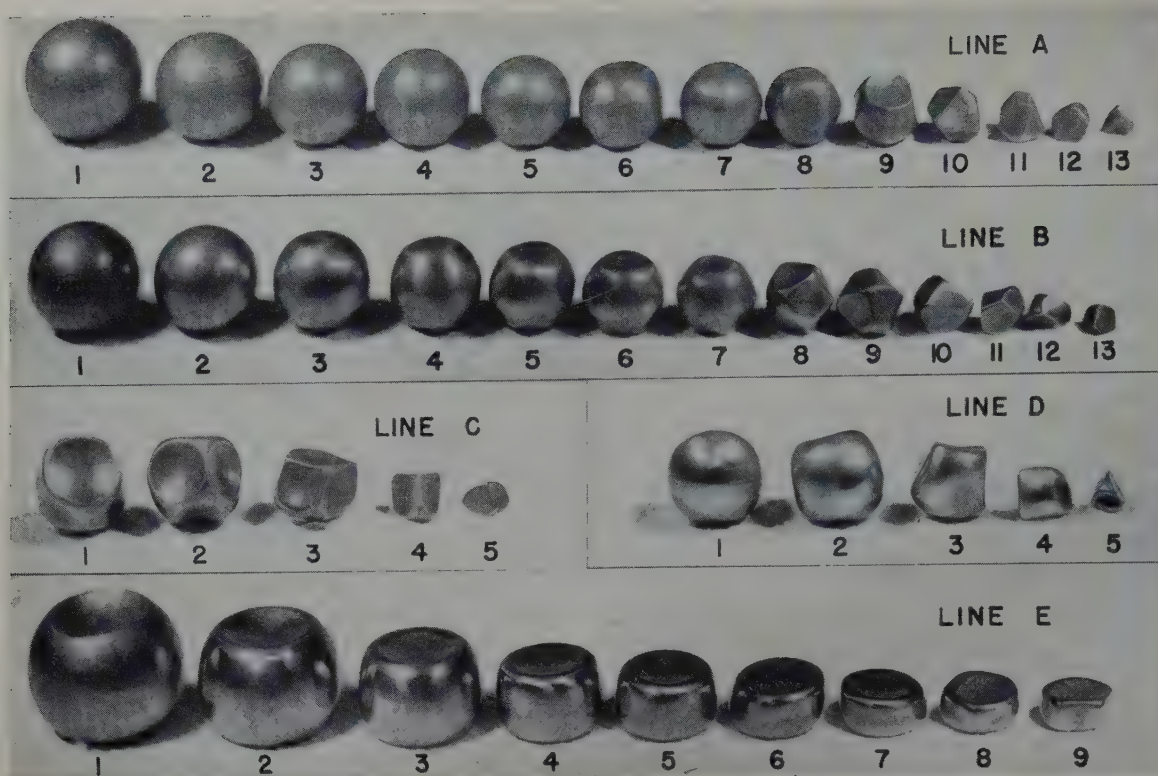


Fig. 1—Ball strings from grinding mills.

any standard. There still remains a great deal of test work to be carried out, and we do not feel that we understand the full possibilities of the machine.

#### Grinding Ball Wear

In our original study of savings to be gained by this unit, we estimated a 27 pct reduction in ball consumption. This was indicated by the slow-speed, ball mill operation at Hayden and other factors. After several months of operation, this saving has not developed, and the ball consumption for the first 16 months of operation stands at 1.06 lb per ton of ore, as compared with 1.05 lb per ton in the conventional mills. In both cases we were using Sheffield Moly-Cop balls, 1 in. diam.

The only explanation that we have at this time is that in the slow-speed tricone we have 34 pct more ball exposure to corrosion than in the small, fast mills. This additional corrosion must offset the 27 pct saving that we should have enjoyed.

We, as many others, have recognized the corrosion factor in all beneficiation apparatus and have ignored it as have others. Very little, if anything, is known about the relative importance of abrasion and corrosion in grinding mills.

We have always thought that the high pH value

of our ball mill discharge (pH 8.5) was due to ferrous hydroxide generated by corrosion of the grinding balls and liners. It is likely that corrosion takes place to a greater degree on a heavy sulphide ore than on one with small sulphide content.

By inquiring, we learned that in some other industries a corrective method in the form of a counter electromotive force has been employed successfully to counteract corrosion. It is known as cathodic protection. We find that there is a 0.26 v drop in potential between our mill pulp and the mill shell. We are actively following up this lead with the help of several friends who have interested themselves in our corrosion problem.

During the past few months we have operated the tricone with a relatively thin discharge dilution, 50 to 60 pct solids. With a conventional speed mill such low dilutions would entail great ball and liner wear. The effective classification in the mill pool keeps a dense bed of pulp in the ball mass under all conditions, as was explained in our first report, and therefore ball wear is apparently not affected by low pulp density.

#### Grinding Ball Action

We considered the ball action of the slow-speed tricone as compared to the action of our conventional fast mills where cascading is prevalent. We came to the conclusion that the action is a combination of two principles. First, round balls rolling against round balls and the shell, as advocated by several authorities. Second, a "mortar and pestle" action of round balls against the concave faces of the smaller balls that have become polyhedrons in the ball paths inside the mass.

In fig. 1 we present a picture of ball strings from

Table I. Typical Screen Analysis of Rake Sands

Mesh	Conventional Classifier	Hydros oscillator
+ 48	23.4	33.6
+ 65	39.0	52.7
+ 100	60.2	75.1
+ 150	79.4	92.7
+ 200	91.0	96.9
--200	9.0	3.1



our grinding mills. String "A" is from the slow tricone mill and string "B" is from the conventional 6 x 12 mill.

The wear action seems to be as follows in the tricone. The balls start out round. At approximately  $\frac{5}{8}$  of an inch the balls start to lose their round shape and begin to form polyhedrons, see ball 6, line A, fig. 1. As such, they cease to roll and remain in a fixed position in relation to the larger rolling balls adjacent to them. This permits the rolling balls to grind a concave surface into the smaller ball, forming a definite mortar for the pestle action of the round rolling balls. This concave face can clearly be seen in ball 9, line A, fig. 1.

The action is identical in the fast mills, but since there is cascading action against the breast liners and great turbulence of the balls in the outer zones, the round balls start to lose their shape much sooner, at approximately  $\frac{13}{16}$  of an inch, see ball 4, line B, fig. 1. The definite concave surfaces do not develop on the polyhedrons until much later in the ball life, see ball 11, line B, fig. 1.

In line C of fig. 1 is shown the characteristic progressive wear, shape and development of the concave face, as the polyhedron gets smaller in the tricone. Ball 1, in line C, shows one single well-defined concave surface in this size ball. Ball 2 has developed two concave faces. Ball 3 clearly shows three concave faces. The last two balls, although very small, clearly show three concave surfaces, indicating that throughout their polyhedron life they have acted as a mortar for revolving round balls.

In line D of fig. 1, a top view of the polyhedron of line C is shown. Note that all five balls have round top surfaces. As one watches the smooth rolling action of the tricone ball mass, the complete lack of cascading balls is obvious and, hence, there is no turbulence in the ball mass. It is, therefore, conceivable that under the first few layers of balls each rolling round ball gets itself a nonrolling polyhedron to work on. Either it or some other round ball stays in attendance of the concave surface as it goes round and round performing the mortar and pestle action. There can be no other explanation to account for the sharp, well-defined edges of the concave surfaces as illustrated in line C, of fig. 1. Were the polyhedrons tumbled around indiscriminately, the edges of the concaves would be rounded off and would not develop.

It is well known that ball slip takes place within the ball mass. The greatest ball movement is next to the shell, and at the center the ball movement is nearly zero. It is, of course, this slip that gives the rolling action to each round ball against its polyhedron. It is also this slip that maintains the rounded top surface of the polyhedron that has developed the concave surfaces on the bottom.

We say top and bottom of the polyhedron, which gives a mental picture of the round ball pushing the polyhedron ahead of it, as is illustrated by fig. 2a. It could just as well be argued that the polyhedron pushes the round ball as is illustrated by fig. 2b. This would account for the rounded surface on the other side of the concaves, just as well. Some few polyhedrons have two concave surfaces which seem to indicate they have a relationship with the round balls as shown in fig. 2c.

This poses a nice question. Of the two forces working in the mill (1) round ball against round

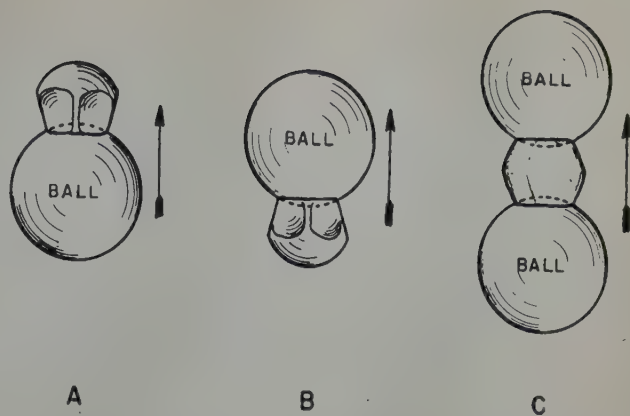


Fig. 2—Diagrammatic relationship of round balls and polyhedrons.

ball and (2) the round ball and polyhedron working as a mortar and pestle, which is the more effective? Can the mortar and pestle action handle as large a particle of ore as two rolling balls? The mortar and pestle, as such, is a sliming device of the first order. Could it be that it is this effect that causes overgrinding in some of our mills today, or is it that larger particles of ore in the concaves prevent the pestle action from overgrinding the fines pocketed therein?

Within the range of ore particle size being ground ( $-14$ -mesh) in the slow tricone mill and conventional 6x12 mill, there is no question of the higher efficiency of the slow tricone. Is it because the balls stay round longer in the tricone or is it because the mortar and pestle action starts sooner on the polyhedron and lasts longer? It would be interesting to know. We thought it might shed some light on the subject to start out the make-up balls with a concave surface.

The Allis Chalmers Co. has for many years manufactured a ball called "concavex." This ball has two concave surfaces, see ball 1, line E, fig. 1. While it has been on the market a number of years, it has never become popular in wet grinding. The mortar and pestle idea was the thought back of making the balls this way.

As of September 27, 1949 we started adding  $1\frac{1}{4}$  in. concavex balls to the tricone. Our idea was that a make-up charge of half 1 in. round balls and half  $1\frac{1}{4}$  in. concavex might contribute to the grinding efficiency. By late January, after four months of observation, it became evident that our grinding efficiency was falling off and the test was dropped. Line E, of fig. 1, clearly indicates the reason for this. The concavex balls quickly start to get flat and thus slide in the ball mass without maintaining concave faces or staying round, and they do some rolling, as will be noted in line E, of fig. 1. They continue to wear down into flats,  $\frac{1}{16}$  in. thick (not shown).

In our opinion, the test did not prove or disprove anything about the mortar and pestle action.

It is with regret that we are unable to contribute some thought on this controversial subject. Cascading and impacting of balls is, of course, essential on coarse ore particles. The only thing that the tricone proves is that cascading and impacting is un-

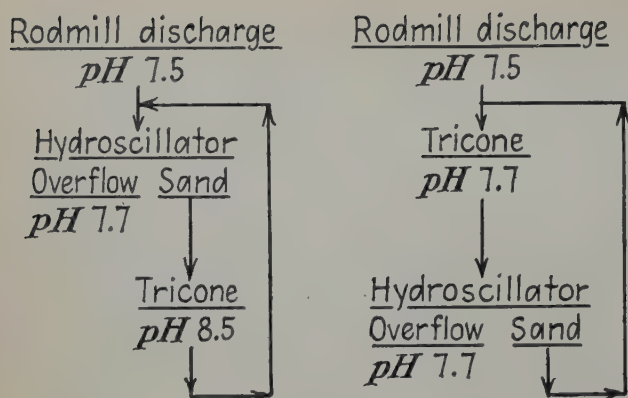


Fig. 3—pH values in grinding circuit.

necessary and inefficient on fine ore particles such as are produced by a fine crushing rod mill.

### Grinding Ball Segregation

It will be recalled that the Hayden operators of the Kennecott Copper Co. reported a mass of small worn balls in the center of the ball mass of their slow speed, 7-ft mills, using 2-in. cast-iron balls<sup>1</sup> which they termed "the kidney." Numerous technicians, including ourselves, have felt that this was to be characteristic of slow speed mills as a whole. The slow-speed tricone, with 1 in., smooth, forged balls does not develop a kidney. Round balls of all sizes down to 5/8 in. are to be found in all parts of the ball mass, and all polyhedrons to the smallest size show the characteristic concave faces and appear in all parts of the ball mass, although to a somewhat less extent on the outside.

### Metallurgical Studies

For the four-month period covered by this report, we abandoned the operation of the tricone and hydroscillator on an efficiency basis for mesh tons, and devoted our efforts to operating the unit to lower the reagent consumption and to produce the best metallurgical results.

From the 2100 tons of daily feed, we float 1300 tons of sulphide mineral as concentrate. Satisfying this sulphide surface with reagents entails considerable cost.

The main feature of reagent saving is to reduce the xanthate consumption in the bulk float. This gives a direct cost saving, of course. When the xanthate-coated particles from the bulk float are transferred to the copper mineral separating cells, a pH of 11.0 is essential. At this pH the xanthate coating on the enormous amount of iron sulphide

surface is removed into the liquor and reacts with copper sulphate in the ensuing zinc separation. Less xanthate in the bulk float promotes a cost saving of copper sulphate. We agree wholeheartedly with L. E. Djingheuzian<sup>2</sup> that "any preparatory machine should be spoken of as a conditioner and if grinding is essential then grinding units become important conditioners."

So far in our investigation, the best grinding efficiency on a —200-mesh basis has been obtained by discharging the open circuit rod mill to the feed well of the hydroscillator, and only the hydroscillator sand feeds the tricone.

However, the best metallurgical results are obtained by feeding the rod mill discharge direct to the tricone, fines and all, and then treating the mill discharge in the hydroscillator. We are not in a position to explain this matter and we believe it is wrong in principle to work for a clean rake sand and then confuse the issue by pouring all of the fines generated by the crusher and rod mill into the tricone. We have long known that the quicker we can grind the ore and get it to the flotation machines, the better are the results, but we likewise know that it does not tell the whole story.

Another factor comes into the picture. In fig. 3 it will be observed that in either mode of operation the feed to the flotation has a pH of 7.7 but that the tricone grinds and conditions the pulp at a pH of 8.5 in one case and at 7.7 in the other case. The higher pH is apparently detrimental, but neither does that explain everything.

### Summary

Reconciling the grinding efficiency with good metallurgy is still a problem. With so many questions still unanswered it is doubtful that we have found the optimum conditions for operating the tricone-hydroscillator circuit. Nevertheless, as of this reporting, we are now enjoying some overall gains. We are equalling the best metallurgical results of the past with the savings shown in table II.

Advantage in maintenance of one large grinding unit over a multiplicity of smaller units is one useful result.

Reduction in flotation time on our three-mineral separation involving cleaners, pumps, and other auxiliary apparatus, indicates a saving of \$0.050 per ton.

### Conclusion

Our experience to date seems to justify the conclusion that the operation of grinding units as a means of preparation and conditioning of ore for subsequent process work is a sound engineering principle.

### References

- W. I. Garms and J. L. Stevens: Ball Wear and Functioning of the Ball Load in a Fine-grinding Ball Mill. *Transactions AIME* (1946) 169, 133; *Mining Technology* (March 1946) TP 1984.
- L. E. Djingheuzian: *Transactions Canadian Institute of Mining Engineering*. (1949) 50, 243-257.

Table II. Cost Reduction

	Old	New	Saving	
	Units	Units	Per Ton	Pct
Power, kw-hr per ton				
Primary and secondary	5.98	4.81	1.17	19.5
Xanthate per Ton	0.38	0.27	0.11	28.9
Copper Sulphate per Ton	0.55	0.39	0.16	29.1
Flotation Time				33.0



# Rheolaveur System of Fine Coal Cleaning

by John Griffen

This paper records over twenty years' experience with the use of the Rheolaveur system in the United States, showing its ability to meet changing conditions caused by the dirtier mine output of present-day mechanical mining methods. Data are given on size ranges handled, number and capacity of units installed, maintenance experience, and operating results on two-product and three-product separations.

IT is not the purpose of this paper to discuss the principles employed in the Rheolaveur system of fine-coal cleaning as these have been fully covered in the technical literature of the Institute.<sup>1, 2</sup> Rather, our purpose will be to record the highlights of over 20 years' experience in the United States with Rheolaveur fine-coal launders, which will indicate their capabilities and costs of cleaning and their adaptability to meet the changing conditions caused by present-day mechanical mining methods.

Rheolaveur fine-coal launders are used in the United States to treat a wide variety of size ranges of fine coal, the coarsest being  $\frac{1}{2}$  in. to 0 and the finest about  $\frac{1}{8}$  in. to 0. In Europe feeds as coarse as  $\frac{5}{8}$  in. and as small as  $\frac{1}{2}$  mm (28-mesh) to 0 are being cleaned. The size ranges usually handled in this country are  $\frac{3}{8}$  to  $\frac{1}{4}$ -in. round to 0.

Thirty five units with a combined hourly feed capacity of 3200 tph have been installed in the United States. Individual units are cleaning as little as 25 tph, while others are cleaning as much as 200 tph. Rheolaveur fine-coal launders offer cleaning units of high capacity and are outstanding in requiring a minimum of building space per ton of input.

JOHN GRIFFEN, Member AIME, is Sales Engineer, The McNally Pittsburg Manufacturing Corp., Pittsburg, Pa.

AIME New York Meeting, February 1950.

TP 2964 F. Discussion (2 copies) may be sent to Transactions AIME before Dec. 29, 1950. Manuscript received Feb. 10, 1950.

Experience has shown that maintenance costs are low. Several installations were made in the Pittsburgh district from 1928 to 1930, and since that time operation has been largely two shifts per day. Minor repairs to liner plates and Rheo boxes have been required during the intervening years, but no major replacement of launders was required until 1947 and 1948. At one of these plants, launders operated for 17 years before they were replaced and during that period over 26,000,000 tons of —5/16-in. coal were cleaned.

The effect of the much dirtier raw coal produced by mechanical loading of the Pittsburgh seam is shown by the following tests. The data in table I summarize the performance of a Rheo fine-coal unit when cleaning hand-loaded raw coal. The —4-in. raw coal fed to the cleaning plant analyzed only 8.16 pct ash and 1.31 pct sulphur and contained 4.6 pct sink, 1.55 sp gr.

Two years later, in 1945, mechanically-loaded coal from the same mine was cleaned in the same plant. The feed, —4 in., then analyzed 20.10 pct ash and 1.53 pct sulphur and contained 20.2 pct sink, 1.55 sp gr. The performance of the Rheo fine-coal unit when cleaning this coal is given in table II.

It will be noted that the +48-mesh coal is cleaned almost as thoroughly as in table I. The large amount of high-ash slimes produced from the dirtier feed are responsible for the higher ash of —48-mesh washed coal in table II.

In this plant the Rheo fine-coal unit did not get the full load of refuse resulting from mechanical

**Table I. Performance of Rheo Fine-Coal Unit When Cleaning Hand-loaded Raw Coal**

Product	Size	Weight, Pct	Ash, Pct	Sulphur, Pct	Sink 1.55, Pct
Feed	5/16 in. +48 Mesh	86.7	8.90	1.42	8.82
	—48 Mesh	13.3	14.21	2.34	
	<b>Total</b>	100.0	9.61	1.54	
Washed Coal	5/16 in. +48 Mesh	79.9	5.45	1.12	1.44
	—48 Mesh	20.1	9.33	1.48	
	<b>Total</b>	100.0	6.23	1.19	
Refuse	<b>Total</b>	100.0	65.86	6.84	90.30

**Table II. Performance of Rheo Fine-Coal Unit When Cleaning Mechanically-loaded Coal**

Product	Size	Weight, Pct	Ash, Pct	Sulphur, Pct	Sink 1.55
Feed	5/16 in. +48 Mesh	86.8	11.45	1.83	10.42
	—48 Mesh	13.2	25.48	2.08	
	<b>Total</b>	100.0	13.30	1.86	
Washed Coal	5/16 in. +48 Mesh	85.0	5.58	1.28	1.78
	—48 Mesh	15.0	17.19	1.66	
	<b>Total</b>	100.0	7.34	1.34	
Refuse	<b>Total</b>	100.0	63.58	3.51	89.80

loading since the Rheolaveur coarse coal unit had already removed over 60 pct of the sink in the —5/16-in. feed coal, and only the balance reached the Rheo fine-coal unit and is reported in table II.

Rheolaveur fine-coal units can readily be operated to produce two coal products of different qualities as well as a refuse. The first grade coal can be of unusual cleanliness for special uses, while the second grade coal product consists largely of middlings for sale as steam coal or for use as mine power plant fuel. In the former case the second grade coal can be made of medium ash content as desired by a customer, while in the second case a higher ash product can usually be utilized.

A recent installation in West Virginia illustrates the possibilities of making a very high grade coking coal as the first grade product, while the second grade coal is a 12.5 to 14.0 pct ash product sold to a nearby electric power station. The coal being cleaned is a ½ in. sq to 0 size from the No. 2 Gas seam, amounting to 120 to 160 tph which is cleaned in two Rheo fine-coal units operating in parallel. Each unit is four launders high, and the overflow products of the top and next lower launders constitute coking coal; the overflow of the third launder is regulating material which is returned to the units as part of the feed, and the overflow of the fourth or bottom launder is the second grade or steam coal. The refuse is discharged by Rheo boxes on the bottom launder.

A float-and-sink test on a typical sample of raw coal is given in table III, and data on the quality of the coking coal, steam coal, and refuse are given in table IV. An examination of the data in these tables shows that the coking coal has an ash content lower than that of the raw coal, floating at 1.35 sp gr. The steam coal undoubtedly contains considerable float at 1.35 sp gr. This is required if the steam coal ash

**Table III. Float-and-Sink Data, ½ in. sq to 0 Rheo Fine-Coal Unit Feed, No. 2 Gas Seam**

Specific Gravity	Weight, Pct	Ash, Pct	Cumulative			
			Float		Sink	
			Weight, Pct	Ash, Pct	Weight, Pct	Ash, Pct
Float 1.35	82.6	5.80	82.6	5.80	17.4	48.50
Sink 1.35; Float 1.40	3.0	13.23	85.6	6.06	14.4	55.85
Sink 1.40; Float 1.45	2.4	19.90	88.0	6.44	12.0	63.04
Sink 1.45; Float 1.50	1.5	24.92	89.5	6.75	10.5	68.48
Sink 1.50; Float 1.60	1.3	30.06	90.8	7.08	9.2	73.91
Sink 1.60	9.2	73.91				
<b>Total</b>	100.0	13.23				

**Table IV. Typical Operating Results with Rheolaveur Fine-Coal Units Cleaning ½ in. to 0, No. 2 Gas Seam Coal**

1948	Ash, Pct		Refuse Pct Weight, 1.45 sp gr
	Coking Coal	Steam Coal	
June	5.27	13.50	6.4
July	5.35	13.75	7.5
August	5.36	13.22	4.1
September	5.41	14.05	4.7
October	5.79	13.85	4.6
November	5.67	13.39	7.3
December	5.55	12.59	6.5
January	5.58	12.58	7.0
February	5.78	13.57	7.2
March	5.10	14.07	5.8
April	4.98	14.05	4.3
Avg	5.44	13.51	5.95
Max	5.79	14.07	7.5
Min	4.98	12.58	4.1

**Table V. Rheo Three-Product Cleaning, 12 to 0.5 mm**

	Weight, Pct	Ash, Pct
Feed	100.0	12.59
Washed Coal	83.3	2.96
Middling	4.9	37.66
Refuse	11.8	70.14

content is to be kept between 12.5 and 14.0 pct because an examination of table III shows that the 1.35 to 1.45 sp gr middlings analyze 16.2 pct ash and the 1.35 to 1.50 sp gr middlings analyze 18.1 pct ash. A typical refuse analyzed 69.55 pct ash.

It is not a usual practice in this country to produce a high-ash middling or second-grade coal for mine power plant fuel, but this is quite common practice in western Europe. I have seen the records of a Rheolaveur washery in the Ruhr, Germany, where middlings of 35 to 40 pct ash were made. The 12 mm (½ in.) to 0 coal was cleaned for a coke plant. Raw coal of this size was first screened at 3.5 mm (⅓ in.) and dedusted at ½ mm (1/50 in.) and about 60 pct of the 3.5 to 0.5 mm was cleaned on dry tables, the balance of this size with the 12 to 3.5 mm, and the middlings from the dry tables were cleaned in a Rheolaveur fine-coal unit. Typical results are given in table V.

This Rheolaveur fine-coal unit was cleaning 140 to 150 metric tph of input.

## References

<sup>1</sup> J. T. Crawford, C. P. Proctor, and J. A. Jounkins: Launder and Table Washing of Fine Coal. *Trans. AIME* (1940) **139**, 269.

<sup>2</sup> Coal Preparation. (1943) New York, AIME, and revised edition (1950).



# Special Methods for the Beneficiation of Glass Sand

by Paul M. Tyler

Higher freight rates and better methods of beneficiation now may make it more economical to open inferior deposits closer to a glass factory than to work higher-grade deposits farther away. Nature of impurities and special treatments as well as common practice of sulphuric acid leaching are described.

**H**ISTORICAL concepts of the economics of the glass-sand industry are changing rapidly. The greatly expanded demand for glass containers combined with higher freight rates on raw materials and manufactured products have induced a migration of glass factories toward densely populated centers and the creation of new standards of place value for sub-standard sand deposits. This migration has been facilitated by the construction of pipelines to bring cheaper natural gas and liquid fuel to large cities and may be further speeded by the adoption of modern mineral dressing methods to permit economical utilization of local raw materials.

Our national resources of naturally high-grade silica sand are abundant, but most of the best deposits are situated so far from large centers of population that it now costs \$3.00 to \$5.00 a ton to carry the sand from the mine to the optimum site for a factory making beverage bottles, scientific and electrical glassware, and miscellaneous blown and pressed ware for local markets. It follows that on the basis of freight savings alone it may be economical to use material from inferior deposits closer to the factory even though the f.o.b. cost of mining and treatment may be much higher than at a higher grade deposit farther away.

Particle-size distribution is a feature of glass-sand specifications. As long as glassmakers insisted upon having their sand principally coarser than 100-mesh, any material was automatically excluded from consideration that could not be purified without fine grinding or other drastic treatment which greatly reduces the size of the quartz grains. However, re-

cent large-scale tests in Norway<sup>1</sup> showed that finely-crushed quartz used in a soda-lime-silica glass batch caused it to melt and become refined appreciably faster than when ordinary coarse (Belgian) sand was employed. Dust losses were negligible. Since the pre-war cost of sand was only 3 pct of the manufacturing cost of the glass, it was deemed economical to pay twice as much for crushed quartz because of the saving in wear and tear on the furnace walls. In the author's unverified opinion, sand ranging in size between 150 and 400-mesh would probably be superior to that in the more usual range of 20 to 100-mesh. Former objections to fine sand (other than dust) were doubtless fostered by the natural concentration of impurities in the finer sizes.

Ordinarily the most objectionable, as well as the commonest, impurity from the standpoint of the manufacturer of any but the cheaper qualities of colored glass is iron. No natural sands are really pure silica. Even water-clear quartz crystals are likely to contain impurities in the form of solid solution as well as inclusions which cannot be eliminated except by chemical treatment that will break down the silica lattice. Pure white pegmatite quartz will usually analyze 0.01 pct or more of iron oxide, and ordinary vein quartz and the quartz grains in most

---

PAUL M. TYLER, Member AIME, is Consulting Mineral Technologist and Economist, Bethesda, Md. AIME New York Meeting, February 1950.

TP 2965 H. Discussion (2 copies) may be sent to Transactions AIME before Dec. 29, 1950. Manuscript received Jan. 4, 1950.

crystalline rocks may be quite impure. Commercial glass sand is almost always won from sedimentary deposits, including unconsolidated beds and lightly-cemented, friable sandstones. Such deposits usually contain grains of other minerals besides quartz, and during and after sedimentation they may be invaded by surface or underground waters, which may precipitate impurities upon the surfaces of the sand grains. It is true that organic acids, such as occur in peaty soils, may reduce ferric iron to form relatively soluble ferrous iron compounds, which are leached out and finally eliminated. As a rule, however, the iron content of a deposit is substantially higher than that of its constituent quartz grains.

### The Mineral Dressing Problem

Given a representative sample of a deposit, the first step in determining its amenability to commercial methods of purification is a petrographic study. Ordinarily, this calls for a screen analysis, each size fraction then being examined by microscopic grain counts, supplemented when necessary by heavy-liquid, high-intensity magnet, and other tests. The object of this preliminary examination is to identify all the minerals and to determine whether the iron occurs principally or significantly in one or more of the following ways:

1. In clay nodules, ferruginous clay bond, or kaolinized feldspar soft enough to be dispersed in water and eliminated by light scrubbing and water washing.
2. As adherent crusts or stains on the surfaces of the quartz grains.
3. As disseminated or penetrated particles in the quartz grains.
4. As discrete particles of limonite, magnetite, ilmenite, pyrite or other heavy iron minerals.
5. As discrete particles of minerals containing iron as an accessory constituent, such as hornblende, garnet, glauconite, or biotite.
6. As an impurity (solid solution?) in discrete particles of muscovite, sericite, rutile or zircon.
7. As adherent iron-oxide crusts or stains which alter (activate) the surface characteristics of heavy mineral grains.
8. Mechanically included in some lighter mineral such as cracked or weathered feldspar in arkosic sands.

Methods for removing soft clay are too well known to require discussion in this paper. They are common to ordinary sand and gravel preparation and correspond to desliming of ore pulps. Mining and treatment methods and costs at a small California plant have been described in detail elsewhere<sup>2</sup> and numerous examples of current practice are covered in various trade journal articles of which only a few are listed.<sup>3, 4, 5</sup> In a mineral dressing laboratory, a simple blunging test is conveniently made in a Fagergren or other type of flotation cell with a 1 : 1 ratio of sand to water. The agitation can be done step-

wise with or without additions of NaOH, Calgon, or other dispersing agents. More drastic treatment is by impact grinding in a pebble mill followed by washing.

Discrete particles of heavy minerals are almost invariably smaller than the average quartz particles and thus are almost ideally prepared for easy removal by gravity separation. Tabling is the time-honored method but the Humphreys spiral and other devices should also be tried. Magnetic separation is sometimes indicated. High-intensity magnetics will ordinarily remove grains that contain as much as 1.5 pct  $\text{Fe}_2\text{O}_3$ .<sup>6</sup> Electrostatic separation, according to the author's experience, is likely to be considered merely as an auxiliary process to separate zircon, rutile, or other commercial minerals from a mixed heavy mineral concentrate, but this situation may change.

The characteristically small size of the heavy mineral particles is also a favorable factor in froth flotation. Flotation likewise may be effective in removing locked, iron-encrusted, or heavily stained particles of quartz and other minerals regardless of their apparent specific gravity. Petrographic examination may indicate whether an iron stain that penetrates deeply into the surface of the sand grains actually covers enough of the surface to activate it for flotation. Anionic collectors (fatty acids) are commonly used for removing iron-bearing impurities, whereas cationic collectors may be more effective for removing feldspar, mica, or sericite. The use of a sulphur-phosphorus derivative of cresol (Aerofloat No. 15) along with pine oil, preferably in a heated pulp, has been patented by Haddan<sup>7</sup> as a method of removing iron impurities. Mica sometimes can be removed in settling boxes or troughs. The Humphreys spiral has been employed experimentally to recover a fine flake mica product from mill tailings<sup>8</sup> but it is not known whether such treatment would scalp off enough mica to clean glass sand properly.

Although the treatment of glass sands by any of the common mineral dressing methods may involve certain minor peculiarities, the general principles thereof are stated in standard textbooks and are quite familiar to all mineral dressing engineers. In this paper, therefore, it has seemed desirable merely to call attention to the importance of possible activation of various nonmetallic mineral surfaces by iron compounds or ions and the complications caused by clay or slime coatings on the proper filming of such surfaces for froth flotation.

Before turning to the consideration of special methods of glass-sand treatment, however, it may be noted that a brief outline of suitable experimental procedures for laboratory tests may be found in a bulletin of the Rutgers University Bureau of Mineral Research.<sup>9</sup> The same reference summarizes available estimates as to typical costs of treatment by specified processes as listed below. These estimates are geared to conditions during and immediately after World War II and, of course, are subject to wide variation according to the material treated, the size of the operation, and the desired quality of product.



**Table I. Approximate Costs of Beneficiation**

Method	Cost per Ton of Sand
Paddle, rake, or screw washers and settling cones	\$0.05 to \$0.15, depending on method and grades desired
Mining and washing unconsolidated sand	About \$0.35
Tabling	\$0.10 to \$0.40
Froth flotation	\$0.10 to \$0.75, depending mainly on cost of reagents
Elutriation	No data
Acid leaching	About \$0.75
Magnetic separation	\$0.10 to \$0.18 (ilmenite removal)
Electrostatic separation	Upkeep low; current requirement may average 4 kw per hr

Approximate costs of beneficiation are given in table I.

**Table II. Preliminary Tests Without a Reducing Agent**

	Pct
25 pct HF stirred 5 min	0.0075 $\text{Fe}_2\text{O}_3$
15 pct HF stirred at intervals over 3½ hr	0.012 $\text{Fe}_2\text{O}_3$
10 pct HF stirred at intervals over 3½ hr	0.012 $\text{Fe}_2\text{O}_3$
5 pct HF undisturbed overnight	0.014 $\text{Fe}_2\text{O}_3$
2 pct HF undisturbed for 48 hr	0.027 $\text{Fe}_2\text{O}_3$

### Chemical Treatments

Acid leaching has been used with technical success with mineral acids alone or in combination with various reducing agents. It has generally been considered expensive, but under the conditions outlined at the beginning of this paper, it may not be too expensive where cheaper methods fail. Although this method may be effective in dissolving small amounts of finely-divided discrete particles of iron and titanium minerals, its principal use is for removing limonite specks and stains. When clay or other impurities are present, preliminary scrubbing and desliming are usually indicated.

Processes using many different combinations of chemicals have been patented and some of these processes have been employed in commercial plants. Among the latter is the Adams process<sup>10</sup> which has been used in England and which was designed to avoid prolonged heating and the many difficulties surrounding the use of strong acids. It employs a water solution containing from 0.25 to 2.0 pct of sodium acid oxalate (tetroxalate) by weight and about one fifth that quantity of ferrous sulphate at a temperature of about 180°F. Sand, which after thorough washing showed an average of 0.07 pct  $\text{Fe}_2\text{O}_3$ , gave an analysis of 0.035 pct after this treatment, and selected sand was reduced in the same plant from about 0.045 to under 0.025 pct  $\text{Fe}_2\text{O}_3$ . Curtin (U. S. Patent 2,198,527, April 23, 1940) discusses countercurrent leaching with hot dilute oxalic acid, the acid being subsequently regenerated by precipitating calcium oxalate from the spent leach liquor and treating with  $\text{H}_2\text{SO}_4$ . The use of  $\text{SO}_2$  is mentioned by Poole,<sup>11</sup> and Gregorjeff and Kaschirina<sup>12</sup> report reduction of the iron oxide content of one sand from 0.16 to 0.01 pct  $\text{Fe}_2\text{O}_3$  and of another sand from 1.5 to 0.25 pct by digestion in a saturated solution of  $\text{SO}_2$  in water at 20 pct solids. The use of zinc hydrosulphite is another possibility

although the writer has not tried it on sand and has not noted references to such use in the literature. Mixtures of  $\text{SnCl}_2$  and  $\text{HCl}$  are reported as being used successfully but would seem to be rather costly.

Sherlock,<sup>13</sup> experimenting with a British sand containing 0.07 pct  $\text{Fe}_2\text{O}_3$ , which remained insoluble in  $\text{HCl}$  except in high concentrations and elevated temperatures, obtained optimum results with  $\text{HF}$  and titanous chloride. The results of preliminary tests without a reducing agent are shown in table II.

The original iron content of the sand used in the above-mentioned tests was 0.045 pct. After water washing this was lowered only to 0.031 pct whereas the desired product was optical glass quality sand with less than 0.01 pct  $\text{Fe}_2\text{O}_3$ . Although the foregoing tabulation indicated considerable action by rather weak solutions, the handling of the acid was still difficult. Accordingly, a search was made for a suitable accelerator. It was found that the addition of a 1 pct solution of titanous chloride to a 1 pct solution of  $\text{HF}$  was effective in removing the coating in 5 min at ordinary temperatures. Titanous sulphate worked equally well but other reducing agents did not. In the plant built to use this patented process (British Patent 555,241, Nov. 7, 1941) the sand remained in contact with the solution for about 10 min. Neutral salts, such as sodium fluoride or sodium silico-fluoride, produced a similar effect to  $\text{HF}$  even in alkaline solution, indicating that the mechanism of the process is first to loosen the clay crust and then to reduce the iron to a soluble salt, presumably a fluo-titanate. No data are given as to consumption of chemicals. Both  $\text{HF}$  and  $\text{TiCl}_2$  are rather expensive, but if they prove more effective than cheaper chemicals on certain sands, they may still be worth considering, especially since the reactions take place at atmospheric temperatures.

Sawyer<sup>14</sup> describes an Ohio plant for treating "glass rock" carrying 4 pct clay and 0.07 pct  $\text{Fe}_2\text{O}_3$  on the surface of the grains. Rock from the quarry was washed and scrubbed with water so as not to break up the sand grains (which are already rather fine); conditioned with  $\text{NaOH}$ , fuel oil, fatty acid, and pine oil; and then treated in flotation cells. After flotation, the partially purified sands are dewatered, dried at 272°F, and repulped. Then  $\text{H}_2\text{SO}_4$  is added at the rate of 25 lb per ton and the mixture run into large wooden tubs where it is heated by steam to 220°F. A higher temperature would decompose the ferrous sulphate. Finally, the sand is washed with water, dewatered, dried, and screened. Sulphuric acid leaching has also been employed in California.<sup>15</sup> Although the cost of the acid is rather low, this process requires acidproof equipment and usually a boiler plant to furnish steam for heating solutions.

### Attrition Scrubbing and Grinding

The attrition scrubber devised by the Bureau of Mines at College Park, Maryland,<sup>6,10</sup> for preparing clean surfaces for froth flotation may be used to rub off the coating on mineral grains without breaking the grains. It is essentially a drastic blunging device with a high-speed rotor and stationary baffles. One model has a vertical shaft carrying radial spokes or blades; another has vertical blades, a design inspired

by the Fagergren flotation cell. The stationary baffles are placed so that the clearance between them and the rotor is only a few times the diameter of the largest grain in the pulp treated. Although the machine may be run on dry sand, it is designed to produce maximum turbulence in a water suspension. The grains in the pulp are forced to collide and rub against one another. While this action provides drastic attrition or "scuffing," the metal surfaces are rubber-covered so as to cushion impact and thus avoid breaking the grains. The rubber, of course, also minimizes wear on the machine due to erosion. Laboratory operating data with this machine on a large number of samples are reported by Dasher and Ralston,<sup>9</sup> and the writer can testify as to successful employment on numerous tests. Unless the stains are exceptionally tough, they can be removed almost completely by an expenditure of 2½ to 10 hp-hr per ton.

A more conventional means of accomplishing the removal of indurated stains and compact crusts is attrition grinding in a pebble mill. Laboratory and pilot plant tests and a design for a 300-ton commercial plant using this method have recently been described by Poole.<sup>11</sup> Essential features of this flow-sheet are preliminary grinding in a 4 x 8-ft rod mill in closed circuit with a 10-mesh screen followed by a magnetic separator and then by a thickener ahead of a 2 ft 8-in. x 7-ft pebble mill operated with ¾-in. pebbles and 60 pct solids. The sand is ground by attrition only in the pebble mill which is in closed circuit with a bowl classifier which discharges a product ranging between 150 and 400-mesh. The raw material is Falls Creek sandstone, a pale buff, soft and friable rock averaging 98.75 pct SiO<sub>2</sub>, 0.12 pct Fe<sub>2</sub>O<sub>3</sub>, 0.67 pct Al<sub>2</sub>O<sub>3</sub>, and 0.10 pct TiO<sub>2</sub>. The iron oxide content of the product was reduced by this treatment to 0.025 pct.

Attrition grinding is defined as a treatment which will reduce the size of sand grains by scouring or rubbing their surfaces but not by cracking or impact grinding of the grains themselves. Since the object is to have the pebbles merely slide over the surfaces of the sand grains, the pebbles cannot be too heavy. According to Poole<sup>11</sup> the top limit of pebble size is about ¾ in. and the optimum ratio of sand to pebble charge is 1.1 : 8. For impact grinding the latter ratio is characteristically much higher, say 3 : 8. Theorists might reason that good results might be obtained by a light pebble charge which would merely stir up the charge and favor self-grinding, but with fine sand the friction between the sand grains alone is likely to be too small to be truly effective.

The cost of the pebble-mill treatment, based on a 300-ton plant and 1943 prices, including 10-year amortization charges, is estimated at \$2.00 a ton.

### Summary

In conclusion, the best method of beneficiating a given glass sand depends essentially upon the nature of the material and, more particularly, upon the mode of occurrence of the iron and other impurities. The simplest treatment is desliming with or without

incidental grinding or scuffing to break up and disperse clay or other semicolloidal iron-bearing material so that they can be washed out in a suitable classifier. When the impurities are present as discrete particles or fine free minerals, standard mineral dressing techniques such as gravity separation or froth flotation are likely to be effective. When the impurities occur largely in the form of a stain or crust on the surfaces of the sand grains, either chemical (leaching) or mechanical (attrition grinding) treatment is indicated. Froth flotation can remove heavily stained grains but only when most of the surface is so covered. Quartz grains containing internal specks or disseminated impurities can only occasionally be removed by gravity or magnetic separation. Iron dispersed or in solid solution in the quartz cannot be removed by any means short of destruction of the crystal lattice.

### References

- <sup>1</sup> F. Gjesmoe: Fine-Grained Sand for Glass Melting. *Journal Soc. Glass Technology* (Oct.-Dec. 1947) **31**, 259-263.
- <sup>2</sup> E. Shaw: Mining and Milling Methods and Costs at the Glass Sand Plant of P. J. Weisel, Inc., Corona, Calif. U. S. Bur. of Mines, I. C. 6937 (1937).
- <sup>3</sup> H. E. Swanson: Washing and Screening for Grade Glass Sand. Oklahoma Silica Sand Co. *Rock Products*. (Dec. 1946) **49**, 78-80.
- <sup>4</sup> E. C. Hoeman and R. C. Redfield: Processing Industrial Sand from River Deposits. *Rock Products*. (June 1947) **50**, 105.
- <sup>5</sup> W. B. Lenhart: Million Dollar Sand and Gravel Plant. *Rock Products*. (July 1948) **51**, 80-84.
- <sup>6</sup> J. Dasher and O. C. Ralston: New Methods of Cleaning Glass Sands. *Bulletin Amer. Ceramic Soc.*, (June 1941) **20**, 187-195.
- <sup>7</sup> British Patent 462,213, April 6, 1936 (Abstr. *Journal Soc. Glass Technology* (1937) **21**, 236A).
- <sup>8</sup> R. Adair, W. T. McDaniel, and W. R. Hudspeth: A New Method for Recovery of Flake Mica from Washing Plant Tailings. R.I. 1, Minerals Research Lab., Asheville, N. C. (Oct. 1949).
- <sup>9</sup> A. S. Wilkerson and J. E. Comeforo: Some New Jersey Glass Sands. Rutgers Univ. Bur. of Minerals Research *Bulletin* No. 1 (1948).
- <sup>10</sup> F. W. Adams: New Process for Removal of Iron from Silica Sands. *Journal Soc. Glass Technology* (1935) **19**, 118-119; Purification of Glassmaking Sands on a Commercial Scale. *Ibid.* (1937) **21**, 409-414.
- <sup>11</sup> J. P. Poole: Glass-Sand Beneficiation Process. *Journal Amer. Ceramic Soc.* (July 1949) **32**, 237-241.
- <sup>12</sup> P. N. Gregorjeff and W. T. Kaschirina: Removal of Iron from Clays and Sands. *Sprechsaal* (1937) **70**, 2. (Abstract *Journal Soc. Glass Technology* (1937) **21**, 84).
- <sup>13</sup> A. Sherlock: Methods for the Purification of Sands for Glassmaking. *Journal Soc. Glass Technology* (1945) **29**, 268-272.
- <sup>14</sup> E. C. Sawyer: Refining of Silica Sand for Glass Manufacture. Ohio State Univ. *Engineering Experimental Station News* (1947) **19**, No. 2, 57-60.
- <sup>15</sup> Brentwood Sand. *Engineering Mining Journal* (Dec. 1937) **138**, 29-31.
- <sup>16</sup> J. E. Norman and O. C. Ralston: Conditioning Surfaces for Froth Flotation. *Transactions AIME* (1939) **134**, 65-80.



# Magnetic Fields Associated With Igneous Pipes In the Central Ozarks

by Charles R. Holmes

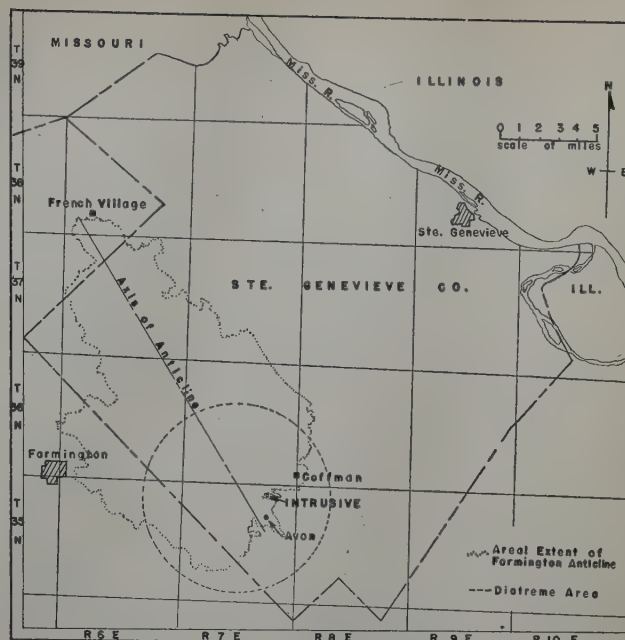


Fig. 1—Location map of intrusive.

MORE than 70 igneous pipes and dikes are known to occur in Cambrian sediments throughout an approximately circular area of about 75 sq miles in southwestern Ste. Genevieve County and southeastern St. Francois County, Mo., across the southern portion of the Farmington anticline, which is outlined in fig. 1. From evidence obtained from fossil fragments taken from some of the intrusives, and from the similarity in composition of the more basic intrusives to those of known age in surrounding states, these pipes and dikes are believed to be at least post-Devonian and probably Cretaceous in age. As first postulated by Rust,<sup>1</sup> these intrusives are believed to represent explosion tubes, or diatremes, punched through great thicknesses of solid rock by gaseous pressure.

The particular intrusive studied is located about 1 mile north of the small town of Avon at an elevation of 880 ft in the E½ SE¼ NE¼ of sec. 2, T. 35N, R. 7E. Outcrop is through the Bonnetterre dolomite in two separated exposures. The larger outcrop extends east and west for about 50 ft along a tributary in a small ravine (fig. 2). The smaller exposure occurs as a few scattered patches over an area 20 ft square about 100 ft to the north. The area between the exposures is covered, but scattered patches of the Bonnetterre dolomite outcrop throughout the area surrounding the diatreme. Where exposed, contact relations show the surrounding dolomite to be shattered and metamorphosed to a fine-grained rock for a distance of 15 to 20 ft from the intrusive.

The igneous rock occurs as a dark greenish-gray porphyry near the center of the diatreme and as a fine-grained, greenish-gray material containing lapilli and metamorphosed fragments of dolomite near the contact with the country rock. The most abundant original mineral of the porphyritic rock is olivine now largely serpentinized as the result of extensive hydrothermal alteration at time of emplacement. The most common constituent of the fine-grained rock, which occurs near the border of the intrusive, is calcite. Mica is a common constituent of both types of rock, occurring as tufts or

flakes. Magnetite makes up about 2 pct of the diatreme and occurs as small irregular grains or streaks of tiny grains. Singewald and Milton<sup>2</sup> have termed the rock of this body an "augite-free alnöite."

Susceptibility measurements of pulverized specimens were made from selected samples taken from both the diatreme and the surrounding country rock. The susceptibility of the Bonnetterre dolomite was found to be less than  $40 \times 10^{-6}$  cgs (centimeter-gram-second) units and that of the metamorphosed contact dolomite less than  $200 \times 10^{-6}$  cgs. Measurements of individual samples of the igneous rock showed a wide variation in susceptibility between limits of  $1000 \times 10^{-6}$  cgs to  $8000 \times 10^{-6}$  cgs, with the maximum difference occurring in samples taken only a few feet apart. The maximum value was exhibited by a highly weathered sample taken from the northern outcrop, while minimum susceptibility was determined from a partially weathered sample taken only a few feet away. These extremes are believed to represent small local concentrations of stringers and grains of magnetite crystals in the igneous rock.

A detailed magnetic survey of the horizontal and vertical components of the earth's magnetic field was made over the diatreme area. Stations were occupied at 20-ft intervals as shown by the grid on fig. 2. When greater detail was desired, intervening stations were also occupied. The field instruments were supplied by the Department of Geophysics, St. Louis University. The magnetometers are the Schmidt balance type constructed by the Ruska Instrument Co. These instruments are temperature compensated and have a standard sensitivity of 10 gamma. Calibration of the instruments

CHARLES R. HOLMES, formerly with the Department of Geophysics, St. Louis University, St. Louis, Missouri, is now associated with the New Mexico School of Mines, Socorro, New Mexico.

AIME San Francisco Meeting, February 1949.

TP 2927 L. Discussion (2 copies) may be sent to Transactions AIME before Nov. 30, 1950. Manuscript received Sept. 6, 1949.

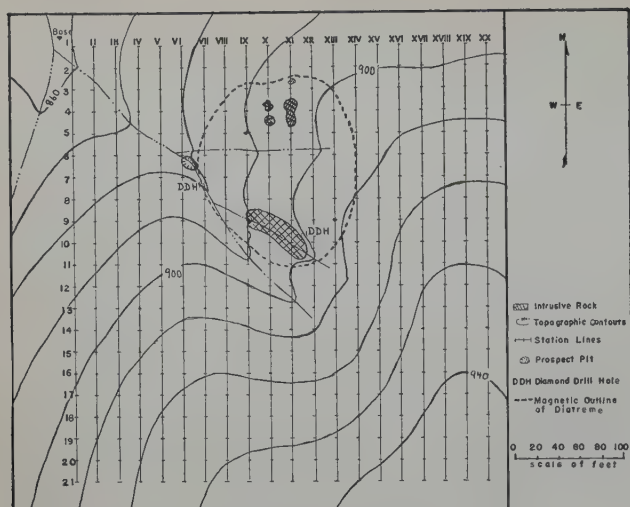


Fig. 2—Geological, topographical, and station map of intrusive.

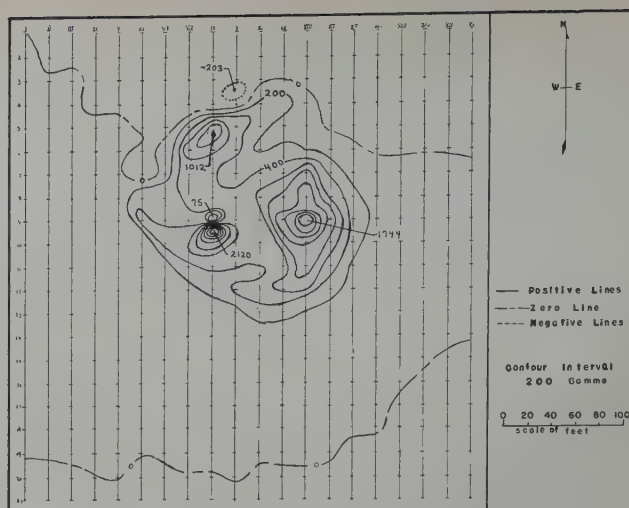


Fig. 3—Vertical isanomalic map.

was accomplished in the field by means of a Helmholtz coil, which was furnished by the manufacturer of the magnetometer. The vertical field balance was found to have a sensitivity of 9.7 gamma per scale division. Sensitivity of the horizontal instrument was determined to be 10.2 gamma per scale division. The sensitivity of each instrument was redetermined at regular intervals throughout the course of the survey and was found to remain sensibly constant. Some slight variation was noted in the second decimal place of the scale constant for each instrument, but the magnitude of change was never sufficient to alter the value in the first decimal place. In the present survey, all calibrations were made at the base station.

A self-recording base station instrument was used to supply diurnal corrections to the data obtained from the field survey of the vertical magnetic intensity. This instrument was also manufactured by Ruska and photographically records temperature changes and diurnal variations in the vertical component of the earth's magnetic field over a 15-hr interval. Timing marks are placed on the record at  $\frac{1}{2}$ -hr intervals. The sensitivity of the instrument was determined to be 10.2 gamma. The records made by the instrument were supplemented by an additional diurnal curve, which was obtained each day from readings taken every 2 hr at the base station with field instruments. Average diurnal variation was less than 50 gamma and usually less than 20 gamma for any single 2-hr period.

The magnitude of the change in the anomalous magnetic field over the diatreme necessitated the use of auxiliary magnets with the field instruments. The moments of these magnets are subject to variation with change in temperature and thus introduce a small unavoidable error into the survey. The level bubbles of these particular instruments are extremely sensitive to the effect of the sun's rays, and, although care was exercised in orienting and leveling the instruments, some small additional error may have resulted. Because of these factors, base lines were run for the survey to provide checks and corrections to the other lines. Base line corrections are applied linearly. Particular care was exercised in taking readings along the base lines. They were run during cool cloudy periods, when disturbing influences were at a minimum. From the base line readings and from numerous other checks

made during the course of the survey, it is believed that the overall accuracy of the magnetic survey after corrections is to within 10 gamma.

Fig. 3 represents an isanomalic map constructed for the magnetic anomaly in the vertical intensity over the body. Three separate positive areas are present within the limits of the anomaly. The most intense high is centered about 10 ft south of station 9 on traverse line IX. This high has a range of over 2100 gamma but is limited in areal extent. Another positive area occurs just to the west of the smaller igneous exposure at station 5, line IX, with a maximum anomaly of 1012 gamma. A much broader positive area occurs centered over station 9, line XIII, northeast of the main outcrop. The maximum value here is 1744 gamma. An inspection of the map reveals that these three "magnetic highs" are superimposed upon a much broader positive anomaly nearly circular in extent.

The horizontal component of the anomalous magnetic field over the same region is shown in fig. 4. The maximum variation in intensity again was found just south of station 9, line IX, where a low of 1820 gamma occurred. The anomaly centered over station 9, line XIII, shows a range of about 1200 gamma and that at station 5, line IX, a minimum value of 1007 gamma.

Fig. 5 represents the north-south profile for the two anomalies centered on line IX. The appearance of the profile indicates that the intensity values near station 9 are abnormal. Since at this station the horizontal profile is displaced far below what should be its normal relation to the vertical profile, with respect to a zero axis, the anomaly cannot be explained by the mere near-surface concentration of magnetic material. The larger exposure of the diatreme outcrops on the line at station 9 as a small cliff about 5 ft high and facing south. The greatest variation in both horizontal and vertical intensities occurs within 5 ft of the cliff face and most probably results from flux concentration at the edges of the outcrop. The anomaly to the north at station 4 is much broader in areal extent in relation to range of magnitude than is the disturbance at station 9. In this case, the displacement of the horizontal curve below the zero axis reflects the normal negative value observed north of the center of a disturbing body in the northern hemisphere. Although there may be other factors



contributing to the magnetic disturbance at station 4, the regular character of the magnetic intensity curves and the elliptical pattern of the lines of the isanomalic maps in the vicinity of the station suggest the concentration of magnetite grains as the basic cause of this local anomaly. Higher than average susceptibility values exhibited by rock specimens taken 10 ft south of station 3 on line X lend credence to this inference. The result of the mineral concentration is also evident in the magnetic intensity values for adjacent stations on lines VIII and X.

Although the data show that the magnetic intensity profiles for lines X through XIII exhibit a considerable diversity in range of intensity and appearance, each line possesses three distinct intensity peaks centered approximately at stations 4, 9, and 12. Blum<sup>2</sup> investigated the vertical magnetic field over two protrudent volcanic pipes in Colorado and observed that the intensity values over the circular edges of the pipes were greater than toward the center and that the southern half of the circumference has a stronger field than the northern half. He attributed the latter to the inclination of the earth's magnetic field. These observations find analogy to the present study with the exception of the relative magnitude of the magnetic field toward the center of the diatreme. The intensity values over this body are a direct reflection of its uneven near-surface configuration. The circular edge of the diatreme is for the most part buried beneath a covering of soil and mantle rock. Thus the point of maximum anomaly will be shifted toward the main outcrop. Other things equal, the relation of the magnitude of the magnetic intensities toward the center of the body to those over its edges will depend on the sharpness of the edge and its distance beneath the surface at the point of observation. Therefore, the edge effects serve to outline the plan shape and extent of the diatreme. The contact of the intrusive with the country rock is exposed just west of station 7, line VII, and may also be observed 10 ft north of station 3 on line XI. Although edge effects are shown by the observed magnetic data for intermediate lines across the body, there are no breaks in the curves for lines VII and XIV. A comparison of the character of these two lines, together with known contact relation near line VII, indicates that line XIV bounds the eastward extension of the body. Then if the

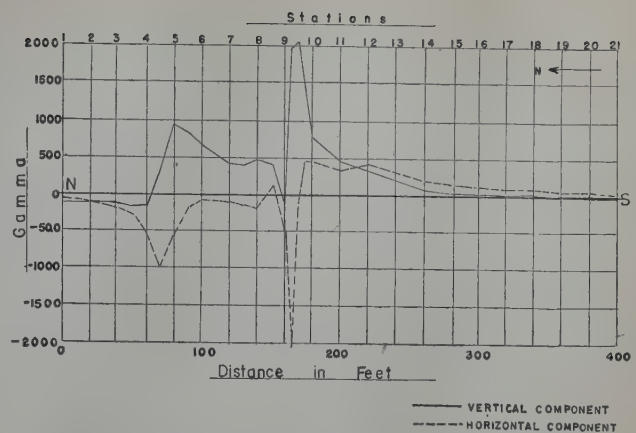


Fig. 5—Magnetic intensity profiles for line IX.

normal southward displacement of the edge effects due to the inclination of the earth's magnetic field is considered, the diatreme can be delineated as an elliptically shaped vertical pipe with the long axis of the ellipse oriented in a north-south direction. The probable outline of the intrusive is shown in fig. 2.

From the outcrop pattern of the intrusive and the sharp changes in intensities of the magnetic profiles north of the main outcrop, it was first thought that the diatreme was not continuous beneath the covered area between the two outcrops but either branched some distance beneath the surface or was separated into two distinct bodies. To test these relations and to determine a possible limiting depth to the diatreme beneath the covered area, a two-dimensional analysis of the body was made for the north-south line across the center of the diatreme. The assumptions on which the analysis is based are: (1) that the body is infinite in extent vertically downward and perpendicular to the magnetic meridian, (2) that there is no variation in susceptibility throughout the body, (3) that near surface effects of flux concentration along corners and edges can be disregarded. None of these assumptions is valid in the actual case, but if the limiting factors are considered, some conclusions still can be made on the structure of the body. For the purpose of analysis, the body was divided into three adjacent blocks based loosely on known contact and outcrop relations along line XI. The northern block was assumed to be 20 ft in width along the traverse line and to lie 5 ft below the level of observation with its northern and southern edges at points 10 ft south of station 3 and 10 ft south of station 4, respectively. The top of the southernmost block was assumed to be at the same level with its north edge at station 7 and its south edge at station 11. Then the elevation of the top of the central block was altered with respect to the other two until the resultant of the curves computed from all three blocks approached the observed curve in character. A depth of 20 ft below the level of observation was found to be the maximum depth that could be taken for the central block and still have the theoretical curves approach those constructed from the field data. Under the assumptions made, it constitutes a limiting depth to the body for the covered area between the outcrops and indicates that the diatreme cannot be taken as two separate bodies. It is readily admitted that a two-dimensional analysis is not directly applicable to the calculation of the exact configuration of a pipe-shaped body. The analysis is undertaken only to show the maximum depth to the covered section, and, as such, serves the purpose.

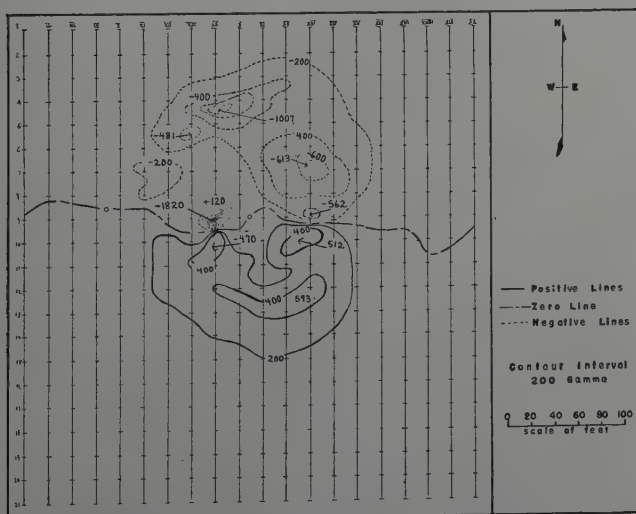


Fig. 4—Horizontal isanomalic map.

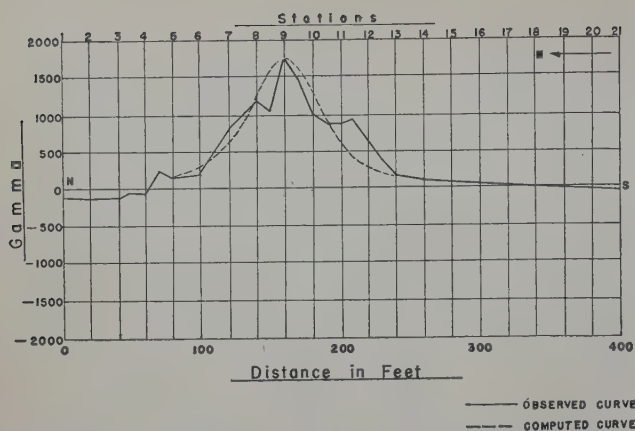


Fig. 6—Vertical intensity profiles for line XIII.

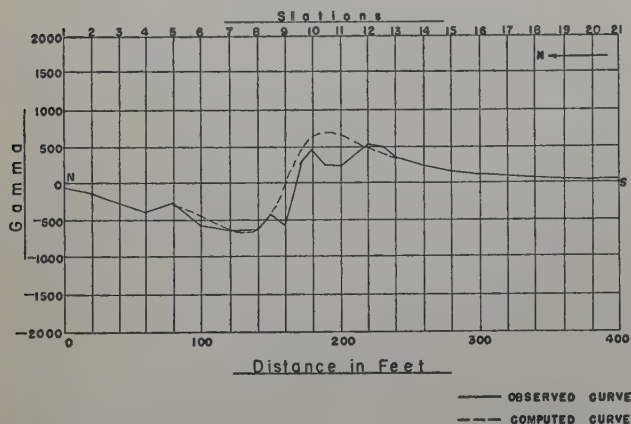


Fig. 7—Horizontal intensity profiles for line XIII.

The appearance of the isanomalous maps suggests that the anomaly centered over station 9 of line XIII results from the concentration of stringers and grains of magnetite and that the resultant magnetic field in the immediate vicinity of the station might be represented by an isolated magnetic pole. Total intensity vectors constructed from the two observed magnetic components intersect along line XIII from 40 to 50 ft beneath the surface at station 9. Figs. 6 and 7 illustrate the correspondence between the actual curves constructed from the observed data and type curves calculated from an equivalent pole at the minimum depth of 50 ft. Even with the pole located at this minimum depth, magnetic intensity values calculated from it for stations east of line XIII are considerably higher than observed values. The rapid decrease observed in the magnitude of the magnetic field east of line XIII indicates that the anomaly centered here cannot be attributed entirely to the effects of mineral concentration but must be at least partly caused by near-surface effects of the underlying body.

A supplemental gravimetric survey was also carried out over the area in an attempt to confirm the magnetic findings. The instrument used was a small Atlas gravity meter, model F, with a standard sensitivity of 0.1 milligal. Two traverses were run: one north-south along line XI and the other east-west along the line of station 7. Density determinations were made from selected rock samples by the method of weighing in water and in air. A few additional determinations were made from rock specimens cut into regular geometrical shapes. The average density of the unweathered porphyritic rock of the diatreme is 2.89. The fine-grained igneous contact rock containing dolomitic inclusions averaged 2.68 and the

densities of the metamorphosed country rock just outside the contact, 2.70. Density of the unaltered Bonnetterre dolomite is 2.73. The density of the weathered alnöite depends on the degree of decomposition and is extremely variable. The average value for highly weathered solid samples is 1.89. These values fix the maximum effective density contrast between the diatreme and the surrounding country rock at 0.16.

The maximum anomaly to be expected over the center of a vertical circular cylinder that is infinite in depth can be calculated from the equation:<sup>4</sup>

$$g = 2 \pi k \xi r$$

where  $r$  = the radius of the body in centimeters;

$\xi$  = the density contrast;

$k$  = the gravitational constant,  
which is  $6.664 \times 10^{-8}$  cgs

Assuming the diatreme to be circular in plan with a mean radius of 2280 cm, the anomalous value that results from a density contrast of 0.16 is only 0.162 milligals. Therefore any interpretation of the gravitational field requires that the resolution of the gravity data be within this limiting value. As conducted, the gravity survey constituted a trial run for the instrument. It since has been observed that the drift of the gravity meter is nonlinear for 3 to 7 min after the instrument is in position for the observation. The possible errors compounded from nonlinear instrumental drift and elevation corrections to the observed data reduce the limit of accuracy of the survey to 0.2 milligals. This value exceeds the magnitude of the theoretical anomaly and indicates that, except under exceptionally favorable conditions, the gravity method is of little value in the investigation of these igneous pipes.

Although the anomalous gravity field is not definitive, the other evidence is sufficient for a solution of the problem. The geologic relations indicate that the body was intruded into the surrounding sediments with the explosive violence of a diatreme. The analysis of the observed data indicates that the diatreme is an elliptically shaped pipe intruded vertically upward into the surrounding sediments with its long axis oriented in a north-south direction. The concentration of magnetic materials and the near-surface effects of flux concentration over corners and edges of the underlying field greatly influence the character and magnitude of points of maximum anomaly over the intrusion. From the edge effects in the anomalous magnetic field and the known contact relations, the causal body is believed to measure about  $160 \times 140$  ft in the plan section. The inference as to the form of the diatreme is in substantial agreement with the results of a previous investigation conducted by the Missouri Geologic Survey.<sup>5</sup>

## References

- <sup>1</sup> G. W. Rust: Preliminary Notes on Explosive Volcanism in South Eastern Missouri. *Journal of Geology*. (1937) 45, 48-75.
- <sup>2</sup> J. T. Singewald, Jr. and C. Milton: An Alnöite Pipe, Its Contact Phenomena and Ore Deposition Near Avon, Missouri. *Journal of Geology*. (1930) 38, 54-66.
- <sup>3</sup> Victor J. Blum: The Magnetic Field and the Geology of the Canyon City Area. PhD Dissertation, St. Louis University. (1944) 186.
- <sup>4</sup> C. A. Heiland: Geophysical Exploration. 148 (1940) New York. Prentice Hall.
- <sup>5</sup> Albert L. Kidwell: Post-Devonian Igneous Activity in Southeastern Missouri. Mo. Geol. Survey. R. I. 4 (1947).



# Discussion\*

## Contents

### A—Metal Mining

### B—Minerals Beneficiation

### F—Coal

### H—Industrial Minerals

#### A—Metal Mining

- Diamond Drilling Quartz-Feldspar Intergrowths. (Paper by L. C. Armstrong. *Transactions AIME*, **184**, 177; *Mining Engineering*. June 1949. Discussion by B. J. Westman.)..... 1148

#### B—Minerals Beneficiation

- The Effect of Mill Speeds on Grinding Costs. (Paper by H. Hardinge and R. C. Ferguson. *Transactions AIME*, **187**, 1127; *Mining Engineering*. November 1950. Discussion by Oscar Johnson and F. C. Bond.) 1149
- Northern Rhodesia Mulfulira Copper Mines, Limited Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills. (Paper by Jack White. *Transactions AIME*, **187**, 96; *Mining Engineering*. January 1950. Discussion by Oscar Johnson and W. C. McKinnon.)..... 1151
- Progress Report on Grinding at Tennessee Copper Company. (Paper by J. F. Myers and F. M. Lewis. *Transactions AIME*, **187**, 707; *Mining Engineering*. June 1950. Discussion by W. I. Garms.)..... 1151
- Behavior of Mineral Particles in Electrostatic Separation. (Paper by Shiou-Chuan Sun, J. D. Morgan, and R. F. Wesner. *Transactions AIME*, **187**, 369; *Mining Engineering*. March 1950. Discussion by O. C. Ralston and F. Fraas, and S. B. Hudson.)..... 1152
- Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units. (Paper by James J. Bean. *Transactions AIME*, **187**, 365; *Mining Engineering*. March 1950. Discussion by F. M. Lewis.)..... 1153
- Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries. (Paper by S. A. Falconer. *Transactions AIME*, **187**, 790; *Mining Engineering*. July 1950. Discussion by D. A. Dahlstrom.)..... 1153
- Concerning the Adsorption of Dodecylamine on Quartz. (Paper by A. M. Gaudin and F. W. Bloecher, Jr. *Transactions AIME*, **187**, 499; *Mining Engineering*. April 1950. Discussion by G. L. Simard and D. J. Salley.)..... 1154
- Measurement of Equilibrium Forces Between an Air Bubble and an Attached Solid in Water. (Paper by T. M. Morris. *Transactions AIME*, **187**, 91; *Mining Engineering*. January 1950. Discussion by H. H. Kellogg and F. X. Tartaron.)..... 1154
- Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite. (Paper by R. Schuhmann and B. Prakash. *Transactions AIME*, **187**, 601; *Mining Engineering*. May 1950. Discussion by Maurice Rey.)..... 1155
- Continuous Countercurrent Decantation Calculations. (Paper by T. B. Counselman. *Transactions AIME*, **187**, 223; *Mining Engineering*. February 1950. Discussion by C. G. McLachlan.)..... 1155
- Preliminary Report of Massco Circuitron. (Paper by A. E. Craig, William J. Tait, and E. P. McCurdy. *Transactions AIME*, **187**, 1038; *Mining Engineering*. October 1950. Discussion by C. M. Marquardt.)..... 1155
- An Improved Method of Gravity Concentration in the Fine-Size Range. (Paper by A. Thunaes and H. R. Spedden. *Transactions AIME*, **187**, 879; *Mining Engineering*. August 1950. Discussion by R. R. Knobler and F. E. Albertson.)..... 1156

#### F—Coal

- Coal Preparation for Synthetic Liquid Fuels. (Paper by W. L. Crentz, J. D. Doherty, and E. E. Donath. *Transactions AIME*, **187**, 507; *Mining Engineering*. April 1950. Discussion by A. T. Cross.)..... 1158
- Cyclone Thickener Application in the Coal Industry. (Paper by M. G. Driessen and H. E. Criner. *Transactions AIME*, **187**, 102; *Mining Engineering*. January 1950. Discussion by M. Rey.)..... 1158
- Kerosine Flotation of Bituminous Fine Coal. (Paper by L. E. Schiffman. *Transactions AIME*, **187**, 1047; *Mining Engineering*. October 1950. Discussion by W. J. Parton.)..... 1159

\* TP 2978

Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal. (Paper by M. R. Geer, H. F. Yancey, C. L. Allyn, and R. H. Eckhouse. <i>Transactions AIME</i> , <b>187</b> , 1057; <i>Mining Engineering</i> . October 1950. Discussion by W. M. Bertholf, J. D. Price, J. F. McLaughlin and H. H. Otto, and W. E. Brown.) .....	1160
Quantitative Efficiency of Separation of Coal Cleaning Equipment. (Paper by W. W. Anderson. <i>Transactions AIME</i> , <b>187</b> , 256; <i>Mining Engineering</i> . February 1950. Discussion by John Griffen, H. F. Yancey and M. R. Geer, and M. G. Driessen.) .....	1162
Operating Data for a Bird Centrifuge. (Paper by A. C. Richardson and Orville R. Lyons. <i>Transactions AIME</i> , <b>187</b> , 381; <i>Mining Engineering</i> . March 1950. Discussion by F. X. Ferney.) .....	1166

## H—Industrial Minerals

Concrete Aggregates for Large Structures. (Paper by E. T. Rexford. <i>Transactions AIME</i> , <b>187</b> , 395; <i>Mining Engineering</i> . March 1950. Discussion by G. B. Walker.) .....	1168
Ground Water in California. (Paper by J. F. Poland. <i>Transactions AIME</i> , <b>187</b> , 279; <i>Mining Engineering</i> . February 1950. Discussion by B. C. Burgess.) .....	1168
Industrial Mineral Economics and the Raw Materials Survey. (Paper by R. B. Ladoo and C. A. Stokes. <i>Transactions AIME</i> , <b>187</b> , 292; <i>Mining Engineering</i> . February 1950. Discussion by Bruce C. Netschert.) .....	1169
Recent Developments in the Manufacture of Lightweight Aggregates. (Paper by J. E. Conley and J. S. Ruppert. <i>Transactions AIME</i> , <b>187</b> , 479; <i>Mining Engineering</i> . April 1950. Discussion by W. B. Mather.) .....	1170
Lightweight Aggregate Industry in Oregon. (Paper by N. S. Wagner and R. S. Mason. <i>Transactions AIME</i> , <b>184</b> , 385; <i>Mining Engineering</i> . November 1949. Discussion by W. B. Mather.) .....	1170
Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries. (Paper by S. A. Falconer. <i>Transactions AIME</i> , <b>187</b> , 790; <i>Mining Engineering</i> . July 1950. Discussion by D. A. Dahlstrom.) (See page 1153.)	

## A — Metal Mining

### Diamond Drilling Quartz-Feldspar Intergrowths

by L. C. Armstrong

#### DISCUSSION

**Burton J. Westman**—Besides decreasing the diamond size, there appear to be two other approaches open to overcome excessive diamond loss and, more particularly, the rapid diamond polish that took place in this quartz-feldspar rock. First, however, a short discussion of the diamond characteristics should be made in order to discuss the situation more clearly.

The West African diamonds which we can assume were used for this drilling have two disadvantages when drilling hard, exceptionally fine-grained rocks. These are rapid polishing and, if the bit isn't "drilled in" carefully, a tendency to damage, resulting in excessive diamond loss. The normal wear resistance of any diamond is inversely proportional to the compressive strength of the diamond, therefore only the highest grade of diamonds free from insipient fractures should be employed. Using upwards of 90 diamonds per carat instead of the 20 to 30 per carat not only decreases the total projected area of the cutting points but increases the stress per point which, in turn, causes greater depth of penetration into the rock. Considering this depth of penetration, therefore, suggests an approach other than increasing the number of diamonds per carat and that is to decrease the actual number of diamond points (also the carat weight per bit) so that the penetrating force on an individual diamond is increased. This penetration increases the chipping or rock shearing action.

It should be considered that diamonds of a sufficient number that present a fairly large projected contact surface will, as in the 20 to 30 per carat size, require an exceptionally high bit force sufficient to cause a great enough "point force" to penetrate the hard, fine-

grained rock and, if this weight is insufficient, the abrading action rapidly polishes the stones; therefore, by decreasing the point size, thereby increasing the point weight, penetration is affected. A similar effect may be had with the 20 to 30 per carat stones by reducing the number of diamond points so as to reduce this contact area and, at the same time, create sufficiently high point weight to affect the penetration. The latter, however, has a limit that is governed by the strength of both the diamond and the setting bond.

The polish on a West African diamond in a given hard, fine-grained rock is directly proportional to the force applied to the bit. This polish is often severe enough to retire a bit after relatively few feet of coring. To overcome this problem the Congo diamond is being employed because this type of stone does not take a polish since it wears away at a uniform rate and presents a minutely jagged surface that continues to cut for the full life of the diamond. The sizes of this diamond available, however, are seldom smaller than around ten per carat.

In general, it can be stated that for fine-grained, hard rock there are four possible approaches to reduce polish and excessive diamond loss. First is the use of numerous small diamonds, second is the use of fewer large diamonds, third is employing the Congo diamond, and fourth is the new type of crown design that has 12 to 24 small waterways, depending upon the bit size, and which has had a marked effect on reducing this polish apparently by increasing the sludging efficiency which practically eliminates the possibility of regrinding the cuttings.

**L. C. Armstrong** (author's reply)—My reaction is that all of Mr. Westman's contributions are based on considerable study and experience. I feel that his suggestions should prove useful to those engaged in drilling resistant, intergrowth-bearing rocks.

**B. J. WESTMAN**, Koebel Diamond Tool Co., Detroit, Mich.



## B — Minerals Beneficiation

### The Effect of Mill Speeds on Grinding Costs

by Harlowe Hardinge and R. C. Ferguson

#### DISCUSSION

**Oscar Johnson**—In my opinion, the effect of mill speeds on grinding costs must be studied along with capital investment and dollars gathered together as profits.

Comparing the entire groups of operators with those who have had the opportunity to make slow-speed mill studies, I think you will find the latter small in numbers. Most managers want the equipment worked to its maximum output. There are, however, some installations where plant and mill sizes are such that they can do the job with reduction of mill barrel speeds.

The past and the present installations of the industry are laid out to get the most capacity for the least capital outlay. This is the case even with the plants of Chile Exploration, International Nickel, Morocco, and Anaconda, now under construction or being changed. The industry recognizes that most all equipment it buys today is good and can be depended upon for efficient performance.

Under this scheme of things, I am doubtful that slow-speed ball mill operation will be generally applicable.

With reference to the U. S. Bureau of Mines laboratory tests, I think table II could have been omitted. It is inconclusive as to maximum efficiency for the low-pulp level mill on hard ore. There should be no question about this point. However, data on mill speeds can be found to substantiate various theories as well as refute them.

Gow, Guggenheim, Campbell and Coghill, in their paper on Ball Milling,<sup>2</sup> believe their 2 x 2 ft laboratory mill reflects results that can be expected from large mills. If so, then referring to their table II, they state, "The conclusion to be drawn from this second series is that high speed, not exceeding 72 pct of the critical, favors capacity, as before, but that with proper conditions of operation high speeds may give as good efficiency values as low speeds. In this case the efficiency values are nearly constant. A horizontal curve would indicate that the amount of grinding was directly proportional to the power expended, and these tests suggest that such a condition can be made to exist in commercial operations."

Table II

(From Paper by Gow et al)<sup>2</sup>

Speed, Pct Critical	32	42	52	62	72	82
Capacity:						
Surface tons per hr (65-mesh)	26.6	42.1	54.4	65.9	74.3	74.1
Surface tons per hr (200-mesh)	56.1	87.4	112.7	137.1	154.2	153.0
Efficiency:						
Surface tons per net hp hr (65-mesh)	35.7	36.3	36.3	35.4	34.3	32.3
Surface tons per net hp hr (200-mesh)	75.3	75.3	75.1	73.7	71.0	66.0
Ore in mill, Lb.	98	100	100	113	122	165

The field performance data, table III, represents much effort in its collection and preparation. But, one must realize that there are many variables that effect the efficiency of grinding mill operation, and too much must not be assumed as to the effect of some specific

change. Possibly with changes in mill speed, the results might be more consistent by also a change in ball rationing, type of ball, volume of ball charge, pulp level and amount of pulp in the mill, pulp consisting, design of liner, circulating load, etc. Also, changes in ore character must be reckoned with when evaluating grinding performance.

At present the Climax Molybdenum Corp. is running at much reduced capacity. Mr. James Duggan informs me that at mill speeds of 17 rpm, they save a \$.025 per ton on liners and \$.025 per ton in power, but, if the demand for molybdenum increased, he would go back to higher speed to obtain maximum tonnage, as the values from the increased tonnage would far more than offset the one half saving at the slower speed.

The Inspiration ran a six months' test between mills running 21 rpm and 23.5 rpm. The slower mills ground 10 pct less ore with a slight saving per ton, but when the reduced plant tonnage was checked back into the actual cost figures of concentration, the high-speed mills with their greater tonnage showed considerable advantage.

To be convinced of possible practical results from the predictions in the conclusions, I think we would have to rely on the analysis of expert cost accountants to furnish the necessary proof figures.

Hardinge and Ferguson are to be commended for the work in preparing this paper. I am convinced that our Massco engineers should go into higher speeds with our equipment.

**Harlowe Hardinge** (authors' reply)—For one, I heartily agree with Mr. Johnson's opening statement that the effect of mill speeds on grinding costs must be studied along with capital investment and dollars gathered together as profits. It was on this basis and for this reason the paper was written.

Mr. Johnson, on the other hand, takes the position that, on the whole, low speeds are not justified from the economic standpoint, basing his principal reason on the fact that lower mill speeds cut mill capacities and hence reduce the gross income from the product produced. There is no denying this point. It is almost axiomatic. It is for this very reason that the overall advantage of lower mill speeds has been discounted and even overlooked. It was for this reason mainly that the paper was written in the first place.

It is one thing to plan an efficient operation at the outset, basing one's figures on the tonnage requirements at the time, and it is quite another to be confronted with the problem of increasing the output of an existing installation at a minimum of capital expenditure.

Economic consideration of a new installation is greatly influenced by referring to an old one. Too often, the analyst assumes that if this practice is followed in the new installation, one would not go wrong. It is just here that he may be wrong. Past practice and low capital expenditure are all too frequently given priority over the engineer's analysis of operating costs. When we are able to start fresh, we should give proper weight to other economic factors which do not exist in an old installation. It is these economic factors that make it possible to spend at the outset just a little more money and get it back in a matter of months and effect big savings for years to come.

**F. C. Bond**—This paper is of considerable importance in that it emphasizes a modern trend to operate ball mills at somewhat slower speeds than formerly. We have checked the data in the paper with that obtained

**Table X. Effect of Low Pulp Load in Mill at Different Speeds**

Critical Speed, Pct	Ore in Mill, Lb	Surface Tons		Loss Eff., Pct
		Per hp-hr	Avg.	
30	75	21.4	21.30	2.74
	50	21.4		
	35	21.1		
40	75	21.6	21.43	2.15
	50	21.4		
	35	21.3		
50	75	21.8	21.60	1.37
	50	21.5		
	35	21.5		
60	75	21.8	21.63	1.23
	50	21.6		
	35	21.6		
70	75	21.7	21.70	0.91
	50	21.7		
	35	21.7		
80	75	21.9	21.90	0.00
	50	21.9		
	35			

from other sources and find ourselves in general agreement with the authors' conclusions.

Decreasing the speed of overflow wet grinding ball mills to about 50 pct to 55 pct of critical ordinarily decreases the cost per ton ground, both in metal wear and in power consumption, with certain limitations. The decrease in metal wear is usually larger than the decrease in required energy; and this enlarges the benefit to be obtained by slower speeds at the present time. In the past decade the cost per pound of grinding media and liners has approximately doubled, while the cost of power has increased only slightly, if at all, so that at present the cost per ton ground in metal wear averages somewhat more than the energy cost. Future cost trends are largely unpredictable.

In any particular installation the decision regarding the proper mill speed must be controlled by local economic factors. The increased initial cost of the larger mill required for slow speed operation must be justified by the expected life of the property, decrease in maintenance cost, etc., as well as the saving in metal wear and power consumption.

There are several points in the paper that deserve comment. The use of "surface tons" as a measure of grinding efficiency is widely questioned because it does not assign sufficient importance to the work done on the —200-mesh portion of the product. However, it may furnish an acceptable value for the "useful" work done according to the way in which that elusive quantity may be defined. It appears to be acceptable in this paper because all of the grinding tests are carried to approximately the same percent passing 200-mesh.

Another important uncertainty is the assumed amount of pulp present in both overflow and low-level discharge ball mills. The ball voids in the test mill contained approximately 75 lb of dry solids. In tabulating the data for overflow mills, the average of three runs at 200, 150, and 125 lb of solids, or 158 lb, was used. This is more than twice the amount of solids contained in the ball voids, and it might be questioned whether the usual commercial overflow ball mill contains this amount of pulp. The low-pulp level mill data are computed for loads of 100, 75, and 50 lb of solids in the mill, or an average value equal to the amount contained in the ball voids. Again it may be questioned whether a low-level mill does not contain ball voids in the upper part of the charge which are free of pulp.

Actual measurement of the amount of pulp contained in grinding mills, both of the overflow and low-level discharge types, as compared with the volume of the ball or rod voids and total volume below the discharge level, would be highly desirable. Such a measurement could be made by diverting and collecting the mill discharge from the instant the mill feed is shut off, and adding this to the pulp retained within the mill after

shutdown. The results might be surprising and illuminating.

Table II is computed on average loads in the mill of 100, 75, and 50 lb. A similar table (X) is given for average loads of 75, 50, and 35 lb, which allows for some unoccupied void space in the ball charge. This table shows that the percent loss in efficiency in grinding chert at low critical speeds becomes larger as the pulp load in the mill decreases. Data are not available for the construction of a similar table on dolomite.

The data appear to justify the following conclusions regarding mills receiving relatively fine feed:

1. Considerable savings may be effected in long time operations by decreasing the speed of wet grinding overflow ball mills to perhaps 50 pct to 55 pct of the critical speed.

2. The saving will be greater in the case of overflow mills than in mills carrying a low-pulp level, and the optimum speed of low-level mills from a cost per ton ground standpoint is somewhat higher than that of overflow mills.

3. The optimum mill speed when grinding hard material is slightly higher than that for soft material.

4. Slower mill speeds may require a larger classifier for the same feed tonnage because of increased circulating load. However, this point has not been established.

**R. C. Ferguson** (authors' reply)—Mr. Bond questions the use of 200, 150, and 125 lb of solids in a mill containing a 796-lb ball charge as being comparable to the amount of solids in a trunnion overflow mill as the average of 158 lb of solids is more than twice the amount of solids required to fill the voids in the ball charge. Likewise, he questions the 100, 75, and 50-lb ore charge's being considered equivalent to a low-pulp level mill as the average charge of solids is equal to the amount that would be contained in the ball voids.

The only published data known to the writer on the amount of dry solids contained in a ball mill are given by E. W. Davis.<sup>4</sup>

This paper covers three tests made to determine the amount of pulp and the dilution in a 3 x 3 ft overflow cylindrical mill. The results found were as follows:

Test No.	Ball Charge, Lb	Dry Solids, Lb	Ratio Balls to Solids
1	1884	955	1.98 to 1
2	2006	606	3.32 to 1
3	2000	259	7.73 to 1

Referring to the ore charges in question, we have:

Ball Charge, Lb	Ore Charge, Lb	Ratio Balls to Solids
796	200	3.98 to 1
796	150	5.30 to 1
796	125	6.38 to 1
796	100	7.98 to 1
796	75	10.60 to 1
796	50	15.90 to 1

It will be noted that the ball to ore ratio of 200, 150, and 125 is in the range found in the 3 x 3 ft overflow mill and that the ratio for the 100, 75, and 50 is below the ratio found in the 3 x 3 ft mill.

Mr. Davis does not show the specific gravity of the ore, but he states that it is magnetic ore from New York State and it was being ground for concentration tests; therefore, the specific gravity of this material should not be above 3.5 and would not affect the above comparisons to any great extent.

<sup>4</sup> E. W. Davis: Pulp Densities within Operating Ball Mills. Transactions, AIME (1946) 169, 155; Mining Technology (May 1945) TP 1843.

OSCAR JOHNSON, *The Mine and Smelter Supply Co., Denver, Colo.*; F. C. BOND, *Allis-Chalmers Mfg. Co., Milwaukee, Wis.*



# Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills

by Jack White

## DISCUSSION

**W. C. McKinnon**—The paper by Mr. Jack White represents a most interesting comparison. To fully evaluate the grinding tests we should have such pertinent data for the two types of ball mills as follows: mill speed, percent solids in the mill discharge, type and contour of liners (particularly shell liners), type of discharge grate used in the cylindrical mill (intermediate or low level), net open area of grate and amount of blinding during tests.

The high consumption of liner steel and balls in the cylindrical grate mill is not surprising if the mill was run at high speed and if it was equipped with a full or low-level type discharge grate.

The results would be of great interest if the cylindrical mill could be operated as an overflow type mill under the same conditions that prevailed in the grinding tests reported.

**Jack White** (author's reply)—As I am no longer at Mufulira I have not the data available to make positive statements but speaking from memory, the speed of both types of mills was 18.2 rpm, the percent solids of the discharge was about 70 pct. Liners were of the usual wave type in both mills. The net open area of the grates was about 45 pct and no blinding was noticed during the tests.

Before I left Mufulira, we converted the cylindrical mill to an overflow type mill, but there was not time to conduct any tests on it. No doubt these will be done in due course and it is suggested that a letter to the Mill Superintendent at Mufulira, Mr. A. A. Finn, would result in data on these tests.

**Oscar Johnson**—We note from Mr. White's interesting paper that all of the grinding units were equipped with the same size classifier, namely 8 x 30 ft.

When you consider the October to December operations with the tonnage and circulating loads, the classifier on the No. 10 mill had a feed of 5047 tons per 24 hr as compared to 3954 tons for units 1 to 9. Thus with about 28 pct more feed to the classifier on unit No. 10, I believe this would account for the lower solids in the

overflow to maintain 5.0 pct on 65-mesh against 6.4 on the other units.

If we assume 138 sq ft of classifier pool surface, the No. 10 unit had 36.6 tons per sq ft of surface as against 28.6 tons per 24 hr per units 1 to 9. I wonder what would be the comparison if the No. 10 unit had a 10-ft wide classifier for equal pool surface.

This should help on the classifier overflow product, and permit unit No. 10 to handle a larger circulating sand load with a resulting increase in unit tonnage. We would also expect some additional advantage of a 10-ft diam mill against a 9 ft based on various investigators' findings that the capacity of a mill varies as the diameter to the 2.6 power.

Mr. White has told me that the block type lining was made from cast manganese steel, the balls of forged steel. This clarifies the discussion as to the kind of material in these parts.

I called Mr. White's attention to the fact that the Marcy mill was running only 17.8 rpm (70½ pct C.S.) which is 10 pct lower than the standard recommendations. Regarding this, Mr. White writes, "your remarks about the speed and ball load are very interesting, and it is my intention to continue with the test when the opportunity presents itself, using the information that you have given me. The test, that was the subject of my paper, was conducted in 1942, and in the seven years since then the Marcy has done a good and steady job."

**Jack White** (author's reply)—If a larger classifier had been used on No. 10 grinding unit, I have no doubt that the results would have been different, and it is possible that they would have shown more advantage to the No. 10 mill. It is also possible that we could have run the overflow density in a larger classifier at the same solids as we did in the classifiers on units 1 to 9.

---

OSCAR JOHNSON, *The Mine and Smelter Supply Co., Denver, Colo.*; W. C. MCKINNON, *Allis-Chalmers Mfg. Co., Milwaukee, Wis.*

---

## Progress Report on Grinding at Tennessee Copper Company

by J. F. Myers and F. M. Lewis

## DISCUSSION

**W. I. Garms**—The authors state that when they added 11 tons of balls to the 45 pct volume ball load, the power needle did not budge. The question arises as to whether any increase in capacity accompanied the 11-ton addition. It is hardly conceivable that all of the balls making up the 11-ton addition were sterile. However, as the work done in the ball mill is not by kilowatt-hours per se or balls per se but by kilowatt-hour-balls, it is possible that the kilowatt-hour-balls in the mill before the 11-ton addition had a different amperage, voltage or robustage than those in the mill after the addition and that the overall outcome was that equal work was done before and after the addition.

The authors can keep the efficiency that they got with the 45 pct volume 1-in. ball charge and get additional capacity by putting more kilowatt-hour-balls to work in the form of the addition of 11 tons of 2-in. balls to the present ball charge, and subsequent make-up additions by weight of 20 pct 2-in. balls and 80 pct 1-in. balls.

**J. F. Myers** (authors' reply)—Mr. Garms points his finger at the 11 tons of "sterile" balls added to the Tricone mill. A fact so interesting to us that we felt it should be reported. Clearly, ball slippage at the existing mill speed prevented any further transfer of power from the shell to the ball mass.

We agree that the kilowatt-hour-balls would increase, were we charging rough 2-in. balls as suggested by Mr. Garms. The assumption is, of course, that we would get a corresponding increase in work done. Our reported data (CIME meeting, Vancouver, 1946) shows a gain of work accomplished of 7.6 pct as we decreased the ball size from 2 in. to 1½ in. and a further gain of work accomplished of 2.1 pct when we went from 1½ in. to 1 in. The small ball results reported by H. R. Banks at Chapman Camp are also very convincing. In the case of 2-in. balls they would take the conventional ball path and start cascading. They would thus destroy the classifying pool over the "foot" of the

rolling balls, which is the key to the overall improvement in the Tricone process.

We do not expect that students of grinding can immediately visualize the possibilities of the classifying pool caused by the small rolling balls. That could not be expected until students have directly investigated the phenomena themselves and explored its possibilities.

Our grinding process with smooth 1-in. balls has reduced by nearly one half the metallic losses in the fine micron sizes of the tailing. This is simply because less of the fine micron sizes are produced. Since the +65-mesh size is the same as formerly, a higher percentage of the intermediate sizes are developed. These sizes have the highest floatability, require the least reagents, and use less floating time.

These factors contribute so heavily to the overall economics that dropping our power grinding gain from 28 pct back to 19 pct is a small detail. However, we feel that this is only a momentary situation and that eventually the best features of the grinding and flotation processes can be brought together, which is as it should be. After all, we are not operating a quarry or gravel pit where only mesh tons are important.

Mr. Garms has very recently started operation of some big 10½-ft slow-speed ball mills with 2-in. cast balls and he reports that the ball paths are of the conventional type as planned. Hence, there exists a turbulent pool at the toe of his ball mass. He has accomplished what he set out to do kilowatthourball-wise.

This poses an interesting question. Is the selective grinding action caused by the classifying pool in the Tricone mill applicable to only ores of high specific gravity or does it have a broad application? The trick is that the percent solids in the mill feed must be relatively low to permit classification in the pool to take place. On a low specific gravity ore, such as a porphyry, can the mill dilution be lowered to a classifying point? That, nobody knows at this writing.

Mr. Garms will, of course, operate his mills as he engineered them, but we predict that he will eventually decide that if his kilowatthourball input were applied to only those ore particles in the mill that needed the power, he would be better off in the overall picture. We predict that he will order some smooth, small balls to be sure there is nothing to this selective grinding action by means of an adequate classifying pool in his ball mill. There is some basis for this prediction, as in the past, Mr. Garms has had great respect for the fact that grinding is for the purpose of preparing flotation feed and not grinding per se.

W. I. GARMS, Kennecott Copper Corp., Hayden, Ariz.

## Behavior of Mineral Particles in Electrostatic Separation

by Shiou-Chuan Sun, J. D. Morgan and R. F. Wesner

### DISCUSSION

**O. C. Ralston and F. Fraas**—Dr. Sun and associates have presented an interesting paper not all of which is comprehended by us. The data assembled measure the deflections of particles in an electrostatic field as a function of a number of independent variables and some dependent variables that are not sharply differentiated. These data are all based on a Johnson type machine of definite, well-described geometry, something not often done in electrostatic separation literature.

One new fact brought out by this technique is the effect of coal dust on admixed pyrite and quartz. The effects are opposite in character, as should be expected and we do not agree with the authors that these effects are negligible.

Fraas<sup>2</sup> also used a multiple cell "distribution analyzer" and gives in fig. 5 of his paper a straight line

plot with no humps or curves. This is not necessarily at variance with Sun's results because Fraas used a larger gap between electrodes and had no evidence of particles adhering to or dropping off the charged electrode.

The section of Sun's paper on effect of surface conductivity contains a speculation that the dielectric constant "represents more or less the electrical conductance of the bulk body instead of the surface of the mineral particles." A simple picture of the meaning of the dielectric constant is that it is the specific inductive capacity of a dielectric when used as the dielectric between the plates of a condenser. It is at once evident that the above speculation confuses *capacity* with *conductance*—two definitely independent variables.

We ask the authors to state in what group or subdivision their garnet belongs; what method and units were used in calculating the data of col. A, table I and their meaning; what was the temperature of the carrier roll and, finally, has any effort been made to investigate the effects of particle shape on distribution in the electrostatic field?

**S. B. Hudson**—I have read this article with great interest. We have been engaged in research work on the principles of electrostatic separation in this laboratory for some time now, and our findings agree with those of the authors in many respects. The article shows evidence of careful and valuable research in the field of electrostatic separation.

A "distribution analyzer," very similar to that described in an earlier article by one of the authors,<sup>21</sup> was incorporated in an inclined plate-type electrostatic separator designed and built in the Melbourne University laboratory in 1948 for investigation purposes.<sup>22</sup> The actual splitting edges were machined from ¼ in. perspex, and the paper hoppers were supported on linen thread immediately below the perspex dividers. These dividers fitted into machined slots in a framework to give accurate ½-in. spacings. The hoppers (staggered) fed directly into a rack of test tubes, which is supported on a vertical pantograph arrangement. The rack was positioned with guides on the horizontal pantograph stand, and this ensured positive alignment when replacing the rack after making weighings.

In later work, when much heavier feed rates were used, of the order of 30 to 40 lb per in. per hr a rack fitted with rectangular metal containers and similarly aligned was used.

Some work was done here on comparing the distributions of minerals when passed separately and when passed as a mixture, and it was found that there was quite an appreciable difference in the two results.<sup>23</sup> However, in our separator the particles do not pass down the plate in a single layer, and this difference is probably caused by collisions of one mineral particles with the other mineral particles.

In most of the investigational work here, the change of the center point of the distribution is measured to establish the effect of a variable, such as voltage. Two minerals (zircon and rutile) have been studied rather exhaustively, and it was found that their distributions are very nearly normal. Owing to the sharpness of the distribution curves, the usual method of obtaining the mean or median was inaccurate, and was not used; instead the mean (also the median), calculated on the assumption of a normal distribution, was used to locate the center point of each distribution and proved satisfactory.

The effect of polarity becomes very apparent in the plate-type separator where frictional charges play a very important part when using highly resistive minerals such as zircon. With rutile, a comparatively conductive mineral, polarity of the electrode has little effect. On the other hand, the magnitude of the voltage has a far greater effect on conductive than on resistive minerals.

**Shiou-Chuan Sun** (authors' reply)—Thanks are extended to Drs. Ralston and Fraas for their keen interest in this paper. Their questions concerning coal dust,



compartment box, conductance, garnet, temperature, and particle shape are answered in the same order. The effect of coal dust on the behavior of both pyrite and quartz was appreciated in this paper. The small difference between curves 3 and 4 and also between curves 5 and 6 of fig. 3 was due to that. In a synthetic mixture of 82 pct coal, 14 pct quartz and 4 pct pyrite by weight, the surfaces of both pyrite and quartz were also contaminated by coal particles. The compartment box used by Fraas, having only eight 2-cm compartments, obviously cannot be compared with the distribution analyzer of this paper. In dealing with dielectric constant, the meaning of the term "electrical conductance" is better expressed as "dielectric conductivity," which is equivalent to relative permittivity, dielectric constant, and specific inductive capacity. The garnet used was rhodolite collected originally from Macon County, N. C. The method of calculation and the meaning of the data in col. A, table I can be found in Taggart's "Handbook of Mineral Dressing." Both the grounded roll and the charged roll of the electrostatic separator were operated at room temperature. No attempt was made to determine the effect of particle shape on distribution in the electrostatic field.

It was a pleasant surprise to learn that a similar distribution analyzer was developed and successfully used in 1948 in the Melbourne University, Australia. The record shows that our distribution analyzer was first used in June 1947. The efficiency of a distribution analyzer, to a certain limit, increases with the decrease of the width of the individual cell. A micro-distribution analyzer has been designed and constructed by the senior author in the Pennsylvania State College for special research purposes. This new distribution analyzer, consisting of sixty  $\frac{1}{2}$  cm cells, was carefully machined from a piece of 1-in. thick lucite.

The authors wish to express their appreciation to Dr. S. B. Hudson for his interest in our paper and his additional information on electrostatic separation.

<sup>21</sup> Shiou-Chuan Sun: Analyser Aids Electrostatic Research. *Engineering and Mining Journal*, (1949) 150, May, 90.

<sup>22</sup> Council for Scientific and Industrial Research, Ore Dressing Section, Melbourne University Laboratory. R. I. No. 372, 1.

<sup>23</sup> Council for Scientific and Industrial Research, Ore Dressing Section, Melbourne University Laboratory. R. I. No. 372, 11.

O. C. RALSTON, Bureau of Mines, Washington, D. C.;  
F. FRAAS, Bureau of Mines, College Park, Md.; S. B. HUDSON, Melbourne University Ore-Dressing Laboratory, Carlton, Victoria, Australia.

## Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units

by J. J. Bean

### DISCUSSION

**F. M. Lewis**—I believe that density recorders are a prerequisite to all well operated ore concentrators. Well designed density instruments are very accurate and give an excellent record of the steadiness of the operation. Controlling the density of a pulp with an instrument, as described by Mr. Bean, is new and the experiences in developing this controller at the American Cyanamid Co.'s Mineral Dressing Laboratory are very interesting and should be extremely helpful to others who are planning on designing or installing controlling density instruments.

Instruments for recording the density of the classifier overflow or flotation feed are rather common in ore concentrators, but there are a number of profitable uses for these instruments that are not too well known.

Density recorders with bubble tubes submerged to different depth in a classifier pool give an excellent

record of the classifier operation. The loading and unloading of the machine, that is so easily masked in the density of the overflow, can be easily recorded if the instrument is installed at some depth in the classifier pool. On some occasions, two instruments with bubble tubes at different depths give a more candid picture of the irregularities in the classifier operation.

When cleaner machines are operated in closed circuit with roughers, a record of the density of the cleaner tails will indicate the circulating load in the circuit and is very helpful in coordinating different operators, because quite often the circulating load will be increasing for some time before the analyses of the products will change.

Density recorders are used to record the variations in the dilution of the underflow from thickening tanks and in at least one plant the instrument controls the underflow.

F. M. LEWIS, Assistant Superintendent of Mills, Tennessee Copper Co., Copperhill, Tenn.

## Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries

by A. A. Falconer

### DISCUSSION

**D. A. Dahlstrom**—Mr. Falconer has done an admirable job of proving the adaptability of the cyclone to the beneficiation of a very difficult size range in the preparation of coal. The addition of the cyclone to other coarser size methods makes it economically feasible for the coal operator to rigorously control his marketed products at a high and uniform quality level down to the 48 and 100-mesh size. Furthermore, it should be pointed out that by utilizing the cyclone as a deslimer or classifier the -200-mesh fraction, usually containing large percentages of undesirable material, can also be quickly and cheaply removed from the clean coal. Advantages of such a step are several: (1) Reduction of ash content, especially where clays and slimes are serious. (2) lower moisture content of the final coal. (3) easier mechanical and thermal dewatering of the fine coal due to higher cake and bed permeabilities.

The author has indicated that adjustable control means should be installed on the cyclone operating on a heavy-media slurry. This also holds true for the cyclone acting as classifier, deslimer or preliminary dewatering agent and is worthy of special emphasis. Such controls, while not necessarily automatic, must be simple, rapidly adjustable, and require a minimum of attention. Three methods available to the operator are (1) throttling valve on the feed line, (2) back pressure valve on the overflow, and (3) adjustable diameter underflow nozzles. The first two, while fulfilling the specifications, suffer the disadvantage of restricting capacity if severe throttling is necessary. The third method has little effect on capacity and easily maintains an underflow of correct moisture content. Several designs are possible, but two find the widest application. The first was developed by Robert Piros of The Truax-Traer Coal Co. and uses a simple casting containing two holders for underflow nozzles which can be rotated about an axis parallel to the cyclone centerline. These holders retain the underflow nozzle flush with the conical section walls. When a change in nozzle diameter is required, the new size is placed in the spare holder and rotated into place. A second design utilizes a soft rubber cylindrical tube which is compressed or released from three sides, causing the inner diameter of the tube to decrease or increase. This is achieved by using a machine-threaded,



perforated cap on the cyclone underflow pipe which acts on a compression ring to increase or decrease the force on the rubber tube. The cap is fitted with horizontal side arms to facilitate easy adjustment.

D. A. DAHLSTROM, *Northwestern University, Evanston, Ill.*

## Concerning the Adsorption of Dodecylamine on Quartz

by A. M. Gaudin and F. W. Bloecher, Jr.

### DISCUSSION

**G. L. Simard and D. J. Salley**—The authors and ourselves<sup>12</sup> independently came to similar conclusions both as to the value of tracer methods for the study of flotation and the general nature of collector-mineral interaction. The dodecylamine-quartz system appears to be simpler than the dithiophosphate-galena one, and the results therefore appear clearer. It is evident from a comparison of the two papers, however, that care must be used in extending the results to other more complicated systems.

In the case of dodecylamine-quartz, the possible influence of micelle formation is of interest. There is no *a priori* reason why complete monolayer formation should necessarily occur at the concentration for micelle formation, since sorption depends on the nature of the substrate as well as on the properties of the solute in solution which alone determine micelle formation. Indeed, the fact that the onset of increased adsorption, point B in fig. 2, occurs at a concentration one tenth the critical value for micelles might imply that micelle formation does not have an influence. Interestingly enough, the data may be handled by the methods used for multilayer adsorption of gases on surfaces. Thus, if the amount adsorbed is plotted versus concentration, a sigmoid type curve is obtained, similar in form to a so-called Type II isotherm<sup>13</sup> generally attributed to multimolecular adsorption. A monolayer value of about 0.12 mg per g may be estimated from the knee of this isotherm. On the other hand, if a plot is made according to the multilayer gas equation for adsorption to an unrestricted number of layers, eq 38,<sup>13</sup> assuming the critical micelle concentration as the saturation concentration, the data above 120 mg per liter fall on a straight line from which a monolayer adsorption of 0.3 mg per g results. These considerations suggest that the monolayer may occupy an area somewhat less than the geometric value of 0.4 mg per g, and that micelle formation may indeed be a factor in the adsorption.

<sup>12</sup> G. L. Simard, J. Chupak and D. J. Salley: Radiotracer Studies on the Interaction of Dithiophosphate with Galena. *Transactions AIME* (1950) 187, 359; *Mining Engineering* (March 1950) TP 2815.

<sup>13</sup> S. Brunauer: The Adsorption of Gases and Vapors. I, Princeton University Press (1943) 149-162.

G. L. SIMARD and D. J. SALLEY, *American Cyanamid Co., Stamford, Conn.*

## Measurement of Equilibrium Forces Between an Air Bubble and an Attached Solid in Water

by T. M. Morris

### DISCUSSION

**H. H. Kellogg**—There is one point that the author has failed to emphasize sufficiently in his paper.

What is commonly called the equilibrium contact-angle (the author's "maximum contact-angle") can have only one value on a smooth, flat, homogeneous surface under a given set of conditions. The equilibrium contact-angle is defined, for such a system, as

the angle between the solid-liquid and liquid-gas interfaces, measured through the liquid. The value of the equilibrium contact-angle is uniquely determined by the value of the interfacial energies of the three intersecting interfaces and is independent of other forces in the system.

When the liquid-gas interface intersects the solid at an edge—as was the case for all the experiments reported in this paper—the orientation of the solid-liquid interface is indeterminate or varies through 90° for a right-angle edge. Mr. Morris has called the angle between the horizontal and the liquid-gas interface for this edge condition a "static contact-angle." I feel that this term is unnecessarily misleading. In the first place, "static contact-angle" sounds too much like "equilibrium contact-angle." In the second place, the magnitude of the "static contact-angle," which I would prefer to call the "supporting angle," is determined by the forces in the system other than those derived from the interfacial energies, hence "contact angle" is misleading. If Mr. Morris had said that the "supporting-angle" is variable and depends on the weight of the particle and size of the bubble and that it has a maximum possible value equal to the equilibrium contact-angle, his discussion would have been more accurate.

**T. M. Morris** (author's reply)—Mr. Kellogg puts forth a reasonable criticism of some of the terminology used in the paper. I agree that the term "static contact angle" may be misleading. Substitution of the term "supporting angle" or "vector angle" may be more suitable.

**F. X. Tartaron**—In this paper the author presents a very interesting mathematical development of the forces present when a mineral particle adheres to an air bubble. Excellent concordance is obtained between mathematical formulation and experimental results.

It is the writer's understanding that when the mineral surface presented to an air bubble is greater than the area of contact, the maximum contact angle is obtained. However, this contact angle represents distortion of the bubble, the normal shape of which is spherical or in cross-section, circular. This distortion produces a force that acts in opposition to the force of adhesion between the bubble and particle. Hence, at maximum contact angle, the force of adhesion between bubble and particle is at a minimum for static conditions. However, when the size of bubble is increased, the size of particle remaining the same (and all other conditions remaining the same), the contact angle decreases, the distortion of the bubble decreases and the force in opposition to adherence of bubble and particle also decreases. Thus, there is stronger attachment between bubble and particle.

When this situation is applied to actual flotation conditions, it is doubtful that it has any significance. The reason is that the bubbles are normally so much larger than the particles, that in substantially all cases, it is probable that negligible distortion of the bubble takes place. In table IV, the author makes computations for bubbles from 0.50 to 2 mm diam. This is from 0.02 in. to 0.08 in. Certainly the bubbles generated in a flotation machine are far larger than this. The surfaces of the author's bubbles range from 0.79 to 12.6 sq mm. The area of contact of the glass rod (0.15 mm diam) is 0.017 sq mm. Thus, ratio of bubble surface to mineral area of contact ranges from 47 to 741 in round numbers. If we take a 1/4-in. diam bubble and a 65-mesh (0.208 mm) particle of cubical shape, the ratio of bubble surface to mineral area of contact is 2931.

Mineral particles do not readily become attached to air bubbles. Taggart has shown that in pneumatic flotation machines collector-coated particles are only temporarily attached to bubbles. They keep falling off and down in the froth but at a delayed rate as compared with gangue. Spedden and Hannan's motion pictures confirm the difficulty of attaching particles to air bubbles. In the agitation froth process, according to Taggart, air is precipitated from the water selectively on to the collector-coated particles and these



bubble-mineral aggregates coalesce to form the mineral-bearing froth. However, in this case, it is difficult to see how the vast quantity of air utilized can dissolve in the water and be precipitated in so short a time. It appears to the writer that in the agitation froth process, the air precipitated onto collector-coated mineral particles facilitates attachment of "coursing" or nonprecipitated air bubbles to the particle. Coalescence takes place by attachment of particles with a small amount of air precipitated onto their surfaces with large bubbles that are pumped into the pulp by the agitator in the flotation machine. Thus, in the agitation froth process, a relatively small quantity of air would be required for precipitation. The nonprecipitated air pumped into the machine in large volume serves to gather up the particles and provide them with the necessary buoyancy. It would make an interesting investigation to verify that large air bubbles attach readily to particles with minute air bubbles on their surfaces.

**T. M. Morris**—Mr. Tartaron states that when a bubble is attached to a mineral surface of large extent, the force of adhesion is at a minimum when the contact angle is a maximum. Just the opposite is true—the force of adhesion is at a maximum when the maximum contact angle obtains, for a given size of bubble. When the size of bubble increases, other conditions remaining constant, the degree of flexibility increases and hence such a bubble-particle system can better withstand disruptive forces than if the bubble were smaller because the internal pressure of the bubble is less for a large bubble than for a small bubble.

I can't agree that all bubbles in a flotation cell are so large that the influence of the size of bubble is insignificant. Surely  $\frac{1}{4}$ -in. bubbles are not the smallest size present. It is reasonable to expect that there is a wide range of sizes present in a flotation cell. Observation confirms this. The lower size would probably be smaller than 0.50 mm.

I agree with Mr. Tartaron that it would be interesting to study the influence of precipitated bubbles upon the attachment of particles to larger bubbles.

*H. H. KELLOGG, Columbia University, New York;*  
*F. X. TARTARON, Jones and Laughlin Steel Corp., Negaunee, Mich.*

## Effects of Activators and Alizarine Dyes on Soap Flotation of Cassiterite and Fluorite

by R. Schuhmann and B. Prakash

### DISCUSSION

**Maurice Rey**—It may be interesting to note that depressing effects can also be obtained from cyclic compounds other than dyes.

One such compound which is a dispersing agent for carbon, pigments and other compounds is known in France by the trade name "dispergine" and in the



United States by the name of Lomar P W (Jacques Wolf & Co.)

Dr. Rey admits that the compound is not very selective and is similar in its action to starch and dextrin.

*MAURICE REY, School of Mines and Metallurgy, Paris, France.*

## Continuous Countercurrent Decantation Calculations

by T. B. Counselman

### DISCUSSION

**C. G. McLachlan**—In the foregoing paper the author has presented a very neat method for calculating the solution recovery for a countercurrent flowsheet. He has, however, based his calculations, as he states, on the assumption that "the concentration of dissolved value must be exactly the same in the overflow and underflow of any thickener." This assumption, which is usually made regardless of the type of calculation used, is not strictly correct because diffusion of the dissolved gold in the solution associated with the solids as it progresses through the countercurrent system is far from being instantaneous, with the result that on a ton for ton basis, the assay of the solution in the thickener underflow is higher than in the corresponding thickener overflow. An agitator introduced into the flowsheet between thickeners will reduce, but not entirely eliminate, this difference. In practice we have found that the best way to take care of this condition—in addition to the use of one or more intermediate agitators—is to circulate 20 pct or 25 pct more solution than called for by the cheoretical calculation. This presents no difficulty provided that adequate thickener capacity is provided in the original design of the countercurrent circuit.

*C. G. MCLACHLAN, Noranda Mines, Ltd., Noranda, Quebec.*

## Preliminary Report of Massco Circuitron

by A. E. Craig, W. J. Tait, and E. P. McCurdy

### DISCUSSION

**C. M. Marquardt**—The problem of the automatic control of a grinding-classification circuit is not nearly as simple as has been indicated and it cannot be universally solved through the application of this device.

The sound emanating from a ball mill such as the metallic clink of the balls against the liners can at times be misleading. The fact that the sound level due to the balls hitting the liners may be at a minimum does not necessarily mean that the mill is properly loaded. I have in mind an ore we treat that has a soft talcy gangue. The soft fluffy nature of this material makes the sound from the ball mill a most misleading parameter that is worthless as a means for controlling the amount of ore fed to the mill.

In every milling plant there are transportation lags due to the time it takes the ore to travel from the mill bins to the ball mill. In many plants there may be several different transportation lags due to varying distances of bins from ball mill. These transportation lags can most seriously affect the operation of a controller. It has been found that under many circumstances the transportation lags can be so bad as to cause the controller to hunt seriously. Hunting can and in many cases does become so serious as to render automatic control impossible because of the surges it causes. While it is possible to compensate for this type of lag, this apparatus does not lend itself readily to accomplish the required compensation. Further, if there are groups of bins at various distances from the ball mill, compensation for one may not be sufficient or may be too great for another.

Each milling plant presents a separate problem. The apparatus as presently described does not appear sufficiently flexible to accomplish this.

It should be noted that the load on the classifier rakes is determined by measuring the current input to the rake drive motor. Variations in line voltage will



also cause a variation in the current input to the rake drive motor. It has been my experience that in many places the changes in rake load that are used for control purposes are of about the same magnitude as the line voltage fluctuations. Each may cause the same change in current to the rake drive motor. One, however, has nothing whatsoever to do with the load on the rakes. The actual power input of the rakes is a much better measurable parameter for control purposes. Plants subject to much voltage variation must consider this factor.

With some persons working on this problem there is some doubt as to the value of the load on the rakes as a control parameter. Are we interested in the load on the rakes or more particularly are we interested in the kind of job of classification being done? There is no doubt that the man on the rakes has some effect on the kind of classification job being done. I seriously doubt that it is a cure-all. Further, where the mineral content of the ore varies over wide limits, the load on the rakes can be misleading, even useless, as a control parameter.

As the ore fed to the ball mill is varied, the water added to the scoop box should also be varied. For proper operation of this device another control on the water line operating in parallel with the belt control should be used to vary the water with the ore.

**A. E. Craig** (authors' reply)—The comments offered by Mr. Marquardt are very interesting and bring up several points which have been included in the development of the Massco Circuitron. Certainly it was not our intention to suggest that the control of the grinding circuit was in any way simple as it has involved several years of effort on our part. We agree also it probably cannot be universally used, but we maintain that there are a great number of instances in which the device can pay for itself in added tonnage and improved metallurgical results.

In installing the Circuitron, the grinding circuit is set to optimum results as near as can be determined by manual operations. The instrument is then tuned in, in a manner similar to that of tuning in a living room radio, to select the combination of classifier load reaction and mill sound which gives the best results in tonnage and grind. It took a long time and a serious effort to control this selection so that we can now definitely pick up the desired sound, eliminating such extraneous sounds as that produced by the gear and other machines away from the ball mill. In the same manner we select just the range of sound which is needed for best control. In this way we make compensation for the difference in the character of the ore, whether soft or hard, whether fluffy or otherwise.

In a direct closed circuit with the ball mill and classifier, the transport lag does not appear to be great enough for serious consideration. This condition might be troublesome in an installation dealing with two-stage grinding running the rod mill open circuit followed by the ball mill and classifier circuit. We have not yet reached this stage, but it is our opinion that the device can be adjusted to give satisfactory performance although some additional engineering will be necessary. We have had no noticeable difficulty due to holding or, on the other hand, from heavy surges. The Circuitron maintains control, changing as required by the feed changes.

The question of flexibility is largely answered by our combination method of tuning the sound circuit in connection with the adjustment of the classifier power current to the greatest advantage.

The question is raised that variation in line voltage on the classifier motor circuit might cause enough variation in the current delivered to the Circuitron so as to upset our control. As a matter of fact, in the controller circuit, the measure of the classifier power is made by a modified watt-hour meter, using a current transformer for the reduction of the classifier current to any desirable value. The actual current delivered to our instrument is small and we have no difficulty at

this particular point. We have had a great deal of difficulty with voltage fluctuations within the control itself so that it has become necessary to use a voltage regulator to insure a constant potential of 115 v in the controller circuit.

We are indeed interested primarily in the work of the classifier, measured by the characteristics of its overflow product, rather than in the amount of sand load, although we do emphasize the fact that the sand load is one point of control in the Circuitron. The work of the classifier is maintained at a high efficiency level by adding the Massco density controller to maintain a constant density in the overflow. This results in good classifier operation on the various tonnages as the density controller is set to hold the density at a predetermined point. The last point mentioned by Mr. Marquardt is controlling the amount of water added to the scoop box. We originally figured that this amount was so small relatively that it needed no control. Later results indicate that this may be an important point and, if so, we plan to make provision for this additional regulation.

*C. M. MARQUARDT, Combined Metals Reduction Co., Salt Lake City, Utah.*

## An Improved Method of Gravity Concentration in the Fine-Size Range

by Arvid Thunaes and H. Rush Spedden

### DISCUSSION

**R. R. Knobler and F. E. Albertson**—Following the testwork done by Thunaes and Spedden, a Sullivan deck plant was built for the Colquiri mill. This plant started to operate in April 1945 and continues in successful operation.

Since May 1950 the Colquiri mill has operated with a completely revamped flowsheet at a rate of 1000 tons per day. The old Sullivan deck section was retained with this new flowsheet.

The "slime plant" in Colquiri consists of one 12-cell Denver No. 18-Sp. Sub-A flotation machine, one 16-ft and one 12-ft hydroseparator, six five-deck Sullivan-frames and six shaking tables for cleaning up the Sullivan deck concentrates, as well as 12 Deister plus three Plat-O tables on which the fine sands, which are not treated on the S-decks, are concentrated.

Only —325-mesh fines are treated on the Sullivan decks. The combined hydroseparator overflows are fed to the S-decks.

From 1946 to 1949, i.e. with the "old" flowsheet and an average mill feed of 800 tons per day an average of 45 tons per day was treated on four Sullivan rougher frames. The two additional frames were used as cleaners. At times, up to 90 and 120 tons of slimes were treated successfully on this Sullivan deck installation.

The average operating data for the S-deck section in Colquiri for the year 1949 were as follows:

Assay Sn, Pb	
Feed to S-decks	2.13
Tails from S-decks	1.03
Table cleaner concentrates	24.4

The tests reported in this paper indicated a tin recovery of 42 pct. The actual plant results for 1949 show a 54 pct tin recovery from the Sullivan decks.

The total average quantity of tin produced from the S-deck section in 1949 was 15 tons of tin per month. This corresponds to 3 pct of the total mill recovery.

The difference between the 6 pct overall recovery-increase expected after the tests in 1943 and the actual plant results given for 1949 is explained as follows. According to the tests, a total of 170 tons was expected to be routed to the Sullivan decks, but because of certain changes in the flowsheet and mill amplifications,



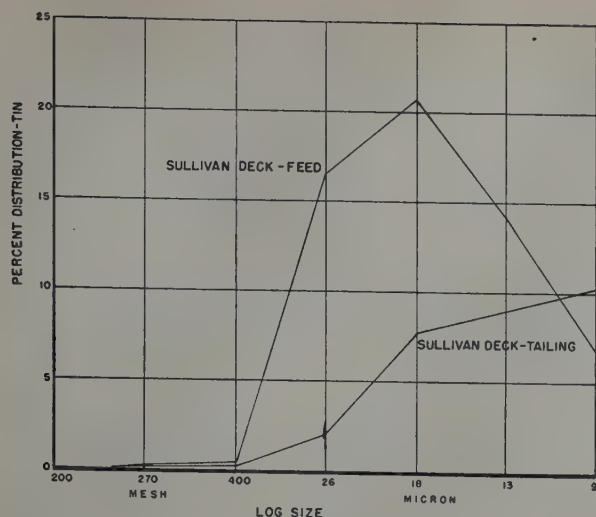


Fig. 5—Percent tin distribution in Sullivan deck feed and tailing, Colquiri ore.

only about one fourth of this tonnage was actually treated on the decks.

With the new flowsheet a higher proportion of fines is produced and more feed is being routed now to the Sullivan section.

The tin distribution of the Sullivan-deck feed and the Sullivan-deck tails is given in fig. 5. These analyses were made on products from the pilot tests for the design of the new flowsheet. They show the high concentrating efficiency of the Sullivan decks down to about 9 micron particles of heavy minerals.

Fig. 6 gives the tin distribution of the corresponding concentrates. These concentrates showed the following assays:

	Assay Sn, Pct
S-deck concentrate	3.2
Cleaner table high-grade conc.	45.6
Cleaner table low-grade conc.	13.2

The Sullivan deck operation is extremely simple and requires very little maintenance and operating expense.

The operation of the whole slime plant in Colquiri, including flotation and fine sand tabling, cost \$0.07 per ton milled.

In the year 1949 only \$317.00 was spent for spare parts and material for the Sullivan decks in Colquiri with a total of 268,600 tons milled.

Although round tables of the Anaconda type recover the same sizes of fine cassiterite as the S-decks, the cheapness and ease of operation of the decks make this machine far superior.

Recently laboratory and mill tests were made with the Colquiri ore to determine the advantage of dispersing the feed to the slime concentration. The results indicate that deflocculation with sodium silicate is helpful for obtaining better hydraulic classification and for obtaining better results in the shaking table cleaning of the S-deck concentrate.

It is planned to use dispersion with sodium silicate in the Colquiri mill for preparing the S-deck concentrate for the table cleaning.

The recovery in this table operation could be increased from 42.9 pct to 63.3 pct by treating deflocculated pulp; at the same time a higher grade concentrate is obtained (21.0 pct Sn against 16.5 pct Sn).

By retabbling, these concentrates are graded up to over 40 pct Sn.

The use of pine oil was also found of some advantage on the feed to the Sullivan decks. It works in the way

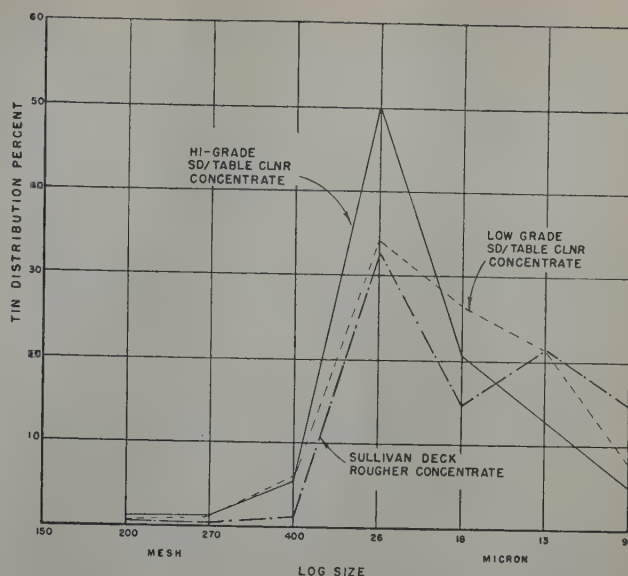


Fig. 6—Tin distribution in Sullivan deck rougher and table cleaner concentrates.

indicated by Thunaes and Spedden and lowers somewhat the tin content of the S-deck tails.

At present tests are being made in our experimental station and in the mill trying to use the Dutch State Mines (D.S.M.) cyclone as classifier and deslimmer for the slime plant in Colquiri.

Such an application of the centrifugal cone was suggested by Professor Spedden to one of us early this year.

The results obtained so far indicate that it will be possible to make the sand-slime separation at about 270/325-mesh with two D.S.M. cones working in series. This will make it possible to route only fine material to the hydroseparators and to make their operation considerably simpler. It is planned to discard then the finest slimes (after dispersion) with the hydroseparator overflow which will reduce the tonnage to the S-decks. Treating more granular material will make the deck concentration also more efficient. Instead of the hydroseparator overflows, as at present, their underflows will be treated on the S-decks.

#### Tungsten Slime Concentration

In 1945 we made tests on the slime tailings from the Bolsa Negra tungsten mill in Bolivia, which is also operated by M. Hochschild, S.A.M.I.

This mine contains wolframite plus scheelite in a proportion of about 9 : 1.

On a slime table tailing of 1.94 pct WO<sub>3</sub> a Sullivan-deck preconcentrate of 7.65 pct WO<sub>3</sub> was made which contained 49.6 pct of the tungsten formerly lost in this tailing.

Of the tungsten fed to the S-deck, 67.6 pct was +9 micron so that a recovery of about 70 pct on the —270 +1500-mesh wolframite is indicated. The S-deck tail assayed 1.15 pct WO<sub>3</sub> and contained mostly scheelite slimes —9 micron.

Because of the high tourmaline content in this ore it was difficult to grade the 7.6 pct WO<sub>3</sub> preconcentrate up to over 29 pct WO<sub>3</sub>.

The Bolsa Negra mill was closed shortly after these initial S-deck tests were made. Under present conditions operations at this mine might be resumed and it is planned to include a Sullivan-deck section in the new mill.

R. R. KNOBLER and F. E. ALBERTSON, Mauricio Hochschild S. A. M. I., Oruro, Bolivia.

## Coal Preparation for Synthetic Liquid Fuels

by W. L. Crentz, J. D. Doherty, and E. E. Donath

### DISCUSSION

**Aureal T. Cross**—This paper, which treats certain phases of the research and development of newer uses for coal, serves to emphasize the types of problems which will need to be solved and can be solved better by coordinated work of chemists and engineers with coal geologists and paleobotanists. It serves as evidence for the need and value of coal petrographic studies, paleobotanical investigations into the origin, distribution and alteration of coal, and for coal geochemical research.

It is necessary to accept the fact that coal is no longer to be viewed as a finished product as mined but as a raw material to be prepared in various ways for wise and economically sound use. We must anticipate and promote new methods of utilization, some of which may seem fantastic or wishful today. It is true that many of the more important methods to which coal is put to use today will be with us for some time, i.e., coke making, steam, heating, etc. But newer uses will probably eventually consume as much or more coal than our present economy and technologic development allows. We have really never gone far from the "cream of the crop" coal seams to supply our needs. It should not surprise us when coal technologists develop methods of utilizing even low rank, high ash coals, or such cast-off materials as the inferior "middling" products of the hydrogenation process, and even bituminous beds of very low Btu rating. We should look forward to the day when methods of utilization are so developed that each coal type can and will be put to the wisest use in order to conserve our resources. We should begin now to consider all bituminous beds of any sort as so many units of energy, or by some other such common denominator, rather than as tons of coal or gallons of fuel. We might then say "so many energy units are available in a certain form, can best be utilized for a particular job, and, by certain preparation methods at costs previously determined in laboratories and pilot plants, can compete favorably physically and economically in a certain job with other types of energy-bearing resources.

The solution of these problems hinges in part on the availability of an advance guard of coal geologists trained to observe and interpret the character of the coal components and their effects upon, reaction to, and role in chemical and physical reactions.

Best uses of the various types of coals are generally predictable by the application of various techniques of the paleobotanist and coal petrographer with reasonable accuracy at costs probably considerably lower than awkward attempts to work out some of the more basic information by chemists. These same researchers also must know their own limitations and turn over to proper specialists problems beyond their fields.

Typical problems which can undoubtedly be solved by coordinated efforts of coal geologist, petrographers, paleobotanists, coal geochemists and others are reduction of even greater amounts of sulphur both mineral and organic, a method for utilization of "fines" or even impalpable coal dust, which are now essentially waste products, assistance in roof support problems, more accurate determination of reserves, and continuity of beds and general correlation.

We should broaden our horizons but we must have the research men to pave the way. They are generally unavailable today. The fuel technologists need the basic and fundamental information that is designated by some at the present time as irrelevant, uneconomical and of academic value only. But if we insist on binding ourselves to present technologies, coal geologists of most sorts will have limited usefulness. Advance designs of old machines, combustion chambers and power converters should not furnish limiting boundaries of our vision in fuel technology.

AUREAL T. CROSS, *West Virginia Geological Survey, Morgantown, West Va.*

## Cyclone Thickener Applications in the Coal Industry

by M. G. Driessen and H. E. Criner

### DISCUSSION

**Maurice Rey**—The influence of cyclone diameter upon the fineness of separation is an important point which, however, cannot be discussed adequately if the injection pressure or the rate of flow are not specified.

Perhaps some light can be thrown on the matter by using the formulas derived by Dahlstrom.<sup>5</sup>

Let us consider several geometrically similar cyclones where the apertures  $e$  and  $b$  are equal and proportional to the diameter  $D$  and which are operated with the same pressure drop.

Eq 8 of Dahlstrom's paper can be written:

$$\frac{Q}{\sqrt{F}} = K D^{1.8}$$

or, if  $F$  is constant:

$$Q = K D^{1.8}$$

$K$  is a constant of proportionality.

Introducing this value of  $Q$  in eq 13 gives elimination efficiency =  $K D^{0.41}$

Elimination efficiency, in the terminology of Dahlstrom, means the diameter of particles, in microns, which divide equally between the underflow and the overflow. This denomination is regrettable and it would be preferable to call it diameter of separation or characteristic size or any other name connoting a size rather than an efficiency.

Whatever the name used, the above equation shows a slow increase in the fineness of separation when the diameter of the cyclone decreases.

For example, if a 14-in. cyclone gives a 400-mesh separation (37 microns), geometrically similar cyclones, having the same pressure drop will give separations at the sizes shown below.

Cyclone Diameter, in.	Diameter of Separation, Microns
14	37
8	29
6	26
4	22
3	20

Now, it seems to me that the curves of fig. 3 of Driessen show a stronger influence of the cyclone di-



ameter than the one computed above. However, the pressure drops of the various cyclones studied are not given so that it is not possible to reach definite conclusions.

**H. E. Criner** (authors' reply)—Mr. Rey's observation as to the rate of increase of particle size with cyclone dimension is not quite correct since the experimental data from which the exponent 0.41 was secured was not from geometrically similar cyclones but from cyclones generally having the same body diameter with varying feed and overflow opening diameters. However, the exponent is close to being correct. We believe it to be 0.5.

The theoretical relationship between particle diameter and cyclone dimensions is:

$$Dp^2 = K \frac{Aj^2}{l} \sqrt{\frac{Cf}{P}}$$

where  $Aj$  is the area of the feed nozzle,  $l$  is the cyclone core length from overflow tube to cone apex.  $Cf$  is the

flow coefficient and  $P$  is the inlet pressure.

The curve referred to by Mr. Rey was meant, unfortunately, to show the trend or the separation size with respect to cyclone diameter rather than an exact relationship between them. The 1½ and 2 13/16-in. cyclone were similar, the 8 in. had a 14° cone angle and the 14 in. a 20° angle so that the cyclones were not similar to each other or the two smaller ones. Furthermore, the results on the 14-in. cone were secured with a higher feed concentration than that used to test the smaller cones. All tests were run at 40 psi inlet pressure.

We are sorry that we cannot furnish Mr. Rey with data from a closely controlled test, however, from the data available to us; after correction for inlet pressure, feed concentration and flow ratio; we have concluded that the particle size varies as the ½ power of the cyclone diameter.

*MAURICE REY, School of Mines and Metallurgy, Paris, France.*

## Kerosine Flotation of Bituminous Fine Coal

by L. E. Schiffman

### DISCUSSION

**W. J. Parton**—Those operators faced with the problem of treating fine coal whether in bituminous or anthracite will find this paper most timely.

I would like to take this opportunity of discussing Mr. Schiffman's paper and at the same time express certain views relating to our Tamaqua plant.

I would like to ask the author what type of impeller and diffuser is used in the Denver cells?

Screen analysis of products from individual cells indicate that coarser material resists flotation and only floats after greater retention time in the last few cells. Also, the need for a scavenger screen to reclaim non-floated coal particles further stresses this point. I have always felt that more efficient means of cleaning coal between 10-mesh and 28-mesh existed than flotation. Reagent and power costs are high for the flotation process.

When floating +28-mesh particles, cell capacity is lowered and some of the particles are lost with the refuse.

The Tamaqua plant of the Lehigh Navigation Coal Co. floats —28-mesh coal and capacity of recoverable coal is 40 tph for 1800 cu ft of Denver cells; or 0.05 tons per cu ft of cell.

At Kimberly 7.75 tph for 600 cu ft of cell gives 0.013 ton per cu ft of cell.

At Bessie 14 tph for 800 cu ft of cell gives 0.017 tons per cu ft of cell.

It would be appreciated if the author would comment on what he feels is the upper size limit of particle to attain most efficient utilization of the flotation process.

The dewatering screw is a very interesting development since it offers a simple way to prepare coal sludge for more complete dewatering by drainage or mechanical dewatering on screen or filters. In other words it could be used to accomplish the same thing as a thickener tank. I would appreciate having the author's comment on how he thinks such a screw dewaterer would work on a froth.\*

The process as used in floating coal at the Bessie and Kimberly plants may be referred to as more of a bulk oil float in contrast to a froth flotation process.

Experiments on increasing capacity of cells is most interesting since we are going through such an experi-

mental period at the present time. Recently a double overflow was installed on our No. 30 Denver cells. So far results are not conclusive.

In reviewing this paper the following comments are made pertaining to investigation of methods for increasing capacity:

**Supercharging:** Supercharged air in matte flotation or for that matter the use of the normal amount of air drawn in by the impeller would in all probability cause such an aeration in the cell as to destroy the buoyant effect given to the coal particles by the excessive amount of kerosine used.

In other words, air creates an agitation zone throughout the cell, creating a boiling and thereby giving a lower recovery in the cell. It would be interesting to know whether the 7 pct increase in recovery was with no air being admitted to the stand pipe.

**Changing Impeller Speed:** The speed of a receded disc impeller for a No. 30 cell as recommended by the Denver Equipment Co. is, I believe, approximately 250 rpm. At this speed and using supercharged air in excess of 8-oz pressure, we have observed a boiling action in the cells. In our flotation we endeavor to obtain some degree of froth flotation using pine oil as a frother. The boiling action as caused by increasing the amount of air added to the cells is detrimental to recovery in froth flotation.

It is our belief that to obtain increased recovery from a cell in froth flotation, additional air must be introduced but at the same time this air must be dispersed throughout the pulp in the form of small bubbles and this can only be done by increasing the speed of the impeller.

Therefore, if Mr. Schiffman decreased the speed of the No. 30 impellers and at the same time continued to use supercharged air, the boiling action may have been increased because larger bubbles developed. The lower recovery as reported could be due to this factor.

Decreasing the impeller speed will definitely decrease the power consumed but may have other disadvantages. First, we believe it will permit "sanding" in the cell and this in our opinion will increase the wear on the impeller and diffuser, especially so, if there is pyrite and/or sand present in the feed.

"Sanding" in the cell when air is used, as in froth flotation, will effect the dispersion of this air and cause boiling.

\* Concentrate rather than a matte concentrate.

When one considers the number of particles of coal to be floated and the maximum load each bubble of air can lift, it would seem to be imperative that greater increase in the number of bubbles is required. In matte flotation the more coal to be floated, the more kerosine required, which accounts for the high reagent consumption as compared to our Tamaqua plant which uses about 0.9 lb fuel oil per ton feed.

As stated previously, the use of double overflows on the No. 30 cells at our Tamaqua plant are being investigated. To date the increase or expected increase in capacity has been disappointingly low. Why? It is our belief that employing an impeller speed of 250 rpm as recommended, the amount of air being introduced is being fully utilized, the same as it was with a single overflow, and therefore no additional recovery can be obtained with the double overflow until we increase the impeller speed to allow more air to be introduced into the cell in the dispersed state.

There is also the possibility that when the cell volume was increased, the agitation was decreased, which would permit "sanding" and help to destroy the good flotation conditions which previously existed.

It is interesting to note that in the flotation of phosphate in Florida, our plant reports doubling the capacity of their cells using more air and increased impeller speed. They also do not have as many particles to float per unit volume of pulp.

**Increased Area and Coal Removal.** When the effective area of a cell is increased, the use of rakes, especially the use of split rakes in place of a double spitz, for matte flotation of coal or the use of double overflow paddles in froth flotation, to effect an increase in the rate of coal removal (increased capacity) would seem to be a logical conclusion providing more coal is raised to the surface by changing conditions in the cell proper.

**Conditioning.** Conditioning prior to flotation would in all probability decrease the retention time at each of these plants and for matte flotation there is a possibility some saving of kerosine would result if conditioning were done at a higher pulp density. The flotation cells are conditioners. Lowering the impeller speed to 207 rpm at the Kimberly plant as reported by the author resulted in the first cell only floating 80 pct of its normal amount and the second cell only 90 pct. This could be attributed to a decreased conditioning action. Insufficient conditioning in the cells could result in a loss of kerosine with the coal product with a resulting increased loss of coal in the refuse.

In summary it is our belief that many factors enter into the problem of increasing the capacity of a No. 30 Denver flotation cell.

**L. E. Schiffman** (author's reply)—The impellers used in the Denver cells are the receded disc type.

Our experience would indicate that 20-mesh is probably the top particle size for the most efficient utilization of the flotation process, but that the top size can be raised to 14-mesh with but little impairment in efficiency. Above 14-mesh there may be considerable impairment, but the need for cleaning coarser than 14-mesh, the simplicity of the kerosine process, and the avoidance of an additional cleaning method led to the selection of 10-mesh as the proper size for Kimberly.

It is true that our floated coal capacity expressed in tons per hour per cubic foot of cell volume seems low in comparison with other flotation processes. According to Mr. Parton's figures the float at Kimberly thus expressed is but 26 pct of that obtained at Tamaqua, and at Bessie but 35 pct. We have always attributed our lower results to the lighter gravity of the material floated. If the average gravity of the anthracite float at Tamaqua is assumed to be 1.60 and that of our bituminous float 1.32, then on the basis of actual volume of the solids floated, Kimberly becomes 31 pct of Tamaqua and Bessie 42 pct, which still leaves much to be desired in the way of capacity.

The test with the speed of the impellers decreased

was made without supercharging. We have not used supercharging at either plant except for a short time at Bessie when that installation first went into service.

The test made by reducing air to the cell impeller was made with the air intake completely closed, but there is an annular space between the outer stand pipe and the impeller drive through which air is drawn. Closing the air intake reduced the area through which air was drawn by approximately 75 pct.

W. J. PARTON, *Lehigh Navigation Coal Co. Inc., Lansford, Pa.*

## Laboratory Performance Tests of the Humphreys Spiral As a Cleaner of Fine Coal

by M. R. Geer, H. F. Yancey, C. L. Allyn, and R. H. Eckhouse

### DISCUSSION

**W. M. Bertholf**—This is an excellent report of a well-conducted investigation, of sufficient scope to provide generally useful information.

Some years ago we had occasion to test the Humphreys spiral on the middling from our table plant. The conclusions reached in this investigation come as no surprise to us. In certain applications it is virtually impossible to improve on the spiral from the monetary returns point of view even though its "efficiency" may be low.

We have noted that in all successful applications of the spiral the feed is "classified," hence we were somewhat disappointed in the showing the spiral made on the Poplar Ridge coal, the only one of the four which could really be considered the equivalent of a classified feed. As was noted in the paper, however, this coal contained a considerable amount of middle and high-gravity material, apparently more than enough to make up for the theoretical advantage of small particles in these categories.

A comparison of particle size in the various specific gravity classes of the four coals is given in table XXV.

**Table XXV. Comparison of Particle Size in Various Specific Gravity Classes. (Unit size, 150 m to 0.105 mm)**

	Black Eagle	Clements	Poplar Ridge	Roslyn
1.3 Float	7.47	10.60	8.68	9.66
1.3 x 1.4	7.89	7.96	7.58	8.20
1.4 x 1.6	7.24	8.36	5.58	6.85
1.6 x 1.8	7.98	8.31	3.42	7.33
Sink 1.8	7.85	7.13	5.09	7.41
Avg	7.69	10.08	7.15	8.38

This tabulation indicates that there was very little difference in the size of the various gravity fractions for three of the coals, Poplar Ridge varying the most. Examination of figs. 3, 6, 7, and 8 reveals a general similarity of behavior for all coals except Black Eagle. Would the authors care to speculate on the reasons for the difference in the distribution curve for the sink 1.60 material in the washed coal of fig. 3 as compared to the others? This difference is particularly noticeable in the 8x14-mesh size. Was there a "shape effect"?

**J. D. Price and W. M. Bertholf**—The performance data on three of the coals studied appear to be typical, but there are certain departures from the general pattern in the case of the Poplar Ridge coal which are large enough to deserve special consideration. For example:

1. Despite the high proportion of sink 1.60 material in the feed, it was not possible to make a corresponding proportion of high-ash reject. (See table XVI.)

2. Even though approximately 70 pct of the feed was 1.40 float with an ash content of 6.6 pct, and this



**Table XXVI. Comparison of Poplar Ridge and Roslyn Coal with Anthracite**

	Ash, Pct			Total Size, Pct		
	Poplar Ridge	Roslyn	Anthracite	Poplar Ridge	Roslyn	Anthracite
8 x 14-mesh	12.1	20.1	22.9	21.6	31.3	17.9
100 x 200-mesh	22.2	24.1	36.0	5.3	7.7	10.9
Under 200	27.1	27.8	54.3	9.5	4.6	13.8

**Table XXVII**

Mesh	Feed to Spiral	Clean Coal from Spiral	Mesh	Feed to Spiral	Clean Coal from Spiral
	Ash, Pct	Ash, Pct		Ash, Pct	Ash, Pct
3/32	14.0	10.5	60	29.7	11.3
3/64	21.2	8.3	80	30.4	14.8
28	22.6	8.5	100	30.5	22.3
35	22.9	8.5	200	43.3	33.4
48	26.0	9.9	—200	55.1	57.5

material was considerably larger than the material of higher specific gravity, the "washed coal" in the outer zones contained 16 pct ash. (See table XVII.)

3. The distribution patterns shown in fig. 7 are considerably different for washed coal and refuse from the corresponding curves for the other three coals.

All three of these "phenomena" point to some peculiarity in the nature of the high-gravity material present in this coal. Could this be particle shape?

Cubical particles would be expected to report farther out in the stream than flat particles retained on the same screen due to the increased centrifugal force and decreased drag. Could it be that the Poplar Ridge "coal" was rather slabby and the "rock" was roughly cubical? Some such feature as this, plus the large proportion of high-gravity material might account for a great deal of the abnormality of the final data.

**J. F. McLaughlin and H. H. Otto**—We wish to compliment the authors on their studies and splendid presentation of their findings. We note that two of the four coals tested were, as we view it in the anthracite area, comparatively low-ash coals and therefore not a difficult cleaning job. The other two coals, Poplar Ridge of Kentucky with 17.2 pct ash, and the Roslyn coal of Washington with 21.4 pct ash, are more like our anthracite coal cleaning problem for total ash but differ considerably by sizes as shown in table XXVI.

Our preparation plants prepare coal from various sources and numerous beds, varying somewhat in specific gravity, inherent ash and other impurities such as bone, slate, and roof rock. To successfully meet these variations and not suffer too great a loss of good material is a problem at each cleaning plant. The Hudson Coal Co. has obtained very good results on finer sizes of coal with Wilmot hydrotators and classifiers down to and including No. 5 buckwheat. The size range is No. 4 buck 3/32 x 3/64 and No. 5 buck 3/64 x 0.

The efficient classifier limit on No. 5 buck seems to be somewhere between 48 and 60-mesh. The Humphreys spirals were installed anticipating economical recoveries below this range. We find the best range for the spirals is between 3/32 and 80-mesh. Cleaning falls off at the 80 and 100-mesh, with no cleaning on —200-mesh, and usually the 150 to —200-mesh material is so high in refuse that it is uneconomical to attempt to recover it.

We agree, from our plant experience, that in the spiral as in the classifier, there is a point where particle size is very important and apparently interferes seriously with the gravity action of the cleaning equipment.

In the summary (p. 1057) the statement, "The coarsest fractions of impurities stratified so far out in the stream that it could not be removed through the refuse ports," has not proved true in our active experience as is shown in table XXVII.

Cleaning starts to fall off at 80-mesh.

"Impurities finer than 100-mesh carried out in main body of the stream and therefore were not removed in the refuse."

This is true in our coal also and, in fact, is true where coal is classified or separated by water. Currents of water cannot be adjusted to suit a range of size from 3/32 to 0. Up to a certain mesh the current is working on the difference in specific gravity of coal and refuse, then particle size takes over and floats off low and high ash particles, and, to control the total ash in the product to market requirements, the fines have to be removed.

We wish to clarify a preceding statement. The Hudson Coal Co., in practically all of its cleaning plants, has Chance process using sand as a separating medium and at Powderly the raw silt feed (3/32 x 0) is sent to the Dorr thickener and thence to the spiral plant or to the silt basin and contains fine sand. That is one of the reasons for the 80 pct ash in the refuse in this case. Powderly spiral plant prepares boiler fuel for one of our power plants, which can burn these fine sizes successfully with 16 pct to 20 pct ash. Our Loree spiral plant is for outside customers and at the moment we are doing a cleaning job with 10.5 pct to 12.0 pct of ash in the prepared coal. We have done as low as 9.9 pct ash in the coal. The refuse ash range is from 48.0 pct to 55.0 pct. We recirculate the middling product from the last turn of the spiral and we have supplemented the spiral with a launder screen 15 ft x 28 in. and clothed with 30x30-mesh screen to drop out the very fine material which is high in ash. The screen is located between the spirals and a coal tank, from which the cleaned product is pumped to settling basins. At Powderly we use a settling tank with slow-moving conveyor on the finished product which, in the overflow, carries a lot of very fine undesirable material to waste.

When only one type of coal in a confined range of size is cleaned, the equipment, whether a classifier or spiral, can be adjusted to give optimum results. This, however, is a difficult thing with us where our feed comes in part from strippings carrying clay and the bulk of it from at least four to six different seams of coal and from scattered openings.

The coal industry and equipment makers are studying the fine coal cleaning problem carefully and have made considerable advance. We have not entirely solved the problem of cleaning 80 to 100 and 200-mesh material. The final answer may be that the cost will not warrant the recovery of these sizes.

**Whitman E. Brown**—Everybody familiar with coal cleaning knows that whenever a device appears on the market with any merits it will only be a short time before Dr. Yancey and Mr. Geer will have some good concrete data available.

During one phase of their investigations, the authors asked us, as manufacturers, to observe the testing and offer advice for possible improvement of procedure. We were unable to offer much for improvement although we were able to show ways of obtaining more information. It is our opinion that on the samples tested the optimum results were obtained by use of the spiral alone. We all know that the response to cleaning will vary with the material. Often one machine alone cannot accomplish the full job. In the past we have been able to do much better cleaning jobs on some coals than mentioned in this paper and conversely on some fine coals we could do practically nothing.

Recently a —4-mesh coal was tested from the West Virginia, Island Creek seam. The feed was 11 pct ash raw coal and the feed rate was 20 gpm at 26.7 pct solids. The results were: (1) clean coal ash, 6.5 pct; wt, 69.1 pct. (2) refuse ash, 50.6 pct; wt, 5.7 pct, (3) middling ash, 14.7 pct; wt, 25.2 pct.

The middling was removed as a finished spiral product and fed to a sizer which produced: coal ash, 8.1 pct; wt, 79.1 pct and refuse ash, 44.0 pct; wt, 20.9 pct.

Combining the two coal and two refuse products shows: finished coal ash, 7.2 pct; wt, 89.0 pct and refuse ash, 47.3 pct; wt, 11.0 pct.

This information is submitted solely to show that the spiral-sizer combination is another effective method in attaining higher recovery at reasonable grade. Reasonable grade is a relative expression and open to discussion.

I quite agree with Dr. Yancey and Mr. Geer that a classified feed is often more desirable than a raw coal.

It was further pointed out that laboratory tests provide little or no information bearing on costs which enter into the selection of cleaning equipment. Assuming that there is a market for the product and a given grade can be maintained the next important thing is, "How much is it going to cost to obtain this product by using this equipment?"

Estimated operating costs involved in a spiral plant are as follows:

Item	Cost per Ton 2000 Lb Ton
1. Operating Labor (\$1.10 per hr)	1.50
2. Power Cost, Pumping	1.30
3. Pumping Maintenance	0.35
4. Feed Distribution Maintenance	0.40
5. Product Launder Maintenance	0.04
6. Spiral Maintenance	0.50
<b>Total Direct Spiral Plant Cost</b>	<b>4.09</b>
7. Screen Maintenance	0.10
8. Dewatering Maintenance	0.10
9. Conveying Maintenance	0.10
10. Fresh Water	0.05
11. Heating (Buildings)	0.02
12. Overhead	0.10
13. Assaying and Plant Control	0.30
14. Hydraulic Sizing	0.60
15. Amortization (10 years)	2.00
<b>Total Plant Cost per Ton</b>	<b>7.46</b>

Installation cost of the equipment is approximately \$1000 per ton per hour. This figure is based on cost of spirals, pumps, tanks, chutes, conveyors, piping, screens, motors, controls and wiring. By judicious planning and working under favorable conditions the installation cost may drop to \$800 per ton per hour.

**M. R. Geer, H. F. Yancey, and C. L. Allyn** (authors' reply)—The observation by Price and Bertholf that the Poplar Ridge coal behaved differently in the spiral than the other three coals tested is entirely correct. The explanation for the different performance on Poplar Ridge coal lies largely in the size composition of the heavy impurity. As shown in table XXVIII the tabulation of the size composition of the impurities in the four coals, the impurity heavier than 1.60 sp gr in the Poplar Ridge coal was much finer than that in the other three coals. Over 36 pct was finer than 100-mesh, and since impurity of this fineness cannot be stratified in the spiral, most of it enters the clean coal or mid-dling products.

**Table XXVIII. Screen Analyses of Impurity Heavier Than 1.60 Sp Gr in Coals Tested in Spiral, Pct**

Coal	Screen Size, Mesh					
	8 to 14	14 to 28	28 to 48	48 to 100	100 to 200	Under 200
Poplar Ridge	10.2	29.6	22.3	1.7	9.3	26.9
Clements	26.8	23.1	18.7	15.6	9.4	6.4
Black Eagle	31.2	24.0	15.3	12.3	7.4	9.8
Roslyn No. 3	26.3	27.7	13.6	14.2	10.3	7.9

A second factor that caused the distribution curves for the Poplar Ridge coal to differ from those shown for the other coals is the refuse-port settings employed. Tests made on this coal with a wider refuse cut gave distribution curves which were much more similar to those for the other coals, but at a distinct sacrifice in the yield of clean coal.

Although particle shape necessarily influences the performance of the spiral, just as it modifies the effect

of particle size and specific gravity in any coal-cleaning device utilizing currents of fluid, no attempt was made in this investigation to evaluate the effect of shape. It is entirely possible that, as pointed out by Messrs. Price and Bertholf, particle shape may have been responsible for some of the differences in the performance observed.

The observation by Messrs. McLaughlin and Otto that in treating anthracite in the spiral the coarsest fraction of heavy impurity does not stratify so far out in the stream that it contaminates the washed coal is interesting. The reason why anthracite should behave differently from bituminous coals in this respect is not clear, but presumably either particle shape or the relative difference in the specific gravity of the clean coal and impurity must be responsible.

*J. D. PRICE and W. M. BERTHOLF, Colorado Fuel and Iron Corp., Pueblo, Colo.; H. H. OTTO and J. F. MCLAUGHLIN, Hudson Coal Co., Scranton, Pa.; and W. E. BROWN, Humphreys Investment Co., Denver, Colo.*

## Quantitative Efficiency of Separation of Coal Cleaning Equipment

by W. W. Anderson

### DISCUSSION

**John Griffen**—The author has called attention to an important phase of coal cleaning since there has been considerable loose thinking regarding efficiency formulae and particularly inaccurate application of ones that have been advanced.

The unanimous acceptance of an efficiency formula has been prevented because the issue has been beclouded by the fact that the objective of cleaning—the control of chemical characteristics of the coal such as ash or sulphur content—is attained by an indirect rather than a direct method. The indirect method is usually to effect separation according to differences in specific gravity. Such separation relies on the fact that the desired chemical qualities are roughly inversely proportional to the specific gravity of the particles being separated. However, the sulphur content of the materials found in raw coal usually exhibit a far from straight line relationship with their specific gravities.

Mr. Anderson meets this issue squarely by basing his formula on the efficiency of separation according to specific gravity only. It is true that machines that effect primarily a specific gravity separation are in no way directly influenced by the chemical analysis of the materials being handled and it can be argued that it is illogical to measure their efficiency by any criterion other than that of separation according to specific gravity.

For the moment, let us accept this basis and analyze the application of the efficiency conception and formula proposed by Mr. Anderson under conditions which he has not considered but that will readily be found. Let us take the separation shown in his table III as an example. He terms this an example of a high order of quantitative efficiency. We will show the weight distribution of the various gravity fractions as between clean coal and refuse in table XV.

Only in the gravity range of 1.50 to 1.70 has material been misplaced and, on the assumption that the cleaned coal and refuse are equal in weight, Anderson shows a quantitative efficiency of 99 pct. The gravity of separation is 1.60 and one third of the 1.50 to 1.60 sp gr fraction is misplaced in the refuse while one third of the 1.60 to 1.70 sp gr fraction is misplaced in the cleaned coal.

Let us now see what effect the gravity distribution of the total products has on the quantitative efficiency calculated by his formula. Still using the data in table



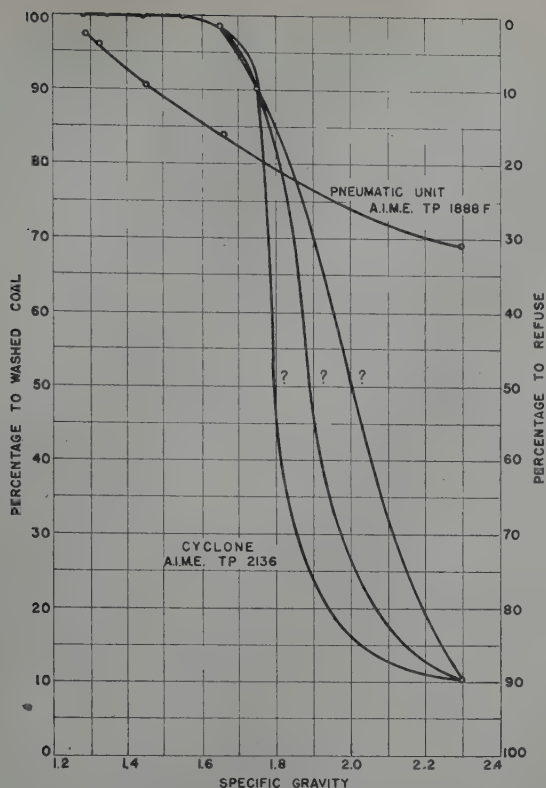


Fig. 4—Distribution curves for cyclone and pneumatic unit.

III and assuming that the clean coal and refuse are equal in weight, we need change only the total values of the 1.50 to 1.70 sp gr fractions to obtain a quite different quantitative efficiency. We need only assume that these values have doubled and immediately the amount of misplaced material is doubled and quantitative efficiency is 98 pct instead of 99 pct. The gravity of separation is still 1.60. We are still assuming that the cleaning unit is misplacing one third of the 1.50 to 1.60 sp gr material into the refuse and one third of the 1.60 to 1.70 sp gr into the cleaned coal. This is a reasonable assumption and experience shows that it is closer to the truth than the assumption that these gravity fractions would be separated in widely different proportions when their quantity is doubled.

In like manner it is readily apparent that should the total products contain one half the amount of 1.50 to 1.70 sp gr material given in table III, the quantitative efficiency would become 99.5 pct.

Such a situation immediately raises the question whether a quantitative efficiency formula that is so significantly affected by such moderate changes in the amount of 1.50 to 1.70 sp gr material is really valid and useful. That these changes are moderate is attested by the fact that they represent only a 4 : 1 change in material plus or minus 0.1 sp gr while the range of such values for all the coals of this country is perhaps 10:1. Taking only one seam of bituminous coal as an example—the Pittsburgh seam—the change from handloading to mechanical loading has often increased to 1.50 to 1.70 sp gr material from 2 pct to 6 or 8 pct.

One of the purposes of an efficiency formula is to establish figures on different coal-cleaning machines which will reveal their relative effectiveness or efficiencies in cleaning coal. As long as a formula is used which gives figures that are seriously influenced by the specific gravity distribution of the material being cleaned, no such comparisons are valid unless the materials being cleaned by two different machines are identical in their specific gravity distribution about the separating gravity.

It would appear to me that the "error area" method or some modification of it offers the most useful tool for determining the efficiency of coal-cleaning equipment which utilize primarily differences in specific gravity. As the literature shows, it is a conventional method of plotting the percent weight distribution of the various gravity fractions such as those given in the last two columns of table A.

My experience leads me to believe that a given unit of coal-cleaning equipment when fed with coals of differing specific gravity distribution will more nearly duplicate the weight distribution of identical gravity fractions than it will duplicate any other measure of separation which I know. Further, there is evidence to indicate that for a given coal-cleaning unit, the pattern of distribution of specific gravity fractions at given intervals of specific gravity each side of a separating gravity will be fairly accurately duplicated if the separating gravity is shifted by adjustments either upward or downward over the median range of gravities.

It is to be hoped that other discussers or later papers will contribute data on the "error area" method which will illuminate the points raised above. It is certain that, if these points can be established, very useful tools will be made available to study and compare the performance of coal-cleaning equipment and to more accurately predict the results that can be attained by coal-cleaning equipment when treating various coals and meeting a variety of conditions.

**H. F. Yancey and M. R. Geer**—Mr. Anderson's paper will be received with interest by all coal-preparation engineers, for the subject of washery efficiency is of unquestioned importance to everyone concerned with the cleaning of coal. Moreover, the author's contention that the basic terms "recovery," "yield," and "efficiency" should be used discerningly will meet with general approval. Even the formal discussion of AIME papers has sometimes shown the confusion that results from improper use of these terms. "Recovery" and "yield" have come to be used interchangeably to indicate the amount of cleaned product obtained in a washing operation, expressed as a percentage of the raw coal or feed. The term "efficiency," however, has no such universally accepted meaning because of the variety of efficiency formulas that have been advanced through the years.

Among the most widely used of such formulas, both here and abroad, is the one advanced by Fraser and Yancey of the Bureau of Mines in 1922, which is the ratio, expressed in percent, of the yield of washed coal actually obtained to the yield of coal of the same ash content that was available in the feed to the washing unit, the latter being determined by float-and-sink. Since this formula has been so widely used, and since the author questions its applicability, an appraisal of its usefulness in comparison with that of the substitute offered by Mr. Anderson seems desirable.

The concept of "misplaced material" is far from new, having been employed by the Bureau of Mines and by numerous European investigators for well over a decade. Those who have used it have considered it a

Table XV. Distribution of Material from Anderson's Table III

Cleaned Coal—50 pct of Combined Products Refuse —50 pct of Combined Products				
Combined Products	Specific Gravity		Pct Wt of Fraction	
	Sink	Float	Cleaned Coal	Refuse
45.0		1.30	100.0	0.0
2.0	1.30	1.40	100.0	0.0
1.5	1.40	1.50	100.0	0.0
1.5	1.50	1.60	66.7	33.3
1.5	1.60	1.70	33.3	66.7
1.5	1.70	1.80	0.0	100.0
2.0	1.80	2.00	0.0	100.0
45.0	2.00		0.0	100.0



useful criterion of washery performance but have not regarded it as an efficiency value, probably because it does not conform to an engineer's idea of efficiency as being the ratio of output to input. In other words, subtracting from 100 the percentage of raw coal that was misplaced gives a complement that might logically be termed the percentage of "properly placed" material, but although this figure indicates deviation from perfect operation, it does not constitute efficiency in the ordinary engineering sense. However, this objection to Mr. Anderson's proposed efficiency value is merely one of terminology and hence is less important than certain specific criticisms.

The principal disadvantage of the proposed formula is that it requires the use of a "distribution" or "error" curve to determine the specific gravity of separation to which the quantities of misplaced material are referred. Having pioneered in this country the use of distribution curves as a means of evaluating washery performance, we know from experience that often the number of specific gravities used in examining washery products is too few to fix the position of the curve accurately. The distribution curves for operation of a cyclone heavy-media unit shown in fig. 4 illustrate that even when as many as six densities are used in testing the washed coal and refuse, wide latitude may be possible in drawing the curve. Sometimes the data do not fix within  $\pm 0.1$  the specific gravity of separation, and variances of this magnitude can change the apparent amount of misplaced material by a ratio as high as 5 : 1. In fact, if Mr. Anderson customarily tests samples of adequate weight on all of the seven specific gravities from 1.30 to 2.00 indicated in his paper he is to be commended, for few engineers feel that they can afford the cost of so much laboratory work. Hence, one of the greatest objections to the proposed formula is the inordinate cost of obtaining the data necessary to insure accurate results.

A second limitation of the proposed formula is that it simply cannot be applied to the data for some less-efficient separations, notably pneumatic units, for which the distribution curve does not cross the 50-pct ordinate even at the highest specific gravity used in testing. This condition is illustrated by the distribution curve for a pneumatic unit shown in fig. 4. With such data the specific gravity of separation, and hence the amount of misplaced material, cannot be determined.

Examples of these limitations might well have appeared in the author's paper had he used the data from actual washery operation rather than purely hypothetical figures illustrating the operation of washing units to produce washed coal of the same quality as the feed, or refuse of better quality than the washed coal. Similarly, examples assuming washed-coal yields of 99 pct or consideration of feeds containing 76 pct of material between 1.50 and 1.60 sp gr do not lend themselves readily to showing the true applicability of the formula.

The several objections to the Fraser and Yancey efficiency formula mentioned by the author, upon examination, are found to be without foundation. The first objection—that the formula gives efficiency values of over 100 pct because of the liberation of coal through degradation in the washing process—concerns a circumstance that occurs only infrequently and can be eliminated by the simple expedient of determining the amount of coal available for recovery from that present in the washery products rather than from the feed—the same procedure employed by the author.

The second objection—that the Fraser and Yancey formula is not applicable to three-product separations—is equally groundless. The yield of middling actually obtained can be related to the amount of material of that ash content present in the feed in just the same way that the efficiency of the washed-coal recovery is evaluated. The procedure is exactly the same as that employed in using the yield-ash curve to estimate the yield and quality of products obtainable from a coal by a three-product separation.

To summarize, the proportion of the feed to a washing unit that is misplaced in the wrong product is a useful criterion of washery performance, as demonstrated by its continuing use over the years. Whether this percentage should be used, or its complement employed as suggested by the author, would appear to be a matter of personal choice—one figure is as revealing as the other. Certainly, however, the author has not demonstrated that the percentage of properly placed material should be adopted as an efficiency value to replace the well-known Fraser and Yancey formula, particularly when his proposal requires a much more costly laboratory procedure to give accurate results. Both criteria are useful, and they should be considered as complementary rather than competitive.

**M. G. Driessen**—Whereas the efficiency of electrical or thermal engines can be clearly defined, it seems that some discussion always arises as soon as the efficiency of coal preparation equipment is described.

We would be very happy indeed if an unbiased efficiency number could be attached to each of the many coal-cleaning apparatus, but unfortunately no method acceptable to all concerned has been proposed as yet.

Mr. Anderson's paper helps us insofar as the error, or distribution curve (see fig. 1, p. 258), is mentioned which at present is the best way of describing the behavior of a wash box or coal cleaner. However the picture would only be complete if error curves would be available not only for the total feed, but also for the different size fractions. A bundle of error curves for several size fractions would give an adequate picture of the performance of a wash box.

As far as efficiency is concerned, there is a parallel between the efficiency of a thermal engine, for instance a steam turbine and a coal washer.

In operating a steam turbine it is impossible to convert the total available heat into energy. The efficiency ratio is defined by the energy actually obtained and the part of the available heat, which theoretically can be converted into energy.

In a similar way the coal operator would be interested to know which part of the saleable coal, present in the feed to the washery, would be recovered. Thus the efficiency of a cleaning apparatus should be defined as follows:

$$\text{Efficiency} = \frac{\text{Saleable coal recovered}}{\text{Saleable coal in feed}} \quad \text{or} \quad \frac{\text{Actual yield}}{\text{Theoretical yield at same ash content}}$$

This formula, at present generally accepted in Europe, had been proposed in this country by H. F. Yancey and T. Fraser as early as 1929. (Bureau of Mines *Bull.* 300). It does not seem that there is any reason to depart from this formulation of efficiency. However the formula has the following inconveniences:

The efficiency is not a constant for a certain kind of wash box, but depends on:

1. The specific gravity of separation. Expressed in other words, the efficiency depends on the definition of saleable coal or on the allowable ash content of the clean coal. It is a well-known fact, that in times of coal scarcity, almost any coal quality can be sold, and the wash box efficiency may go up to 100. Therefore a curve should be prepared for each wash box, showing the efficiency in function of the specific gravity of separation, or in function of the ash content. If the other variables (2) and (3) are kept constant, this will be the only curve necessary to judge the wash box performance.

2. The specific gravity consist and the size consist of the coal. This difficulty can be overcome by the definition of a *standardized specific gravity consist*. Whereas in a small country one standardized specific gravity consist would cover practically all the available coal seams, it might be advisable for this country



to agree on different standardized coals for different districts. Also the size consist should be standardized.

3. The efficiency depends on the capacity of the wash box. One might agree to state the efficiency at the rated capacity or else have efficiency curves for full load, overload and half load of the machine.

### Conclusion

1. The best way to describe the performance of a coal-cleaning apparatus is a bundle of error curves for each individual size fraction.

2. The efficiency in which the coal operator is interested is the

actual yield of the wash box.

theoretical yield at the same ash content

3. These efficiencies should be plotted for a *standardized product* in function of the specific gravity of separation or in function of the desired ash content of the clean coal.

Note: By plotting the error curves on probability paper and by adjusting the abscissae with the help of logarithms, a straight line can be obtained which can be characterized by one number only. The number is called "l'ecart probable" (probable deviation) by the French and indicates the deviation from the specific gravity of separation for the ordinates at 25 or 75 pct.

**W. W. Anderson** (author's reply)—The comments of Messrs. Driessen, Yancey, Geer, and Griffen were greatly appreciated by the author for two principal reasons:

1. It is well known that these men have had a wide experience and have given considerable thought over a long period of time to the subject of coal cleaning equipment performance.

2. Their various discussions served to emphasize points of agreement and disagreement in regard to measures of performance.

The published literature of the AIME demonstrates how provocative is the subject of the measures which define the performance of coal-cleaning equipment; yet the fact that there is some agreement among coal preparation engineers offers hope for eventual clarification of the language and mathematics required to describe performance accurately.

As mentioned in the author's paper, numerous investigators, starting with David Hancock in 1912, have contributed fundamental ideas which have gradually expanded the basic understanding of the separations effected by coal-cleaning equipment. However, progress toward a complete understanding has been exceedingly slow, and 38 years after David Hancock's proposal of the Hancock chart there is still disagreement regarding certain vital points. Many coal preparation engineers at our coal mines today still have only the vaguest notions regarding the true performance of the equipment for which they are responsible.

Much of the recent literature in which performance values are reported deals with new types of cleaning equipment not to be found in most coal preparation plants. Moreover, the articles have been written primarily from the viewpoint of applicability, rather than from the standpoint of precise definitions of the terms used to describe performance. As a consequence, except for a few isolated instances, much of the true significance of the curves of distribution and the discussions of improperly placed material have failed to be transmitted from the academic level to the practical level of the coal-cleaning plants at the mines.

It was precisely this void which prompted the author to submit a paper dealing principally with the methods used for expressing performance. In large measure, the paper was a review of the relationship between distribution curves, gravities of separation, and the amount of improperly distributed material; and because an attempt was made to confine the discussion in a thorough manner to measures of quantitative efficiency, parts of the paper perhaps were overly simplified to the exclusion of a more comprehensive subject matter. Certainly there is much more to be said on the

subject of coal-cleaning performance than is contained in the paper on quantitative efficiency; but there must be a widespread understanding of a few basic terms, such as "recovery," "yield," "efficiency," "gravities of separation," and "improper distribution," prior to the general acceptance of a more complicated subject matter dealing with bundles of error curves, error areas, and qualitative efficiencies.

Incidentally, none of the discussers mentioned qualitative efficiencies directly; but it can be inferred from some of the discussion that an acceptable method for determining qualitative efficiency, which would measure performance in terms of chemical characteristics such as ash content, would not only be a welcome additional measure of performance, but would also help to clarify some of the present confusion in respect to quantitative efficiency values.

It was particularly gratifying to learn of the unanimous approval of the discussers in regard to the difference between the basic terms "recovery" and "efficiency." These terms have been misunderstood by many coal preparation engineers, and the author attempted to demonstrate, by numerical example, the fundamental difference. Furthermore, even in recent literature there has been a misconception of the terms "efficiency" and "point of separation." Therefore, it was the hope of the author that the very simple explanations of some of these basic terms, as given in his paper, would serve to exemplify the differences in meaning. After all, if authors at the academic level are confused, it is little wonder that preparation engineers at the mines cannot discern the true significance of these terms.

There is also unanimous agreement by the discussers that curves of distribution and determination of the amount of improperly distributed material are worthwhile to the evaluation of performance. It is at this point, however, that concurrence of thought ceases.

M. G. Driessen mentions that "a bundle of error curves for several size fractions would give an adequate picture of the performance of a wash-box," and he discusses the fact that operating conditions influence the curves of error and the efficiencies of separation. John Griffen gives a numerical example to demonstrate the same point, and questions the value of an efficiency measure which is influenced by the operating conditions, specifically, in his example, the quantity of near-gravity material.

As stated in his paper, it is the opinion of the author that "the operating conditions may be the cause for the result, but they are not the result, and should not be confused therewith." The fact that efficiency varies with the operating conditions is accepted by engineers in almost all other fields of activity. Why it should be so difficult for coal-preparation engineers to accept the same basic fact with regard to coal-cleaning equipment is difficult to understand. It is not expected that a steam generating unit will have the same efficiency at half load that it does at full load; nor is it expected that an automobile engine will operate with the same efficiency at a speed of 90 mph that it will attain at 45 mph. Then why should a piece of coal-cleaning equipment be expected to have the same efficiency under one set of operating conditions that it may have under another? It simply does not make sense to anticipate any such result.

Messrs. Yancey and Geer discuss at length the value of the amount of misplaced material and admit that the efficiency determined in terms of the washery products rather than the feed is proper procedure; but they deny that the improperly distributed material can measure the efficiency of separation and doubt that the liberation of coal through degradation in the washing process concerns a circumstance that frequently occurs. An instance is cited wherein a curve of distribution on a pneumatic unit showed no point of separation, and the statement is made that the efficiency determined in terms of misplaced material "cannot be applied to the data for some less efficient separations, notably pneumatic units."



The author is in accord with Messrs. Yancey and Geer that efficiency in terms of misplaced material cannot be determined for this particular curve because there is no point of separation in accordance with specific gravity. Hence, there can be no efficiency in terms of specific gravity.

However, this does not mean that all pneumatic operations will give a curve of this type; nor should it be inferred from the discussion of Yancey and Geer that a distribution curve such as this is peculiar to pneumatic machines alone. Many curves for pneumatic machines have crossing points at definite gravities of separation; and, hence, they can be judged for efficiency in terms of misplaced material. Moreover, curves of distribution for other types of separating equipment sometimes have no crossing points for specific sizes, indicating that no true separations in accordance with specific gravity were accomplished in these sizes.

Since the author stated in his paper that if there was no point of separation in terms of specific gravity, there could be no quantitative efficiency in accordance with his proposed formula, the example cited by Messrs. Yancey and Geer seems to be neither valid nor applicable to the argument.

Furthermore, the author also cannot concur with Yancey and Geer in their statement that "the liberation of coal through degradation in the washing process concerns a circumstance that occurs only infrequently." Rather, it is the belief of the author that this occurrence is more frequent than infrequent. The soft, low volatile coals of central Pennsylvania, southern West Virginia, and Alabama all break down in the washing process. In addition, jig plants are one of the most numerous types of cleaning devices used, and a large proportion of these jigs operate with material from secondary elevators being recirculated through a crusher to the head of the jig. In such case degradation of a portion of the feed is a premeditated, deliberate action of the operator.

In regard to the statement that the author's method requires much more costly laboratory procedure to give accurate results, this statement is definitely misleading. In the first place, the use of the Fraser and Yancey formula requires fractionation of the raw feed; whereas the author's proposed method does not. In the second place, if the test engineers have any idea at all of the approximate gravity of separation, it is not necessary to develop the complete curve of separation, but only that portion at or near the 50 pct abscissa. In the third place, the author's proposed method requires no laboratory analyses, other than float-and-sink separations; ash contents are not needed, as for the Fraser and Yancey formula.

The fact that the author's theoretical data showed test gravities from 1.30 to 2.00 should not be reason to befuddle the issue because the author purposely tried to give a complete explanation of the proposed method for the benefit of those who are not familiar with distribution curves. The proper number and choice of gravities is dependent on the judgment of the test engineers. In many cases, the gravities which determine the top and bottom ends of the curves of distribution are unnecessary; hence, they can be omitted from the test procedure. It seems obvious, however, that a theoretical discussion would not be complete without showing a full range of gravity consist.

Summarizing the discussion, there are points of agreement in regard to the basic definitions of the terms "recovery," "yield," and "misplaced material," and the proper method for determining the point of separation. However, there is still no accord in regard to an acceptable method for learning the value of the quantitative efficiency of separation.

It is strongly suspected by the author that the Fraser and Yancey formula applied to washery products rather than to the washery feed may give virtually the same answer as the author's proposed method for securing quantitative efficiency from the amount of

misplaced material. However, as pointed out, the Fraser and Yancey formula requires at least the ash contents on the float-and-sink fractions, whereas efficiency determined only on weights of float-and-sink products is a quicker and cheaper method for obtaining the result.

In addition, it must be remembered that the area under the distribution curve is only another way of expressing the percent of misplaced material. Hence, it is difficult for the author to understand the need for a complete distribution curve, especially since the heavy gravity end of the curve is very expensive to attain. As most investigators know, the high gravity end of the curve is seldom a true plot of experimental data. Reference to fig. 4 in the discussion by Messrs. Yancey and Geer lends support to this statement, as will reference to published literature.

In conclusion, the author again wishes to thank Messrs. Driessen, Yancey, Geer, and Griffen for their discussions. The author also wishes to repeat that he had no intention of claiming credit for the concept of improperly distributed material, and so stated in his paper. The author does believe, however, that the worth of this concept has been underestimated. Otherwise, the proposal that it be used as a basis for defining quantitative efficiency of separation would not have been made.

---

J. GRIFFEN, McNally Pittsburg Mfg. Corp., Pittsburgh, Pa.; H. F. YANCEY and M. R. GEER, Bureau of Mines, Seattle, Wash.; M. G. DRIESSEN, deceased.

---

## Operating Data for a Bird Centrifuge

by A. C. Richardson and Orville R. Lyons

### DISCUSSION

**F. X. Ferney**—We are pleased that this paper was presented at this meeting and thank Mr. Richardson and Mr. Lyons for their effort and work in preparing it. We agree with the authors that it was unfortunate that more comprehensive data were not available when the paper was prepared, this being particularly true since we had no knowledge that the paper was being written and could not make available data which were in our process files. Further, it is our opinion that because the data are not sufficiently comprehensive the results which are presented are erratic and will lead to false conclusions. As we see it, there are three specific reasons why the data which are presented cannot be considered accurate.

1. The preparation plant was not at equilibrium while the samples were being taken. This is indicated by the fact that the percent of solids in the circulating water varied from 3.9 pct to 10.7 pct during the period when the plant was sampled.

2. We do not believe that the samples of Bird feed which were taken were representative and could indicate the true feed to the filter. Evidence of this is shown in the data where the dewatered product leaving the centrifuge contains more coarse material than the feed to it, that is, material which is +6-mesh in size.

3. We question if the data concerning screen analysis in the fine sizes is accurate and would conclude (from the data) that these tests were made by a dry-screen method. We have found that wet screening is always necessary to obtain a true picture of the quantity of —200-mesh fines which is present.

The test work described in the Lyons-Richardson paper was conducted some time ago and the tests were carried on during a period when the preparation plant had not ironed out all of their start-up problems. Since that time most of these problems have been corrected and now this is a smooth running, well-operated prep-



aration plant. Recently engineers of the Bird Machine Co. visited this plant and carried out extensive tests over a two-week period. During these tests, every effort was made to operate the plant with a closed-water system. However, it was not possible to keep the water system 100 pct closed for the quantity of fresh water entering the plant (most of this was gland water on the pumps) exceeded the quantity of water which was leaving with the coal and refuse and, therefore, a small bleed-off was necessary. This amounted to about 25 to 30 gpm. However, the bleed-off was not sufficient to prevent excessive build-up of solids in the circulating water and at the end of the test period, the solids content in the circulating water was about 25 pct. It is appreciated that this is a higher solids content than is desired for the ideal jig operation, but one of the purposes of the tests which we carried out was to determine the rate of build-up so that proper recommendations could be made to close the water system and prevent excessive solids in the circulating water. Since this preparation plant has been operating, the operators have learned that the quantity of  $\frac{1}{4}$  in. x 0 coal which is entering the preparation plant is considerably greater than was first anticipated. Originally it was expected that the quantity would be between 40 and 50 tph. Actually it averages between 80 and 90 tph.

When the plant was built, just one Bird filter was installed. At times this filter handles in excess of 60 tons of coal per hour but it cannot handle all of the fines being brought into the plant. To permit acceptable plant operation, screen changes have been made, the purpose of the change being to prevent the total quantity of  $\frac{1}{4}$  in. x 0 coal from passing to the sludge tank and becoming feed for the filter. The screen change which was made to replace the  $\frac{1}{4}$ -in. screens with 2-mm wedge wire in two thirds of the screening area. This caused a substantial quantity of the  $\frac{1}{4}$ -in. +10-mesh coal to remain with the coarser sizes.

The present feed to the Bird filter amounts to approximately 50 to 55 tons of coal per hour and the coal product leaving the filter carries about 14 to 15 pct moisture. The feed to the filter contains a considerable quantity of fines, averaging between 20 and 25 pct —200-mesh. The dewatered product from the centrifuge contains about 15 pct —200-mesh fines, the filtrate 7 to 8 pct solids. The solids in the filtrate are extremely fine, average particle size being about 10 to 12 microns. The operators appreciate that their overall moisture is higher than it should be and they are planning to install a second Bird filter in the near future which will allow them to handle all of their  $\frac{1}{4}$  in. x 0 coal in Birds. The use of this second filter will also allow them to operate with a closed-water system and the build-up of solids in the circulating water will not exceed 15 pct. Similar results are being obtained in other plants. Of course, if it continues to be necessary to introduce more water into the plant than is removed with the product and refuse, then a small bleed-off will be required.

There are presently about 85 Bird filters operating in the coal industry. Collectively they have a capacity in excess of 50,000 tons of coal per day. When desired, it has been found possible to completely close the water circuit without excessive build-up of fines. In plants operating with a closed-water system, Bird filters are being used in two ways. If complete recovery of feed solids is desired, all of the fines can be retained in the solids product discharged from the Bird filter. Usually this will mean a  $\frac{1}{4}$  in. x 0 product containing about 12 to 14 pct moisture. If a lower moisture is required, a bleed-off can be employed which can be diverted to a polishing type Bird filter. The dewatered product from the "polisher," because of its high-ash content, is usually sent to refuse and the clarified water returned to the circuit. The solids in the dewatered product from the Bird polisher usually contain from 25 to 30 pct ash.

One plant employing a system of this type with a polisher handling approximately 200 gpm has reduced the moisture in the product from the "Bird dryer" from 14 pct to 7.5 to 8 pct.

Earlier we have said that the data in the Richardson-Lyons paper indicated that accurate feed samples had not been obtained. We should point out that we have had similar experience when attempting to sample Bird feed. This is particularly true where jigs or launders are used to clean the fine coal. In plants where tables are employed, we find that the Diester distributor or similar device that splits the main stream into a number of smaller streams is an aid to sampling. Where we have been able to obtain accurate samples, we have observed that some degradation occurs in the Bird filter. The  $\frac{1}{4}$ -in. +6-mesh material will be reduced by approximately one third. That is, if there is 30 pct of this size material present in the feed, we will have about 20 pct retained in the cake. The —6-mesh +10-mesh will also be degraded on the order of about 10 to 20 pct. The vast majority of the coal which is broken down will report in the sizes from 14-mesh to 100-mesh. The quantity of —200-mesh is increased slightly, approximately 2 pct. That is, if there is 20 pct —200-mesh present in the feed, we find a total of about 22 pct by combining the dewatered product and the solids present in the filtrate. Most of these extreme fines, which are formed in the Bird filter, are retained in the cake and they do not have any serious effect on the circulating water.

We are hopeful that these comments together with the information obtained by Lyons and Richardson will indicate the results which are being obtained with Bird filters. It is our practice to conduct extensive tests similar to those carried out by Lyons and Richardson wherever possible.

**Orville R. Lyons** (authors' reply)—Relative to Mr. Ferney's objection to presentation of data obtained at a preparation plant not operating under equilibrium conditions, the authors believe that data for a plant where difficulties are being encountered or under fluctuating load conditions are of much more value to a preparation plant operator or a prospective operator than data obtained at a smoothly operating plant. The data presented show the interrelationships of some of the factors affecting the performance of one particular Bird centrifuge when the machine was being operated under a diversity of conditions.

The authors must take issue with Mr. Ferney when he questions the reliability of the samples. Every effort was made to obtain representative samples and the degree of correlation shown by the various graphs clearly proves that the samples were representative. It is true that there are some incongruities in the screen analyses presented but these screen analyses represent only a few of the samples taken and we believe that any one who has ever sampled a preparation plant will admit that it is virtually impossible to prevent the occurrence of some queer results. The authors feel that their original explanation of these discrepancies is still valid.

The authors must also disagree with Mr. Ferney's conclusion that the screen analyses were made by dry screening methods. The screen analyses were actually obtained by a combination wet and dry screening method in which the finer sizes were wet screened and only the coarsest sizes dry screened.

The authors wish to point out that they were not attempting to discredit the Bird centrifuge as a dewatering device. Their intent was and is to make available to the general public operating data concerning the Bird centrifuge. The authors appreciate Mr. Ferney's desire to point out the results to be expected when operating a Bird centrifuge under equilibrium conditions and wish that he would present a paper containing the results of the tests that the Bird Machinery Co. has conducted on its own equipment under operating conditions.

*F. X. FERNEY, Bird Machine Co., South Walpole, Mass.*



## Some Factors in Selection and Testing of Concrete Aggregates for Large Structures

by Elliot T. Rexford

### DISCUSSION

**G. B. Walker**—In the paper reference is made to two methods of preventing reactivity between cement and aggregate in concrete structures. A third method is suggested: removing the reactive components from the aggregate before mixing. This may be done by means of a difference in the specific gravity between the desirable and the deleterious components, if a difference exists.<sup>16</sup> There are many commercial plants in operation making a separation between the wanted and unwanted components of mineral ores and coal, based on specific gravity difference. These plants use heavy-media separation processes in which the separation is made in a suspension of a magnetic material such as magnetite or ferrosilicon in water. The medium is held at a specific gravity between that of the desirable and undesirable components. The desirable, being heavier, will sink and the deleterious, being lighter, will float.

During 1948 and 1949 about 200,000 yd of gravel were beneficiated by heavy-media separation at the airfield of the R.C.A.F.<sup>17</sup> for use as concrete aggregate. At the Rivers plant, the main objective was to separate the soft shale from harder components of a local gravel deposit. This has been accomplished for two operating seasons and is a commercial and technical success. The concrete made with gravel cleaned by heavy-media separation tested slightly higher in compressive strength than the gravel from the nearest commercial source. Two seasons of operation and 200,000 yd of cleaned gravel produced have shown that, cost-wise, heavy-media separation is a process of sufficiently low cost to process economically a low-value material such as gravel.

The Rivers plant is particularly attractive from an economic standpoint because the freight rate on suitable gravel was \$2.51 per yd from the nearest source.

The Rivers gravel beneficiation plant is a relatively small, prefabricated unit having a feed rate of about 80 tph. However, heavy-media separation plants having feed rates of up to 2000 tph are in operation and larger plants could be built.

The successful removal of reactive components from gravel by heavy-media separation would depend on the difference in specific gravity between the desirable and undesirable components. The specific gravity of the media used in these processes can be and is, in commercial practice, held to within  $\pm 0.01$  of the desired specific gravity so that the specific gravity difference does not need to be large. In one commercial plant beneficiating magnesite, successful operation is obtained in separating materials, some of which have a specific gravity difference of as little as 0.02.<sup>18</sup>

Another prerequisite to successful separation by heavy-media separation is that the good and the troublesome components must be substantially free of each other at a size of about +10-mesh since the most effective size range is from 10-mesh up to 8 in., according to present commercial practice.

In certain types of concrete structures, color is of some importance. The staining and spotting of concrete by certain components in the aggregate after aging and weathering is a problem. A separation based on specific gravity difference might also prove useful in eliminating color-producing or stain-producing particles from aggregates.

**Carlton Leith** (in place of author)—Mr. Walker has presented an interesting commentary on the application of heavy-media separation for removal of poten-

tially reactive constituents from concrete aggregate. Although the technique is neither new nor unique, it has not been used extensively in the processing of aggregate, partly because most operators do not wish to increase their production costs if their product is usable without special processing. In the selection of materials for large structures it is the cost per yard of the concrete in place, including all of the factors involved in placing the concrete, which will determine whether any portion of this total cost is excessive. On this basis, perhaps the prevention of reaction between cement alkalis and coarse aggregate by the elimination of potentially reactive material from a conveniently located gravel deposit is more practical than might be expected.

A more important consideration is that much of the reaction which causes deterioration of concrete results from the attack by the alkalis of the cement on constituents of the *fine* aggregate. The practical limiting grain size for effective heavy-media separation is about 10-mesh, although in cases of extreme gravity differences grains as fine as 48-mesh can be treated if dual magnetic cleaning circuits are included in the flow scheme. Thus, even if the difference in specific gravity between the potentially reactive material and the innocuous constituents of natural sand were great, which would be a very unusual condition, the lower limit of grain size for effective heavy-media separation would preclude its use for beneficiation of fine aggregate, as the average concrete mix calls for approximately 20 pct of the —No. 4 material to be finer than 48-mesh.

<sup>16</sup> G. B. Walker and C. F. Allen: Beneficiation of Industrial Minerals by Heavy-Media Separation. *Transactions AIME*, 184, 17: *Mining Engineering*, January 1949, TP 2503H.

<sup>17</sup> C. V. Trites and J. D. Shannon: Mining Process Applied to Runway Construction. *Journal Engineering Institute of Canada* (April 1949).

<sup>18</sup> Northwest's Magnesite's Heavy-Media Separation Plant. *Mining World* (December 1947).

G. B. WALKER, American Cyanamid Co., Stamford, Conn.

## Ground Water in California

by J. F. Poland

### DISCUSSION

**B. C. Burgess**—Prior to hearing this paper presented at the San Francisco meeting, I travelled by car from Yuma, Ariz., across south-central California and up through the San Joaquin Valley. After hearing the paper I returned by the coast route to San Diego then along the Mexican border to Yuma. The striking feature of these trips, related to this paper, was the dry river and creek beds at a time of year not the dry season. Contrasted with this was the wasteful use of water in irrigation canals and laterals.

Poland says "at least 90 pct of the ground water pumped is used for irrigation." That being the case, it would appear that first consideration should be given to conservation of this use. Probably a small part of this is actually reaching the plants. Some measure of this loss between pump and crops would be revealing.

Methods that might be employed to reduce this loss are pipe lines and spraying of some crops such as the banana planters are using in Central America. Another idea which might be investigated is to line canals and ditches with some water-repelling mineral like pyrophyllite. Perhaps a comparatively thin sprayed-on



lining of such mineral would suffice. A weed-killer might be incorporated in it.

Conservation and better utilization of the ground water supply would appear to offer a better solution of the problem than any efforts that can probably be made toward increasing recharge of the aquifers or legally restricting pumping.

**J. F. Poland** (author's reply)—Mr. Burgess suggests that conservation and better utilization of the ground-water supply offer a better solution than any efforts in increasing recharge of the aquifers.

In California ground-water basins, the "irrigation efficiency" (the amount of water actually consumed by plants) ranges from about 40 to possibly 90 pct. In several large areas, the efficiency has been estimated at about 50 pct, although the average for the State may be nearer 60 pct. The greater part of the water not consumed by the plants percolates to the water table and usually back into the ground-water supply, but some runs off as waste to nearby streams. Although it is inefficient to pump water and have it return to the ground-water supply, such water is not actually wasted. It is doubted that, at most, more than 10 to 20 pct of the ground water pumped for irrigation is actually wasted by running off to the sea, passing into an unusable shallow water body, or being lost through evapotranspiration by native vegetation. Thus, probably not more than one to two million acre-feet is wasted at most. Even if the greater part of this waste were eliminated by improving methods of irrigation, the total amount so conserved would be small compared to the foreseeable needs in increased recharge to ground-water basins. In the eventual full development of the San Joaquin Valley, for example, the amount of ground water required in dry periods may be several million acre-feet a year greater than present pumpage. This increased draft can be provided for only by increasing recharge and storage in wet periods.

Mr. Burgess has made a good point in stating that conservation and better utilization of the ground-water supply is needed. However, these measures will not be sufficient. The demand for ground water is increasing rapidly, and the proper development of the total water supply of the State calls for increased use of ground-water basins as storage reservoirs. Their combined capacity is far greater than that of all surface storage reservoirs that are now contemplated in California.

---

*B. C. BURGESS, Mining Consultant, Monticello, Ga.*

---

## Industrial Mineral Economics and the Raw Materials Survey

by R. B. Ladoo and C. A. Stokes

### DISCUSSION

**Bruce C. Netschert**—It is unfortunate that the authors of this paper consider it necessary to begin with an expression of concern over possible false interpretations of the word "economics." In their preoccupation with the definition of economics, they have adopted a definition of mineral economics which is, to say the least, unduly narrow.

Just as the broad field of economics is not confined to a study of the profitability of business concerns, but includes the problems of production, distribution, and consumption as they pertain to society as a whole; so an inclusive definition of mineral economics should not be confined to the determination of the profitability of mineral-producing and processing enterprises, but should include the significance of the unique characteristics of mineral resources as raw materials for the use of society. Such features as the exhaustibility and localized, haphazard occurrence of deposits, the existence of a secondary (scrap) supply, and the increasing cost of operation during the life of a mining enterprise

are obviously factors which concern those businesses which are producing mineral raw materials, since they partially determine the profitability of such enterprises. They are also of concern to society as a whole, however, as characteristics of one of the basic elements of the economic system. In the last analysis, the contribution which mineral economics can make as a means of determining and guiding social policy with respect to the production and utilization of mineral resources is perhaps more important than its use as a basis for determining the cost accounting procedure of individual firms.

The list of "economic factors peculiar to the industrial minerals" which the authors present is in reality an application of such a broad definition of mineral economics. An inconsistency appears, however, in the inclusion of items 8 and 9 in the list. As this writer sees it, the point in question is: What influences do the characteristics of industrial minerals have on the characteristics and operating procedures of industrial mineral enterprises which are not present in the metallic mineral field? In answering this question with items 8 and 9, Messrs. Ladoo and Stokes do not recognize that there are two distinct types of differences between the two fields of enterprise. There are, on the one hand, important basic economic distinctions due to inherent economic characteristics of industrial minerals which do not pertain to metallic minerals. On the other hand are those characteristics of the industrial mineral enterprises peculiar to them alone, but which are superficial and temporary, in that they may be changed or eliminated at the discretion of the managers of those enterprises.

The lack of adequate research and development in industrial mineral production, processing, and marketing (item 8) is not due to an inherent characteristic of industrial minerals. It is true that one may perhaps describe the field of industrial mineral enterprise in terms of such a deficiency, just as one could, until recently, point to a similar lack of research and development in the coal industry; but unless it can be shown that the deficiency has been wholly or partially due to the very nature of industrial minerals themselves it is not an "economic factor peculiar to the industrial minerals" but a temporary characteristic peculiar to the industry. In the writer's opinion, the authors have not demonstrated that the former relationship exists.

Similarly, item 9, the "influence of technologic developments," is also not inherently peculiar to industrial minerals. Nowhere in the discussion of this item do the authors mention anything that is not equally applicable to the field of metallic minerals. This is not meant to imply that the specific technologic developments which the authors list are of equal significance in both fields. It does mean that such a statement as, "technological advances together with new consuming areas to provide markets make deposits commercially valuable which once were of no interest" cannot be considered as an argument that technological developments have significance in the field of industrial minerals alone.

In considering the problem of stockpiling, the authors note that the stockpiling of nonstrategic materials might be desirable if future wartime needs could exceed domestic production capacity, but dismiss this as hardly adequate to justify such stockpiling. The problem, however, can be stated in broader economic terms, i.e., the real costs (as distinguished from the money costs) of prewar *versus* wartime production. In other words, it might well be that the labor and capital required to produce a given amount of a certain mineral raw material could be used more efficiently in another industry. In such a case it is obviously advantageous to stockpile the material in prewar times rather than forego the benefits of additional production in another line of endeavor during wartime under conditions which demand the optimum use of all resources, including manpower and capital. To the writer's knowl-

edge, this issue has been largely overlooked in discussions of stockpiling policy.

The Raw Materials Survey which the authors describe is a laudable example of thinking in terms of the broad definition of mineral economics set forth above. In avoiding the policy of fostering local mineral self-sufficiency in an attempt to utilize all local mineral deposits, whatever their economic worth, the Survey, along with the economic profession, sees the goal of economic activity as the satisfaction of human wants through the utilization of the scarce means of production with the least cost to society.

BRUCE C. NETSCHERT, *University of Minnesota, Duluth, Minn.*

## Recent Developments in the Manufacture of Lightweight Aggregates

by John E. Conley and John A. Ruppert

### DISCUSSION

**W. B. Mather**—The growing national importance of lightweight aggregates to the construction industry makes this paper a timely presentation of the most recent developments in the field. It is presented in a clear, concise manner readily understandable by those unfamiliar with the subject of lightweight aggregates.

The classification of lightweight aggregates into two groups based on their physical properties as suggested by the authors should be adopted by the industry. The value of this dissertation however, could be enhanced by reference to a paper by Kluge, Sparks and Tuma.<sup>25</sup> The latter tabulates the physical properties of lightweight aggregate as compared to sand and gravel concretes.

The whole problem of expanded clay and shale aggregates has been ably presented by W. G. Bauer in a series of five papers.<sup>26</sup> The addition of these references to the bibliography would be of value to those interested in the subject.

The authors' discussion of the production of bloated aggregates by sintering machines is of paramount im-

portance since many qualified individuals familiar with the industry agree that the high cost of rotary kiln operations is detrimental to the industry and that appreciable expansion of present markets can best be secured by the development of lower cost operating equipment based on sintering machines, stationary kilns, etc.

**John E. Conley** (authors' reply)—The authors appreciate greatly the kind remarks of Mr. Mather and concur completely with him that the subject of lightweight aggregates is timely and of great importance to the construction industry. His suggestion of bringing the literature references up-to-date is certainly an excellent idea.

### References

<sup>25</sup> R. W. Kluge, M. M. Sparks and E. C. Tuma: Some Properties of Lightweight Concrete Aggregates. *Journal Am. Concrete Institute* (May 1949) 625-642. Also, *Pit and Quarry*, 42, No. 3, Sept. 1949, pp. 175-177, 193.

<sup>26</sup> Wolf G. Bauer: Mechanics, Techniques and Economics of Expanded Clay-Shale Aggregate Production. *Pit and Quarry*, 41, I, July 1948, pp. 71-73; II, Sept. 1948, pp. 93-96; III, Dec. 1948, pp. 91-95; IV, June 1949, pp. 87-90, and V, Nov. 1949, pp. 119-121.

## Lightweight Aggregate Industry in Oregon

by N. S. Wagner and R. S. Mason

### DISCUSSION

**W. B. Mather**—A minor recommendation that may be offered to improve the paper is the inclusion of a map of Oregon showing the general location of the various deposits. This is of especial importance to persons like myself who are not familiar with the various areas and counties mentioned.

**N. S. Wagner and R. S. Mason** (authors' reply)—We agree that an index map of Oregon showing the location of the various nonmetallic deposits would add to the paper (fig. 1).

W. B. MATHER, *Southwest Research Institute, San Antonio, Texas.*

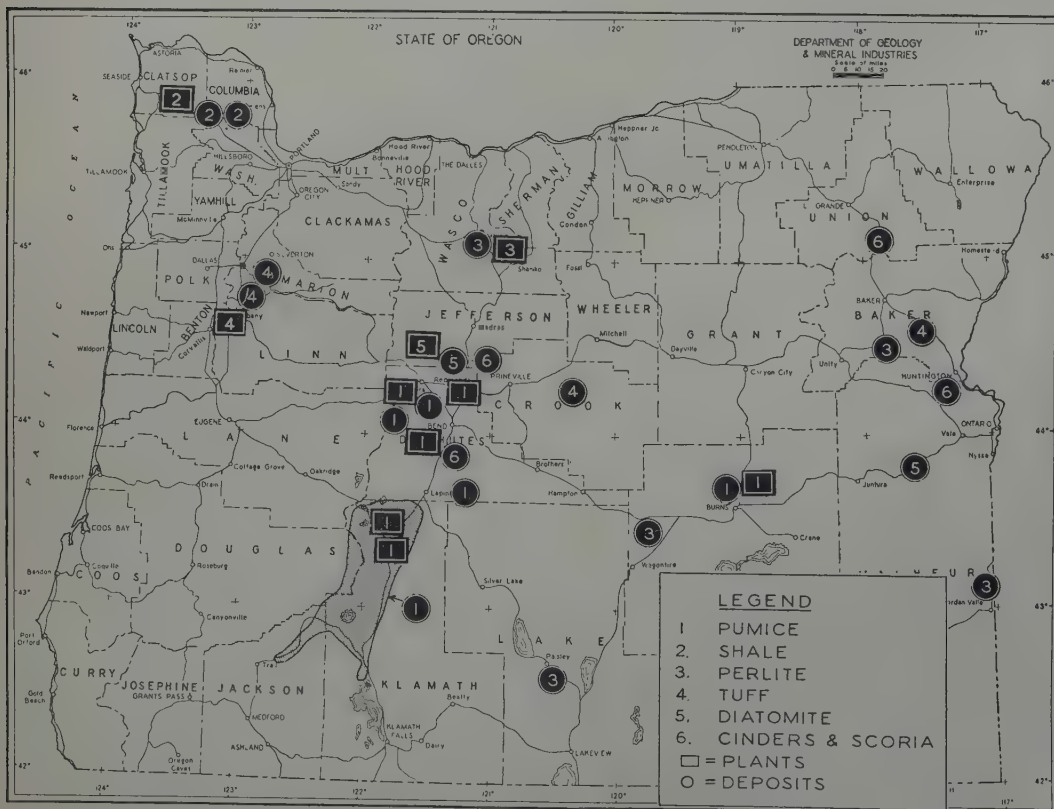


Fig. 1—Non-metallic deposits of Oregon.



# Aspects of Structure and Mineralization Used as Guides in the Development of the Picher Field

by Joseph P. Lyden

**In the Picher field, structure made openings for the circulation of the mineralizing solutions by flexing, shearing, and fracturing the sedimentary beds. This structure is used with the spatial and genetic relationship of the ore minerals, sphalerite and galena, and the gangue minerals, dolomite and jasperoid, in prospecting for ore bodies.**

**T**HE Picher Mining Field, fig. 1, which lies between Baxter Springs, Kansas, and Commerce, Okla., is the most intensely mineralized and the largest zinc-lead ore producing area in the Tri-State District of Missouri, Kansas, and Oklahoma. It has a vast lateral extent of underground workings that is unequaled for the observation of the geology of zinc-lead ore bodies in sedimentary horizons; and thousands of churn drill holes in the unmined areas supplement the underground observations with useful data for the interpretation of the geology.

This paper is confined to the ore deposits of the Picher mining field. The reader is referred to the bibliography for the general geology of the Tri-State district which has been ably covered by many geologists.

The writer was co-author of several publications on the Tri-State district.<sup>1-6</sup> This paper is necessarily a repetition of parts of those publications but contains a few changes and additions on the deposition of the minerals. The parts of this paper concerning mineralization maps and pipe slump-structures are not covered in the above publications.

## Geology

The geology of the Picher mining field is similar to that of the entire Tri-State district because the sedimentary horizons that are found here are found throughout the district, and the ore bodies in the respective horizons have similar characteristics. The beds that contain the ore bodies in the Picher field are illustrated and described in fig. 2.

The five unconformities found in the mines of the Picher field are indicated on the geologic section,

fig. 2. The unconformity shown at the base of Meramec (B bed) is relatively unimportant since B bed was mostly eroded from the area of the Picher field before the Chester beds were deposited. B bed has been observed in the cuttings from only a few churn drill holes.

The Pennsylvanian (Cherokee shale) formation is the top horizon found in the Picher mining field. It has been eroded from the southeastern part of the field, but because of the regional dip of the sedimentary beds to the west, it attains thicknesses of 100 ft to 200 ft along the west edge of the field. It is 200 ft to 300 ft thick in the Miami trough. It was deposited on the Boone limestone surface, which contained relatively few shallow erosion channels and very few depressions. Subsequent to the deposition of the shale, solution of limestone from underlying limestone horizons by circulating solutions caused the overlying beds, including the Cherokee shale, to slump into the larger openings formed by the solution and removal of the limestone.

## Chert

The origin of the chert in the Tri-State district has been a controversial subject among geologists. Tarr<sup>7</sup> advocated a syngenetic origin for the chert, postulating that it formed from colloidal silica deposited in the Mississippian seas. Leith, Dake and Leighton<sup>1</sup> argued in favor of silicification of erosion

JOSEPH P. LYDEN, Member AIME, is Geologist, Eagle-Picher Mining and Smelting Co., Cardin, Okla. AIME New York Meeting, February 1950.

TP 2979 AI. Discussion (2 copies) may be sent to Transactions AIME before Jan. 31, 1951. Manuscript received Aug. 7, 1950.



Fig. 1—Map showing underground workings in Picher Field.

surfaces, postulating the solution by meteoric waters of silica from weathered areas and redeposition in the limestone on and near the surface. Fowler, Agar, Gregory and the writer<sup>6</sup> advocated, and Giles<sup>8</sup> favored the theory that the chert is epigenetic, that it replaced the limestone, and is hydrothermal in origin.

Chert in varying amounts, fig. 2, replaced limestone, forming nodules in the more massive limestone beds, C, E, J, K, M, and R, replaced alternate limestone strata in the thin bedded limestone beds G-H, O, and Q, and more or less completely replaced limestone beds 15 ft to 35 ft thick, such as D, F, L, N, and P beds. In areas where deformation was intense, chertification was also intense and often all the beds were completely chertified in zones 100 ft to 150 ft thick. Chertification<sup>6</sup> took place during Mississippian time before the Chester limestone, which contains no chert, had been deposited.

Two types of chert occur, cotton rock chert and dense to glassy chert.

Cotton rock chert is a calcareous chert that permeated and partly replaced the limestone with cryptocrystalline silica. The ratio of silica to limestone varies considerably.

Tripoli, a light porous mass of silica that easily crushes to a very fine powder, is a residual product of cotton rock beds that we exposed to weathering. The limestone and silica in these beds were originally about equal in amount. The limestone was removed by solution, leaving the insoluble silica.

Dense and glassy chert, which characterizes complete silification of the limestone, is found in and close to the centers of disturbance, as are the ore bodies; whereas, cotton rock, which is characteristic of partial chertification, is more common in the respective beds in places remote from the centers of disturbance.

#### Ore Horizons

Ore occurs in all the beds shown below the Cherokee shale horizon in fig. 2. The limestone beds were the favorable ore horizons. The limestone was replaced by the ore minerals, sphalerite and galena, and by the gangue minerals, jasperoid and gray dolomite.

The chert beds, D, F, L, and P, fig. 2, were not favorable ore horizons because they consisted of dense massive chert which resisted replacement by the sulphide and gangue minerals. However, in the mineralized areas along the shear trends, the chert horizons were intensely sheared and fractured, mak-



ing openings in which the ore minerals were deposited in sufficient quantity to make low grade ore. There is a thin 12 in. to 18 in. limestone horizon in D bed that occasionally is mineralized with enough sphalerite and galena to make ore.

Shale is found in structural and solution openings in the ore horizons. This shale has been referred to as residual and transported or introduced shale. The residual shale resulted largely from the solution of shaly limestone in J bed. The transported or introduced shale was carried into structural and solution openings by circulating water. Fowler and the writer<sup>1</sup> and McKnight<sup>2</sup> postulate the source for much of the transported or introduced shale as the overlying Cherokee formation and believe that the shale was carried down from the base of the Cherokee through openings, many of which are found in the slump areas. Much of this residual and transported shale was replaced by the sulphide and gangue minerals. The Cherokee shale horizon, however, contains very little mineralization because it was but little penetrated by the mineralizing solutions.

### Structure

The structure of the Tri-State field has been a controversial subject among geologists. The reader is referred to the discussions by Leith and Dake<sup>3</sup> and the article by Sales<sup>10</sup> for their opinions.

Structure is of the utmost importance because it was through the openings, caused by deformation of the beds, that the circulating mineralizing solutions reached the limestone horizons in which the ore bodies were deposited.

The three periods of deformation with which the ore bodies are associated are the following:

1. Pre-Warsaw Deformation: This period of deformation is relatively unimportant in the field as a whole.

2. Post-Warsaw-Pre-Pennsylvanian Deformation: Deformation during this period produced most of the regional structures with which the ore bodies are associated. The anticlinal structures shown in fig. 3, the pipe slump-structures shown in figs. 4 and 5, the Bendelari trough shown in fig. 1, and early movement in the Miami trough belong to this period.

3. Post-Pennsylvanian Deformation: Deformation which had started in Mississippian time and continued past Pennsylvanian time produced the "Miami Trough" or "Miami Fault" as it is commonly known. This fault, fig. 1, contains along its trend a series of elongated graben fault blocks, 500 ft to 1200 ft wide, and one half to three fourths of a mile long, that dropped vertically 100 ft to 250 ft. Ore bodies have been mined within and along the edges of this trough for a distance of eight miles.

### Structure Maps

Fig. 3 is a regional structure map that shows the structure on the top of L bed in the Bilharz Mining Company's leases. This area was selected as a type example because it shows unusually well the relations between the ore bodies and the anticlinal flexures.

Regional structure is shown by sea-level datum contours, at 5-ft intervals on the top or bottom of a bed that is below the slump horizons, and on which the most reliable information is available. The top of L bed, fig. 3, the top of N bed, fig. 5, and the top of O bed are the horizons often used for making structure maps.

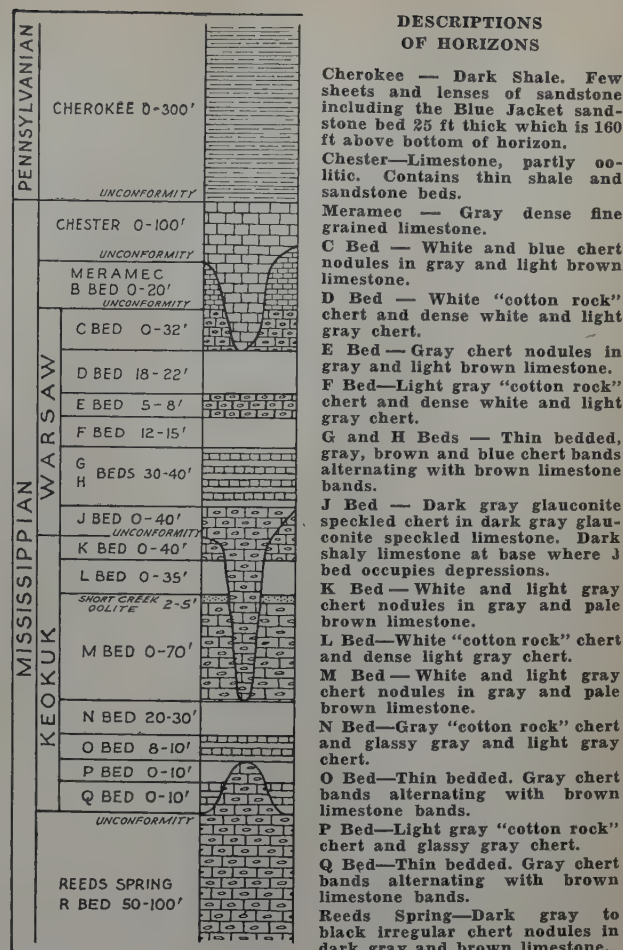


Fig. 2—Geologic section with descriptions of sedimentary horizons in the Picher field.

Structure maps are used in preliminary drilling campaigns to determine the locations and trends of structures and thereby help in making locations for new prospect drill holes. They are used in subsequent development work to determine the favorable parts of all the structures and to make sure they have been prospected in all favorable horizons.

### Regional Deformation

Regional deformation produced two types of structures, characterized first, by flexing of the beds, fig. 3; and second, by strong fault zones, such as the Miami trough, fig. 1. Flexing of the beds caused shearing across them and also caused movement on the bedding planes and unconformities.

Shearing across the beds is most prominent along a fold at the points of sharp flexing. The strong shearing in the flexures is usually at steep angles of 75° to 90° to the bedding, cuts across one or more beds, and is often continuous a mile or more in one direction. Ore bodies following the shearing are referred to as "runs," are long and narrow, vary from 5 ft in height in weak shearing to 100 ft or more in strong shearing, and average approximately 20 ft.

Movement on the bedding planes and unconformities opened them to the mineralizing solutions. In the horizons above N bed, movement on unconformities and bedding planes was mostly confined to areas within the shear zones, whereas in the horizons below N bed, such as Q bed and the un-

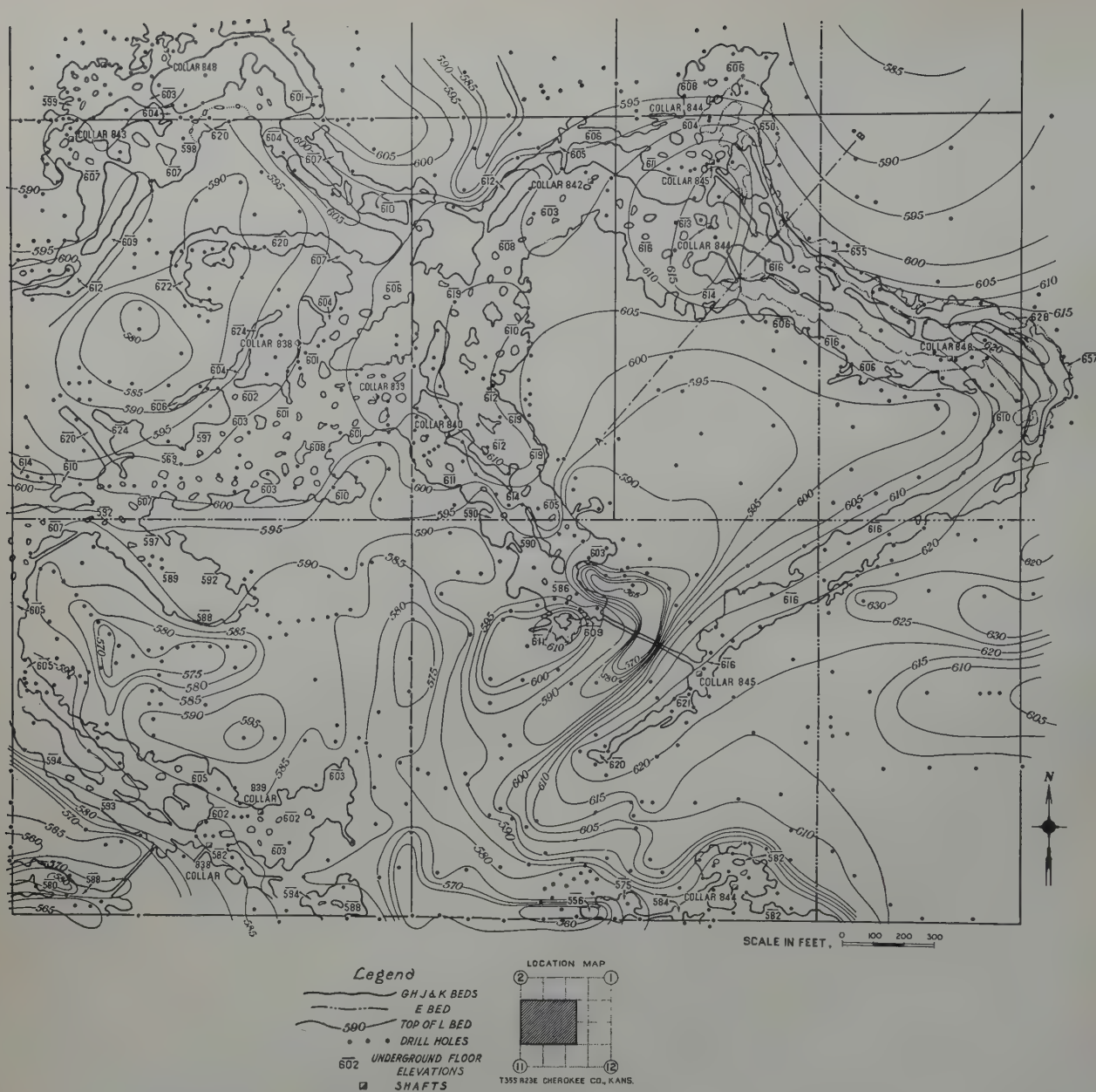


Fig. 3—Structure contour map on top of "L" bed.

Bilharz Mining Co. leases.

Eagle-Picher Mining and Smelting Co., Geological Department.

conformity at the top of the "Reeds Spring" formation, movement was on broad flat flexures where there was little relief of the stresses by shearing across the beds. The ore bodies in O bed, which is known as the "sheet ground" horizon, are characteristic of this type of deformation. They average about 10 ft high and often occupy areas 100 acres or more in lateral extent.

Anticlinal, monoclinical, and synclinal structures contain ore bodies along the strong shear zones. Anticlinal structures are the most important structures in which the ore bodies are found. Most of the monoclinical structures are really parts of large anticlinal structures and can be largely included with them. Synclinal structures, while important in many parts of the field, are often barren and much less mineralized than the anticlinal structures.

In places where a mineralized syncline is flanked

by mineralized anticlines, it is common to find M bed mineralized only in the syncline and the upper beds G-H and K mineralized only in the anticlines.

### Shale Maps

Shale maps are made with 5-ft contour intervals on the bottom of the Cherokee shale horizon. They outline the post-Cherokee-age slump areas which formed because of settling and caving of the beds into solution made openings in underlying limestone horizons. Shale maps were used by engineers and geologists in the early development of the Picher field when it was observed that ore bodies often occurred around the outer part of the slumps, or "shale sinks" as they were locally known. It is a simple matter to contour the bottom of the Cherokee shale because it is easily identified in the churn drill cuttings and its depth is recorded in the many



drill hole logs. Shale maps are also useful in showing whether the structures in the Mississippian horizons formed before or after the Cherokee shale was deposited.

### Slump Structures

Slump structures are secondary structures found throughout the mining field on the trends of strong regional structures. They resulted from the caving of beds into large solution-made openings in underlying limestone horizons.

The slumps formed before, during, and after the period of sulphide mineralization, and those that are mineralized often contain ore bodies from the base of the slump zone to its top. Usually the most favorable part of a slump area for sulphide mineralization is its outer edge, where favorable openings were produced when the slump block parted en masse from the surrounding rock.

The graben fault blocks in the Miami trough are slump structures, but are only partly the result of solution of underlying limestone beds. They are largely the result of regional forces that separated the fault walls and allowed long narrow blocks to drop within the fault zone.

### Pipe-Slumps and Circular Shearing

Figs. 4 and 5 illustrate two pipe-slumps that occur in the Eagle-Picher Mining and Smelting Company's Webber mine. They are cylindrical in shape, are approximately 100 ft in diam, and are 300 ft or more in height. They parted from the surrounding rock on vertical-circular shearing and dropped en masse approximately 30 ft. As shown by the contours in fig. 5, they formed near the centers of basins in a synclinal flexure. Ore bodies occur in *M* bed around the pipes. The north pipe is not mineralized in *M* bed and the Short Creek oolite limestone at the top of *M* bed is exposed a few feet above the floor of the mine workings. The south pipe is partly mineralized in *M* bed and contained ore in the outer rim. Drilling has not been deep enough within and around these pipe-slumps to determine their depth and the horizon beneath them in which, presumably, openings formed to allow them to slump. The writer believes that the weight of the pipes caused the basins to form over solution-made openings, and, as the basins formed, tension forces were set up around the areas of the pipes. The resolution of the forces, the weight of the pipe, a compressive force, and the tension forces set up in the area around it, caused each pipe to part on circular shearing from the surrounding rock and slump into a solution-made opening.

Pipe-slumps are a type of structure that can seldom be observed, especially in vertical section, and for this reason there is but little reference to them in the literature. These structures are important ore-bearing structures, not only in the Picher field but throughout the Tri-State district. They formed in both pre-Pennsylvanian and post-Pennsylvanian time. The pipe-slumps of pre-Pennsylvanian age, those shown in figs. 4 and 5, extend from below *O* bed to the Cherokee shale. The pipe-slumps of post-Pennsylvanian age extend into the Cherokee shale, and a few that can be observed underground formed above *N* bed where the solution of limestone, largely in *M* bed, allowed the overlying horizons to slump.

In some pipe-slumps the zone of vertical-circular shearing and the pipe both contained ore which was

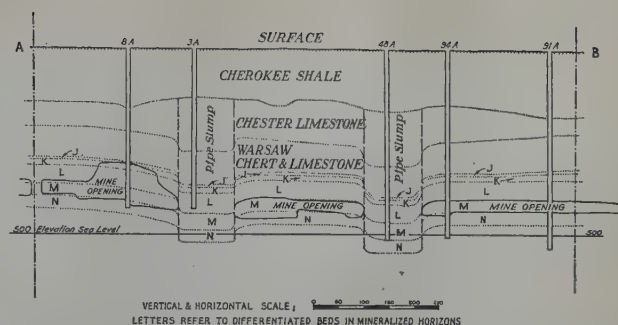


Fig. 4—Vertical section A-B, Webber mine, showing pipe slumps.

mined as one cylindrical orebody, whereas in some others the zone of vertical-circular shearing contained the ore which was mined as a circular ore body around the pipe which contained little or no ore.

Vertical-circular shearing occurs on small domes in the Picher field and probably is the result of regional deformation which forced the beds upward to form the domes. This contrasts with vertical-circular shearing resulting from slump of the beds into an opening. The forces in both cases acted in much the same manner; the beds were forced up into a dome in one case and forced down into a basin in the other. In both cases tension forces were set up around the areas of the pipes and the resolution of vertical compression forces with horizontal tension forces caused vertical-circular shearing.

Man-made pipe-slumps, one of which occurs at Negaunee, Mich.,<sup>11</sup> are similar to those that formed in nature. They are circular, extend several hundred feet to the surface, and formed over large undercut areas that were allowed to cave.

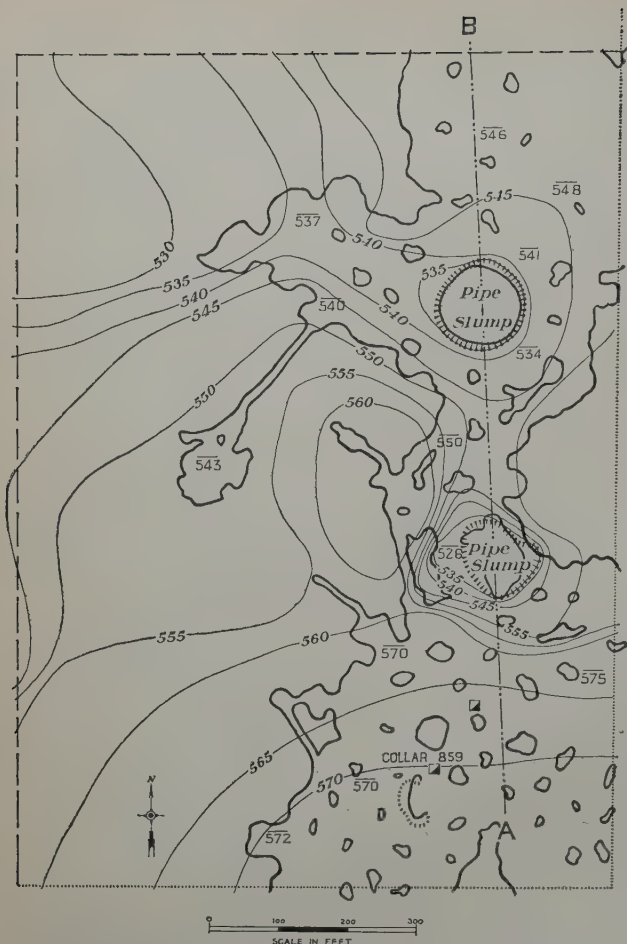
### Chimney Ore Bodies

Several chimney ore bodies have been mined in the Picher field. As a rule they formed in pipes or on circular shearing and have been mined as continuous vertical orebodies 30 ft to 100 ft or more in height and are associated with structural deformation of different ages. While most of them are found extending above the main ore horizon of an area, a few extend below.

### Residual Material and Breccia

Residual material such as chert nodules, limestone boulders, shale and clay, together with introduced material such as transported shale and a little sand collected over the bottoms of the large solution-made openings in the limestone beds. Where the overlying beds settled or collapsed into these openings, the residual and introduced material was compressed by the weight that crushed the loose chert and limestone boulders and squeezed the shale and clay into available openings. This crushed residual material of the premineral slumps became mineralized and the shale, clay and limestone were replaced by ore and gangue minerals which cemented the unreplaced shattered chert to form breccia.

Breccia, resulting from both regional and slump deformation, consists of chert fragments in a matrix of one of the gangue or ore minerals or combinations of them. Jasperoid is the most common of the matrix minerals and in the jasperoid area, fig. 6, is



WEBBER MINE near TREECE KAS.; in NE $\frac{1}{4}$  SE $\frac{1}{4}$  of Sec. 12-T35S R23E.  
Contours on top of "N" Bed. Mine Workings in "M" Bed.

**Fig. 5—Structure contour map for the top of N bed, Webber mine, showing locations of pipe slumps.**

the exclusive matrix mineral except for calcite and occasional minor amounts of the ore minerals.

Calcite breccia is common because calcite filled openings in shattered zones and cemented broken rock fragments that resulted from deformation throughout the field at the close of the period of sulphide mineralization.

Under some of the slump areas there is an overlapping laterally of premineral and postmineral slumping. The residual material that was under slump blocks before and during the period of sulphide mineralization was mostly mineralized and changed to breccia. The residual material that collected in openings in the beds after the period of mineralization, is of course, not mineralized and remains unaltered.

#### Mineralization Maps

Mineralization maps, fig. 6, are made on plan maps of the underground workings and show the shearing and respective areas of mineralization in and around the shear zones. They are usually made on one mineralized limestone bed, but occasionally, when two or more contiguous limestone beds are mineralized, they are grouped and one map serves them all. Mineralization maps have been used in the office of G. M. Fowler at Joplin for many years, and maps published by the U. S. Geological Survey<sup>9</sup> under the direction of E. T. Mc-

Knight show dolomite areas in parts of the Picher field.

The mineralized zones are divided into three parts depending on the prominence of certain minerals in each part. These parts are referred to as the dolomite, the sphalerite-galena and the jasperoid areas on the mineralization maps.

The dolomite area occupies the inner part of the area of mineralization. It contains gray, coarsely crystalline dolomite that replaced the original limestone and also contains pink dolomite crystals that deposited in openings. Where the dolomite area is large or where mineralization was weak, the center of the area frequently contains much unreplaced limestone.

The sphalerite-galena area is the ore area and eventually is all mined. It lies between the dolomite and jasperoid areas but also contains both dolomite and jasperoid. It occupies most of the area within the mine workings in fig. 6.

The jasperoid area makes a fringe around the sphalerite-galena area, is the outer part of the mineralized area, and consists almost entirely of jasperoid and chert. It varies from a few feet to several hundred feet in width and its outer edge is bordered by limestone areas.

There are many places in the mines where the sphalerite and galena mineralization pinches out along a shear zone, but the dolomite continues on one side of the strongest shearing in the zone and the jasperoid continues on the other. By following the strong shearing that separates the gray dolomite and the jasperoid areas, the sphalerite-galena mineralization often comes in again in a short distance.

#### Minerals

**Dolomite:** Two types of dolomite, the gray and the pink, occur in the Picher mining field. The gray dolomite replaced limestone and residual material, whereas the pink dolomite was deposited in openings.

The gray dolomite is coarsely crystalline. Individual crystals are pale pink in color, and the interstices between the crystals contain dark gray residual material from the limestone that was replaced which gives the rock as a whole a gray appearance. Gray dolomite is the principal mineral in the dolomite areas, fig. 6, where it was deposited early in the period of mineralization. It is found in the sphalerite-galena areas, but does not occur in the jasperoid areas.

Pink dolomite was deposited as crystals in openings, following the deposition of much of the gray dolomite. It occurs mostly in the dolomite areas but was also deposited contemporaneously with sphalerite, galena, and jasperoid crystals in openings in the sphalerite-galena areas.

**Jasperoid:** Jasperoid is the most abundant mineral found in the Picher field that was deposited during the period of sulphide mineralization. It is found in all three areas. It was practically exclusive in replacing limestone and other material in the jasperoid area where the sphalerite-galena content is too low grade to mine. It was deposited contemporaneously with sphalerite and galena in the sphalerite-galena area, and in some intensely mineralized areas it replaced gray dolomite in the dolomite area late in the period of sulphide mineralization. The jasperoid that replaced other minerals varies in color from dark gray and dark





Fig. 6—Mineralization and shearing map for G-H, J and K beds.

Bilharz Mining Co. leases.

Eagle-Picher Mining and Smelting Co., Geological Department.

brown to black, consists of microcrystalline quartz crystals, and breaks with a splintery or conchoidal fracture. The jasperoid that deposited in openings consists of dark quartz-crystal-aggregates, in which the crystals, though minute, are visible with a hand lens, and in places are intergrown with sphalerite, galena and dolomite crystals.

A form of jasperoid often referred to as "sand" occurs in places where microcrystalline quartz crystals had only partly replaced the original limestone, and subsequent solution of the residual limestone by circulating underground waters left the quartz crystals as a porous, loosely knit mass in which fossil outlines are common. Much of this material resembles loose sandstone and crushes easily into sand-like particles, which is the reason it is referred to as sand. This type of jasperoid grades gradually to the more massive types of jasperoid where mineralization was more intense.

**Sphalerite:** Sphalerite is most plentiful along the shear zones that acted as channels for the ore-bearing mineralizing solutions. It diminishes in quantity away from the shear zones and disappears almost entirely toward the centers of many of the dolomite areas and toward the outer parts of the jasperoid areas. It replaced limestone, shale and dolomite but did not replace chert or jasperoid. It was contemporaneous with jasperoid and gray dolomite in replacing limestone, shale, and clay in the matrix of residual material, and later in the period of mineralization it was deposited as crystals in openings.

**Galena:** Galena, usually in small amounts, was deposited contemporaneously with the sphalerite. Much of the galena was deposited late in the period of sulphide mineralization. It occurs more abundantly in the outer edges of the sphalerite-galena areas and in the top bed of a mineralized zone, such



as *E* bed, fig. 7. It is found as crystal growths in openings on pink dolomite and sphalerite crystals. It replaced limestone, shale and dolomite but did not replace chert or jasperoid. It was contemporaneous with sphalerite, gray dolomite, and jasperoid in replacing limestone, shale, and clay in the matrix of residual material.

**Marcasite:** Marcasite is common but is not plentiful. It was deposited late in the period of sulphide mineralization. It is most commonly found in the outer edge of the sphalerite area and is often associated with shale. It occurs as crystals on other minerals in openings and often makes a coating that caps and completely covers other crystal aggregates. Massive marcasite is found occasionally in the outer edges of the ore bodies. Rarely, botryoidal marcasite is found coating calcite crystals.

**Pyrite:** Pyrite occurs sparingly and is usually found associated with marcasite. Both minerals were deposited at the close of the sulphide period of mineralization. A small amount of pyrite was deposited early in the period of sulphide mineralization and is more common in *J* bed.

**Chalcopyrite:** Chalcopyrite occurs in openings as small crystals commonly found on the inner edge of the sphalerite-galena area and in the dolomite area on sphalerite and dolomite crystals. It has been found sparingly in a few mines in the Picher field as massive chalcopyrite that was deposited contemporaneously with sphalerite.

**Enargite:**<sup>12</sup> Minute enargite crystals are found very sparingly on the inner edge of the sphalerite-galena areas and within the dolomite areas where it is found on pink dolomite and chalcopyrite crystals.

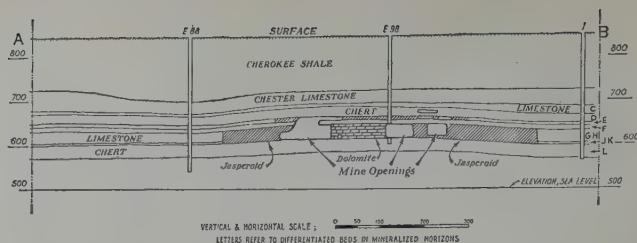
**Quartz:** Macroscopic quartz crystals were mostly deposited at the close of the period of sulphide mineralization. The microcrystalline quartz is the jasperoid and it was deposited throughout the period of mineralization. Clear quartz crystals ½ in. or more in thickness are found occasionally lining vugs and openings in *O* bed. Minute but visible quartz crystals commonly line small openings or fractures in chert beds and small quartz crystals are found as coatings on dolomite, jasperoid, sphalerite and galena.

**Calcite:** Calcite is the latest mineral that was deposited and is found as crystals lining caves and openings in the mineralized area. It is found as filling in shattered zones that formed in the ore bodies after the dolomite, jasperoid, and ore minerals had been deposited, and it is most abundant in the jasperoid area. Occasionally, marcasite crystals can be seen in calcite crystals.

### Paragenesis of the Minerals

The relative order of deposition of the minerals in the Picher mining field as interpreted by the writer is shown in the paragenetic diagram, fig. 8.

There is a difference of opinion among geologists as to replaceability and age relations between jasperoid and the sulphide minerals sphalerite and galena. Many geologists are of the opinion that jasperoid, dolomite, sphalerite, and galena were deposited contemporaneously. Others<sup>13, 14</sup> who studied samples collected for the purpose of determining the paragenetic relations are of the opinion that the deposition of jasperoid began before any sulphides were deposited, that all the sulphides are younger



**Fig. 7—Vertical section A-B showing mineralization in limestone horizons in Bilharz Mining Company's mines.**

than the jasperoid, and that much of the sulphide material in the jasperoid occurs as automorphic crystals that replaced the jasperoid.

The writer believes that the dolomite, jasperoid, sphalerite, and galena were deposited contemporaneously and that the sphalerite did not replace the jasperoid for the following reasons:

First, Smith and Siebenthal<sup>15</sup> show from a microscopic study that the general process of replacement of the limestone by the jasperoid started with the deposition of a few scattered crystals of quartz in the limestone, and with increase in proportion of quartz the limestone decreased until the rock consisted of a granular aggregate of quartz crystals with scattered ragged grains of calcite, residual from the limestone, which finally disappeared. It was noted that sphalerite was found in the limestone with the first appearance of the quartz but was not found in limestone wholly free from quartz, which led to the conclusion that the sphalerite and quartz developed simultaneously. A micrograph of sphalerite in jasperoid and dolomite shows sphalerite crystals with ragged boundaries in the jasperoid, and crystal outlines in the dolomite, indicating interference of crystal growth by the jasperoid.

Second, jasperoid that was deposited in openings as quartz-crystal cluster-aggregates is often intergrown with small sphalerite crystals showing contemporaneous deposition, each capping the other as the growth proceeded.

Third, the glauconite particles which vary from pin point to sand grain size and which are scattered throughout the limestone in *J* bed were not replaced by the sulphide minerals, the gangue minerals, or the chert. Ore specimens that contain considerable glauconite show it concentrated around the sphalerite, galena and dolomite crystals, and scattered through the jasperoid, indicating that as the dolomite, sphalerite and galena crystals grew in the limestone, the glauconite was mostly pushed aside and the microscopic jasperoid crystals were subsequently deposited around it. A few particles of glauconite can occasionally be seen by magnification in the sphalerite and dolomite crystals but rarely in the galena crystals from *J* bed. If sphalerite had replaced jasperoid in *J* bed, it would contain considerable glauconite.

Fourth, as observed and agreed to by all geologists, the chert which was deposited first was not replaced by the ore or gangue minerals. Chert and jasperoid are chemically the same; both are silica. The difference is in crystallization, the chert being cryptocrystalline and the jasperoid microcrystalline. A chemical process that would affect one should have a similar effect on the other.

Fifth, shearing and shattering in the areas of mineralization continued throughout the period of



sphalerite mineralization, producing new fissures and openings which were repeatedly filled with ore minerals, sphalerite and galena, and gangue minerals, jasperoid and dolomite. This repetition of deposition of sphalerite and galena indicates that they were deposited both early and late during the period of mineralization.

From the foregoing described occurrences of the minerals, both as to time and place of their occurrence, it can be seen that they lend themselves to a classification that can be used to make mineralization maps, fig. 6. These maps separate the mineralized and unmineralized areas of the particular horizon, or horizons, being investigated and divide the mineralized area into areas characterized by the prominence and presence of certain minerals.

The objective of the mineralization maps is, of course, to help find ore bodies. When prospecting an area by churn-drilling, they outline the dolomite, sphalerite-galena and jasperoid areas and show whether or not the sphalerite-galena area has been thoroughly prospected; and in the mines where both the sphalerite and galena mineralization, on the trend of an ore body, becomes weak or plays out, they indicate the most favorable place in the mineralized area to prospect for them.

### Acknowledgments

This paper was prepared at the request of Elmer Isern, President of the Eagle-Picher Mining and Smelting Co.

The writer wishes to acknowledge the assistance of R. K. Stroup, Chief Engineer of the Eagle-Picher Mining and Smelting Co. and the personnel of the Engineering Department, for help in preparing the maps.

P. K. Hurlbut, Jr., assisted the writer in mapping the geology of much of the area described and offered suggestions.

O. M. Bilharz, graciously gave the writer permission to publish geologic maps of his mines.

George M. Fowler, with whom the writer was co-author of several published papers on the Tri-State district, offered criticisms and suggestions, but may not necessarily agree with some of the statements.

### References

- <sup>1</sup> G. M. Fowler and J. P. Lyden: The Ore Deposits of the Tri-State District. *Transactions AIME* (1932) **102**, 206-251.
- <sup>2</sup> G. M. Fowler and J. P. Lyden: The Ore Deposits of the Tri-State District. (discussion) *Economic Geology* (1933) **28**, 75-81.
- <sup>3</sup> G. M. Fowler and J. P. Lyden: The Miami-Picher Zinc-Lead District. (discussion) *Economic Geology* (1934) **29**, 390-396.
- <sup>4</sup> G. M. Fowler and J. P. Lyden: Sequence of Structural Deformation in the Oklahoma Mining Field. *Transactions AIME* (1934) **15**, 415-418.
- <sup>5</sup> G. M. Fowler and J. P. Lyden: The Ore Deposits of the Tri-State District. (discussion) *Economic Geology* (1935) **30**, 565-575.
- <sup>6</sup> G. M. Fowler, J. P. Lyden, F. E. Gregory, and W. M. Agar: Chertification in the Tri-State Mining District, (with discussion). *Transactions AIME* (1935) **115**, 106-163.
- <sup>7</sup> W. A. Tarr: The Miami-Picher Zinc-Lead District. *Economic Geology* (1933) **28**, No. 5, 463-479.
- <sup>8</sup> A. W. Giles: Boone Chert. *Geol. Soc. Amer., Bulletin* (1935) **46**, 1815-1878.

PRE-SULPHIDE PERIOD	MISSISSIPPIAN	TERTIARY?
LIMESTONE		
CHERT	---	
GLAUCONITE (J BED)	-	
SULPHIDE PERIOD		
GRAY DOLOMITE		---
JASPEROID		---
SPHALERITE		---
GALENA		---
PINK DOLOMITE		---
CHALCOPYRITE		---
MARCASITE		---
PYRITE	---	---
ENARGITE		---
QUARTZ		---
CALCITE PERIOD		
CALCITE		---

Fig. 8—Paragenetic diagram for minerals in the Picher Mining Field.

<sup>9</sup> E. T. McKnight, R. P. Fischer, C. C. Addison, K. R. Bowie, J. M. Thiel, M. F. Owens, Jr., and F. G. Wells: Maps Showing Structural Geology and Dolomitized Areas in Part of the Picher Zinc-lead Field. Oklahoma and Kansas. U. S. Geol. Survey, Tri-State zinc-lead investigations preliminary maps 1-6.

<sup>10</sup> R. H. Sales: The Ore Deposits of the Tri-State District. *Economic Geology* (1933) **28**, No. 8, 780-786.

<sup>11</sup> C. W. Allen: Subsidence Resulting from Athens System of Mining at Negaunee, Michigan. *Transactions AIME* (1934) **109**, 105-202.

<sup>12</sup> A. L. Ransome: Enargite and Polumbojarosite at Picher, Oklahoma. *American Mineralogist*. (1935) **20**, 799-895.

<sup>13</sup> Edson S. Bastin: Contributions to a Knowledge of the Lead and Zinc Deposits of the Mississippi Valley Region. Geological Society America, Special Paper 24.

<sup>14</sup> John Ridge: The Genesis of the Tri-State Zinc and Lead Ores. *Economic Geology* (1936) **31**, 298-313.

<sup>15</sup> W. S. T. Smith and C. E. Siebenthal: Description of the Joplin District in Geologic Atlas of the United States, U. S. Geol. Survey, Folio 148 (1907) 20 pp.

<sup>16</sup> C. H. Behre, Jr., A. V. Heyl, Jr., and E. T. McKnight: Zinc and Lead Deposits of the Mississippi Valley. 18th International Geological Congress, London, 1948, p. 46-52.

<sup>17</sup> J. S. Brown: Ore Genesis (1948) 160-172. Hopewell, N. J. The Hopewell Press.

<sup>18</sup> W. H. Emmons: The Origin of the Deposits of Sulphide Ores of the Mississippi Valley. *Economic Geology*, **24**, 221-271.

<sup>19</sup> G. M. Fowler: Oil and Oil Structures in Oklahoma-Kansas Zinc-Lead Mining Field. *Amer. Assoc. Petr. Geol., Bulletin* (1933) **17**, 1436-1445.

<sup>20</sup> G. M. Fowler: Structural Control of Ore Deposits in the Tri-State Zinc and Lead District. *Engineering and Mining Journal* (1938) **139**, No. 9, 46-51.

<sup>21</sup> G. M. Fowler: Tri-State Geology. *Engineering and Mining Journal* (1943) **144**, No. 11, 73-79.

<sup>22</sup> W. H. Newhouse: The Temperature of Formation of the Mississippi Valley Lead-zinc Deposits. *Economic Geology* (1933) **28**, 744-750.

<sup>23</sup> C. E. Siebenthal: Mineral Resources of Northeastern Oklahoma, U. S. Geol. Survey, *Bulletin* 340 (1908) 187-228.

<sup>24</sup> Samuel Weidman: The Miami-Picher Zinc-lead District, Oklahoma. Oklahoma Geol. Survey, *Bulletin* 56 (1932) 177 pp.

# Mechanized Cutting and Face Stripping in the Ruhr

by R. R. Estill

THE rank of the Ruhr coal ranges from a high volatile bituminous coal to an anthracite, depending to some extent on the original depth of the seam. The average Ruhr coal corresponds to a soft bituminous American coal of a coking quality. The average thicknesses of individual coal seams being mined are also comparable (59 in. against 65 in. in the United States). However, consideration of seam conditions and mining conditions other than those just mentioned emphasizes differences rather than similarities with United States soft coal. In general, the Ruhr seams now being mined are much more folded and inclined than American seams. Dips of 20° and 30° are common in seams now being worked, and 30 pct of the coal reserves in the district are in seams dipping more than 35°. Only on the tops and bottoms of folds do we find rather flat coal seams. In addition to the folding there is extensive displacement by cross faulting plus a certain amount of strike faulting of an overthrust nature, which results locally in doubling or omission of seams. Because of the long history of mining in the Ruhr, nearly all coal lying near the surface has long since been mined out, and we find that the average depth of mining is at present about 2300 ft below the surface. Deep mining, folding, and faulting result in seam conditions requiring a great deal more roof support than one finds in American soft coal mines. In fact only in the anthracite district and the Rocky Mountain and Pacific coal fields do we find somewhat similar conditions. It is easy to say, therefore, that the problem of mechanization of coal cutting and loading in the German mines is quite different from that which we have so effectively met in America with our mobile cutters and loaders, duck bill loaders, and a room and pillar system of mining our drift and slope mines.

Partly because of more limited coal reserves, the traditional German mining system is largely the longwall method, which gives an almost complete coal recovery. Backfilling must be extensively practiced to protect the longwall faces, the over and underlying seams and workings, and especially the surface industrialized areas and barge canals. The German engineers have accordingly concentrated their efforts on the design of cutters, loaders, and conveyors suitable to longwall methods rather than room and pillar methods.

Undercutters with cutter bars like American models have been in use in the Ruhr since well before World War II. In 1941 they accounted for 8.5 pct of the production. This percentage, of course, includes coal which was undercut but nevertheless had to be broken down with air hammers or with explosives. The most common of these cutters is the

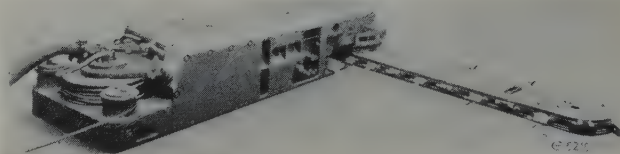


Fig. 1—Standard short wall bottom cutter (air operated).

Eickhoff Standard cutter (see fig. 1). This machine does about 95 pct of the undercutting in the Ruhr today, and is available with either compressed air or electrical power and in at least four different sizes. A variation of the cutter is this one with two cutter bars (fig. 2). At the end of 1947 about 200 of these machines and similar cutters were accounting for 13.2 pct of the total production, a production which was, however, only 60 pct of the 1941 production rate, so that the actual cutter tonnage was only up to a small amount over 1941.

In 1941 about 3 pct of the production was accounted for by shearing machines making their cut perpendicular to the longwall face. They were similar to those used in the States. These machines are today considered obsolete and now account for only 0.7 pct of the total production. They are located at only a few mines and at present do not seem to have much of a future in the Ruhr.

For the future, the Ruhr miner is looking forward to rather extensive mechanization of face work, with two major types of equipment being developed almost simultaneously. On one hand there is the development of cutter loaders for use in relatively hard coal. They represent the further extension of ideas developed after relatively long experience with the Eickhoff cutter. On the other hand there has been since 1942 an intense interest in the Ruhr in the development of face-stripping methods, particularly by the Kohlenhobel (coal plow) and its modification.

At the end of 1947 these cutter loaders, Kohlenhobels and scrapers together were actually accounting for only about 1.4 pct of total production while air hammers still broke 77.1 pct and as much as 1.2 pct was actually broken by hand picks. However,

R. R. ESTILL, Member AIME, formerly American Chairman, Ruhr Control, United States Military Government, is now General Superintendent, United States Coal and Coke Co., Mexico, Ky.

AIME New York Meeting, February 1950.

TP 2988 F. Discussion (2 copies) may be sent to Transactions AIME before Jan. 31, 1951. Manuscript received Dec. 28, 1949.

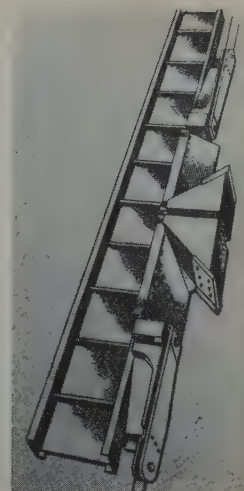


the Committee on Mechanization of the German Coal Mine Administration has recently estimated that it would ultimately be possible to reduce the tonnage of coal production by air hammers to less than 50 pct. Whether or not the air hammers will largely be replaced by stripping machines or by cutter loaders remains to be seen, but it is evident that the most promising machine is at the moment considered to be the Kohlenhobel. The Kohlenhobel (fig. 3) is quite successful in continuous soft coal seams with fairly good geologic conditions. The field of the cutter loaders is essentially in continuous harder coal seams. The air hammer and where necessary, drilling and blasting must continue to account for about half of the Ruhr production because so many seams are disturbed by faulting or other geologic phenomena. One member of the Committee of Mechanization has estimated eventually perhaps 25 pct of Ruhr coal could be mined by the Kohlenhobel and its modifications or by scrapers, and another 25 pct by cutter loaders, cutters and by blasting. However, in the case of coal seams of medium hardness, which constitute 20 to 30 pct of the total, there will be some competition between stripping and cutting machines.

### Cutter Loaders

For a closer look at the trends in design since the beginning of fully mechanized coal production in the Ruhr, we turn first to the cutter loaders since they were the first to be developed. In about 1938 and 1939 there was an industry-wide, concerted effort to achieve mechanization. The prize winning machine in a competition sponsored by the Bergbau-Verein (coal mining trade association) was the Eickhoff Rhine-Prussian cutter loader. This machine consists of a conventional cutter bar above which is a revolving toothed bar of readily adjustable height. In operation, the machine undercuts for about 18 in. and is then stopped to allow the toothed bar to be swung upward in an arc, thus breaking off a block of coal the height of the bed, the depth of the cut, and 18 in. thick. The paddle-type loader moves this coal into the conveyor. This model runs on top of the conveyor, and advances itself by means of a self-contained winch. Some of the other models of the

Fig. 3—Standard Kohlenhobel (coal plow).



Eickhoff have had two cutter bars, the upper one adjusted to the height needed to trim the roof. In some other models the lower cutter bar makes a sharp upward turn at its outward extremity (fig. 4). Such a model, combined with an upper cutter bar (fig. 5) can of course cut a block of coal completely free from the seam. As employed to date, however, including this latest arrangement, (fig. 6) certain difficulties have been experienced with these machines in that the lower cutter bar does not completely undercut the seam but instead leaves a wedge or rectangle of uncut coal on the footwall. The machine has therefore had to be supplemented by hand work. The newest models have a down-bent lower cutter bar.

There are two ways of working a longwall face with one or more of the Eickhoff cutter loaders. The first method for working in soft coal uses two double chain flight conveyors. The forward or face conveyor is straddled by the machine, and conveys the coal away from the machine. The second conveyor is used only for backfill. The second method for working hard coal is to operate with only one conveyor, which unit must serve both for coal and backfill. In this method the machine does not straddle the conveyor.

These Eickhoff cutter loaders have required two machine operators, about five timber men, four backfill stowers, and four to five servicing the conveyors and winches. Efficiencies of 20 to 30 metric tons per face-man shift have been realized.

The Eickhoff cutter loaders have largely been designed to operate in seams thicker than 4½ ft. Another company, Deutsche Maschinenbau A. G., has been more active in designing cutter loaders for thinner seams, those down to about 3 ft in thickness. One of their DEMAG designs consists of the usual 6-ft cutter bar and paddle-type loader. The somewhat unusual factor is that it has a vertical shearer of adjustable height operating from the far end of the loader, like the English Meco-Moore machines which it closely resembles. This model skids itself between the face and the conveyor and does not leave a wedge of coal below the undercutter, but it cannot be used if the undercut coal does not immediately fall since there is no cutter at the hanging wall. Under the correct seam conditions these cutter loaders have shown comparably good performance, up to about 20 metric tons per manshift with an eight man crew.

In the field of cutter loaders the newest design is

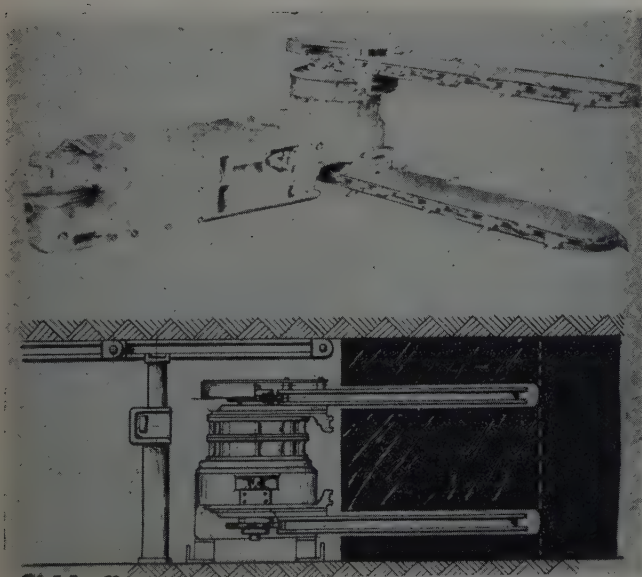


Fig. 2—Eickhoff dual bar cutter (straight bars).



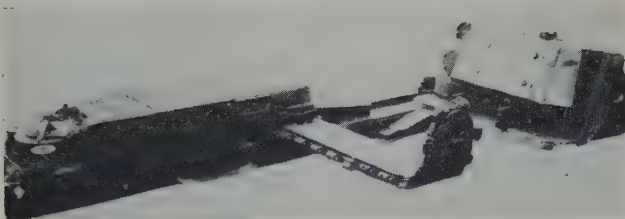


Fig. 4—Cutter loader with turned up cutter bar.



Fig. 5—Frame cutter loader.

the frame cutter (fig. 7). This frame cutter is still very much in the development stage, but experimental models are being watched with great interest by the Production Branch of the German Coal Mine Administration. The advantage of frame cutters and frame cutter loaders is considered to be that in hard coal, which will not fall by simple undercutting or even a double cut, or an undercut and a shear, the frame cutter is positive in that it completely surrounds its entire cut.

One of the Eickhoff machines under development designed to make a complete cut in a relatively thin seam of hard coal is a double-chain cutter with the roof cutter arm directly over the undercutting arm and with the far ends of both cutter arms connected by a revolving bar studded with spikes. Initial tests with this machine produced too much dust. However a modified version is to be tried in a different seam.

For cutters and cutter loaders in hard coal in steep seams, the Germans have turned to modifications of the foregoing types, since most of these have not been designed to work in seams steeper than 20°, or at the most 30°. One such cutter is currently being tried in the Dorstfeld mine. Here the dip of the seam is 60° to 70°. The cutter machine slides on its bottom and one side, its bottom being on the footwall and its side on the packing. Two ordinary chain-cutter arms do the cutting. The machine is moved along the face by winches on the cutter and in the upper gate road. In the present experiments this cutter works only from bottom to top, but it is planned to try it from top to bottom in order that packing operations can be carried on simultaneously. The angle of repose of the packing is of course a maximum to facilitate gravity movement of both coal and packing material.

Another cutter for a steep seam in hard coal is designed to operate under similar face conditions but instead of two chain-cutter arms it has one undercutting arm and one overlying spiked rotating bar with a cogwheel at the far end for breaking down the undercut coal.

Summarizing the German developments in cutters

and cutter loaders for use in seams of harder coal, the only machine in widespread use in the Ruhr today is the Eickhoff chain undercutter. A large variety of other cutters and cutter loaders are under development or testing. The tonnage of coal suitable for these coming cutters and cutter loaders amounts ultimately to perhaps 25 pct of the total Ruhr output. It is therefore expected that in the next five or ten years several of the types of machines we have just seen may come into much more general use.

### Face-Stripping Machines

There was little or no interest in the Ruhr area in the possibility of mechanized face stripping until 1942 when the Kohlenhobel (coal plow), fig. 3, at Ibbenburen was described and the results of experiments with it became generally known. Ibbenburen is a small hard coal district in Lower Saxony about 100 miles northeast of the Ruhr. The Kohlenhobel was given immediate attention after that in the Ruhr, and has today forged far ahead of the cutter loaders in total output. One reason for the interest is that it is suitable for the softer coal seams of a wide range in thickness. These seams are now largely worked by hand-held air hammers. Another reason for the popularity of the Kohlenhobel is its simplicity of construction and its lack of moving parts. Of course the most important reason for its popularity is that it enables the output per face-man shift to be approximately doubled. My last report indicated there are 30 to 35 Kohlenhobels in operation, and they produce about 4 pct of the total output.

Actually the development of the Kohlenhobel at Ibbenburen in 1941 and 1942 was a rediscovery of a machine and principle patented by Wissemann in the Ruhr in 1912. The Wissemann machine was not as simple in construction, nor was it reversible, otherwise its action was about the same. The Kohlenhobel is shown in detail in fig. 8 where the upper sketch is the side view from the conveyor side and the lower sketch is the plan view. As the plow is pulled along the longwall face, the coal is split from the solid by the face *d*. The continual movement of the plow lifts the bottom coal upward along this face, fracturing the seam above the plow and breaking it down while at the same time plowing the coal over onto the conveyor.

The use of the coal plow of course creates some unusual problems and procedures. The solution of one of these problems is shown in fig. 8 where the plow is revealed as actually two plows joined in the middle. Experimentation proved the necessity of permitting a minimum amount of vertical movement to allow for irregularities in the bottom. No hori-

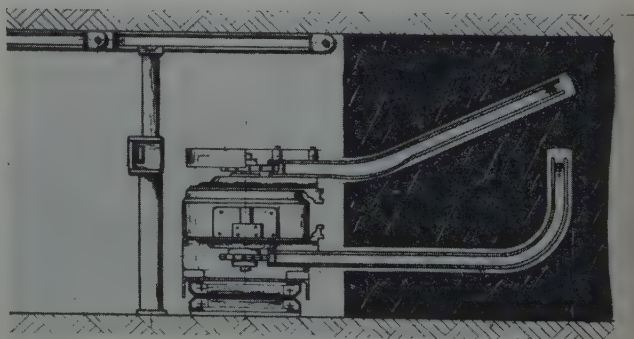


Fig. 6—Eickhoff dual bar cutter (bent bars).



zontal movement is permissible, however. Other matters of design of the plow itself, particularly those of the shapes, sizes and angles of the cutting edges, are now standardized to conform with the nature of the coal, roof, floor, and thickness of the seam. Two sizes of plows are in general use. The smaller plow is 1 ft 4 in. high, about 20 ft long and 2 ft wide and is for coal seams up to 4 ft thick. The larger plow shown here is 2 ft high, about 20 ft long, 2 ft 4 in. wide and weighs about 3 metric tons. The height of the discharge is usually about 11 in., which is just sufficient to feed directly into the usual armored double-chain conveyor.

The depth of the cut in the softer coals where most of the plows are in use has been more or less standardized at about 1 ft, although it has been found desirable to make only a 6 or 8-in. cut in thick seams or in somewhat harder coal. The cutting edges are ordinarily made of a nonalloyed steel of 350 Bhn. It is only recently that experiments have been under way with Widia alloy of 600 to 700 Bhn. It has been found advisable to keep a number of spare cutter edges on hand at the face and to watch the cutter edge carefully, otherwise a broken edge may jam the conveyor or cut the cables. The blades are adjustable for height so that in case of a weak bottom the plowbottom can be made quite flat and in the case of a solid footwall the blades can be lowered to permit a small pressure on the bottom.

Another adjustment of the plow which is variable with bottom conditions is the height of the idler for the cable connection. The more solid the bottom, the higher can be the cable connection and the less tendency will there be for the plow to climb in the seam. If a plow is used on a flat seam, a climbing tendency is uncommon, but if a plow is pulled downhill in an inclined seam it will occasionally climb. For this reason a coal plow used in an inclined seam generally cuts only on the up trip and merely cleans its cut on the down trip. When a plow encounters a section of coal which is harder than the average in the seam, the practice is to run it back and forth until it has been able to chop its way through into softer coal.

There are several variations of the standard plow, most of them designed for thinner seams or for somewhat harder coal seams. One such plow newly designed is the Stufenhobel (step plow) which makes cuts in only one direction but at two elevations rather than one (fig. 9). The upper cut is made first, and its butting blade is preceded by four under-



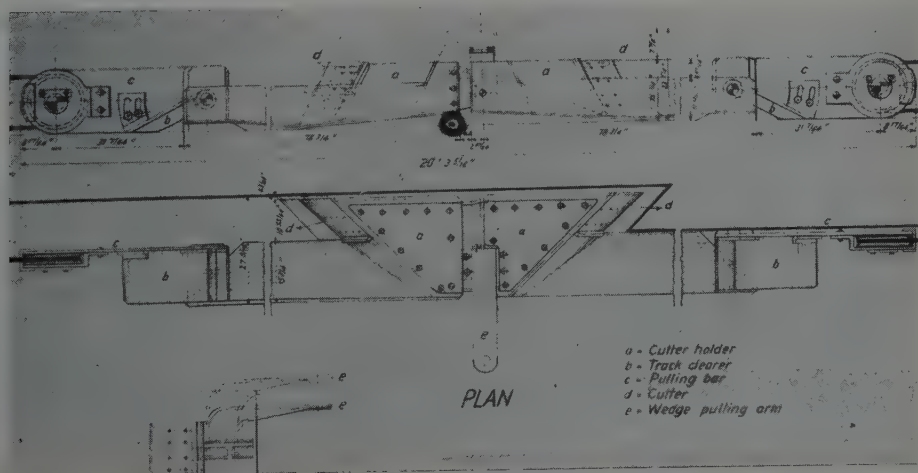
Fig. 7—Experimental frame cutter.

cutting chisels, each extending the cut of the preceding chisel from 5 to 7 to 9 and finally 18-in. depths. The upper cut is at the level of the top of the conveyor. The lower knife scrapes the bottom and elevates the bottom coal by a ramp to conveyor level as in the conventional Kohlenhobel. The Stufenhobel is made by Deutsche Maschinenbau A. G. (DEMAG) and its first tests have been successful. Its present tests are being conducted in somewhat harder, thinner coal than that in which a conventional Kohlenhobel is usually employed.

The Flottmann Co. in 1945 brought out what they call the Schlaghobel (striking plow) which is very much like the conventional coal plow with the exception that four air hammers are built into each vertical cutting edge, to permit the plow to chop its way through harder coal seams (fig. 10). This first Flottmann Schlaghobel developed some design difficulties. The two lower hammers customarily become jammed, leaving only the two upper still hammering, as a result of which the Hobel showed a strong tendency to climb. The newer models are better in this respect but still have difficulties. Some German engineers consider that the depth of cut, 1 ft 8 in. is too great for hard coal.

A very similar activated plow constructed by the Hauscherr Co. has not had difficulties with its three air hammers jamming, but it has a high air consumption.

The earliest and perhaps the most successful of the plows with an activated cutting edge was the so-called Schnell-Hauer (quick miner) produced by the Westfalia Lunen Co. in late 1942. This model operated in only one direction, making a cut 2½ ft deep. A vibrating action was given to the cutter



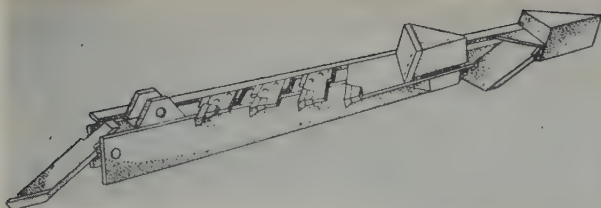


Fig. 9—Stufenhobel (step plow).

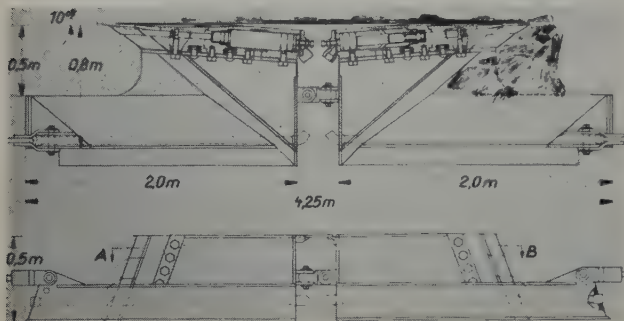


Fig. 10—Schlaghobel (striking plow).

edge, a series of chisels, by eccentrics, and the plow was able to reach a production of 360 tons per day in experiments in 1943. Encouraged by these experiments, the company developed a Schnellhobel designed to go in both directions (fig. 11). The eccentrics are located in the middle of the body (fig. 12). Unfortunately the results with this plow have been unsatisfactory so far and its further development is at the moment completely stopped. Its deep cut and the resulting large size of some of the broken coal caused both face-stripping and conveyor difficulties, and in addition the air consumption is too high.

Despite the unsatisfactory results today with coal plows of the activated-cutting-edge type, they are relatively certain in the future to receive further experimentation because of their promise in the harder coal seams.

Returning now to the conventional, proved, unactivated Kohlenhobel one can easily see that its use in a longwall face means that the mining practice with respect to conveyors, roof support, packing, and motor and winch installations is necessarily quite different from mining practice in a standard longwall face worked with the usual air hammers. An installation currently in use at the Friedrich Heinrich mine is shown in fig. 13. The coal plow is at the left end of the face. The winch positions are in the gateways, and the double-chain armored conveyor is illustrated in its position between the face or the Kohlenhobel on the one hand and the first series of props on the other. The proper anchoring of the winches, pullies, and deadmen has been a problem which has been fairly well worked out. With a cutting width of 1 ft and a rate of travel for the plow of 20 ft per min in coal of medium hardness, a winch must normally pull with a force of 5 to 8 tons and have a maximum pull of about 20 tons. The cables themselves must have a breaking strength of 50 to 60 tons. There has been some experimentation with chains rather than cables because of their lower elasticity and more positive pull.

The conveyors used along a coal plow face must

not only have a large capacity but they must be capable of withstanding lateral pressures of 2 tons or more. This is because the force which holds the Kohlenhobel into the face is transmitted to the Kohlenhobel through the conveyor, or in another sense, the Kohlenhobel is confined to the face by the rigid emplacement of the conveyor. Perhaps the most common type of conveyor in the Ruhr today is the shaking conveyor, but for a Kohlenhobel face or, for that matter a cutter-loader face, the armored double-chain conveyor (fig. 14) with bottom return meets the requirement of low height and the necessary strength to be scraped against or ridden upon. The principal difficulty with this conveyor is its high power requirement. Experimentation with this conveyor is at the moment on rolling rather than sliding crossbars between the chains and also with a belt in place of the chains and flights.

In order to move the conveyors sideways each time the Kohlenhobel has passed any given point and to anchor it there, several processes have been devised. The first and still most common system is that of air cylinders which press against the side of the conveyor every 20 to 25 ft. The next most popular system is the Ruckschlitten (sliding wedge). This sliding wedge is towed by a cable from the coal plow and presses on the conveyor as it slides along the front side of the front row of posts. Sometimes the sliding wedges are pulled by a special winch. This method has the advantage of maintaining a very straight longwall face. The disadvantage is that the wedge is inclined to knock out posts or push them back. Difficulties of this type have caused several mines to abandon this system.

Another system of advancing the conveyor was developed in one of the small sub-bituminous coal mines of the Bavarian Alps and has now been tried in the Ruhr with fair results. It consists of a simple pantograph braced against heavy cribbing. The miners can advance this simple lever system stepwise by hand tools.

Timbering and packing are considered essential to the safety of the Kohlenhobel front. Since there can be no post between the rear side of the conveyor and the coal face, a distance of about 3 ft 7 in., caps must be extended from the posts out over the conveyor and the plow space. The posts and caps usually required are steel or sometimes aluminum and are designed for easy attachment and detachment of the caps and for only the proper amount of give in the

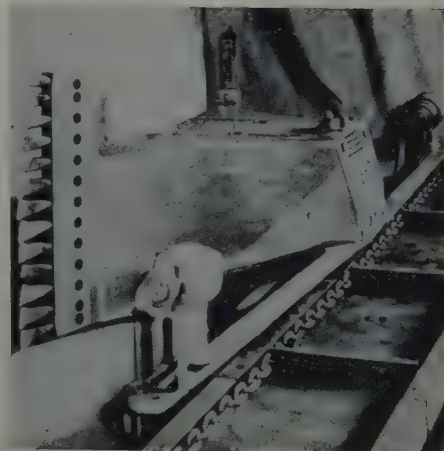


Fig. 11—Schnellhobel (quick miner).



posts. Backfilling at a Kohlenhobel face may be accomplished by dummy road packing if the roof will permit it. Solid packing is at present only feasible with pneumatic stowing since the double-chain conveyors do not permit slinger or hand stowing. Ruhr practice in pneumatic stowing is relatively efficient, and there is quite a bit of it done, approximately 12.9 pct.

The conventional Kohlenhobel and the other coal plows which we have discussed up to now are designed to be used in flat or nearly flat seams. The working of the coal plow in steep seams is dependent upon the development of other designs.

One such type of coal plow for steep seams has been designed by the Hausherr Co. This machine is lowered to its working position and anchored by compressed air jacks between the hanging and footwalls. A swinging arm with a plow head is set into operation by a compressed air cylinder. To date, as could be expected, its capacity is considered to be too small and its air consumption too large to be of real interest.

There were in the experimental stage several other types of plows for use in steep seams at the time the author left Germany in December 1948, but unfortunately he was unable to obtain any pertinent data prior to his departure.

### Scrapers

Two interesting developments in face-stripping methods which are quite different from the previously-mentioned coal plows are the Ibbenburen scraper train and the scraper train development at the Minister Achenbach and the Julia mines. Fig. 15 is a schematic view of the Ibbenburen installation designed for the purpose of mining coal seams only 18 to 22 in. thick with a wet roof. A 50-m-long system of scraper boxes, at each end of which a Hobel is attached, is pulled backwards and forwards along the coal face by means of two winches. The individual boxes of the train have hinged flaps which glide over the coal when moving away from the middle gateway conveyor but which move the coal to the middle gateway when the direction of cutting is reversed. Each half of the longwall face was 164 ft long. No attempt was made to support the roof or to backfill. The advance of the train into the face was regulated by the position of the pulley blocks on the side gateways. The speed of the train was about 30 ft per min and the height of the cutter head and boxes was 16 in. The weight of the complete train was about 10 tons, and the pull on the rope while cutting was normally 3 to 4 tons with a maximum of about 10 tons. The face was developed in such a way that the pitch of the seam helped hold the scraper train against the coal face. For thicker

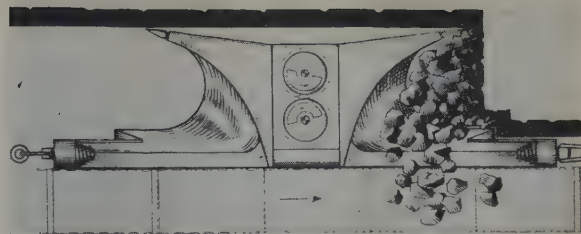
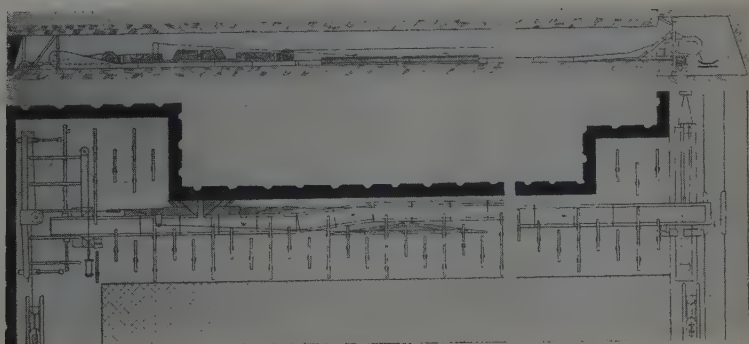


Fig. 12—Activated Kohlenhobel (coal plow).

seams the power requirement would be greatly increased and probably impracticable so that this scraper is especially for very thin seams. The particular seam at Ibbenburen was worked out by this method. The train is not now in use, but a future attempt would probably be made with longer faces since a large part of the objection to this method is the extraordinary gate road development necessary, especially as the gate roads have to be driven down the dip. The necessary rigidity of the train is another handicap.

Still another scraper train or scuttle scraper was in operation, the Minister Achenbach. It has advantages over the Ibbenburen scraper train in that fewer boxes are required and it can work longer and thicker seams and is very flexible. With respect to the ordinary Kohlenhobel, the Minister Achenbach scraper train has the advantage in that it can be used where the seam is offset by small faults or is somewhat folded or otherwise irregular on its footwall. In this system the boxes are pulled up and down the coal face with a stroke of about 80 ft by means of a double-drum winch. As the scraper boxes move away from the gateway in which the conveyor is located, the flap ends of each scraper box glide over the loose coal, while on the trip toward the conveyor they move this coal to a position where the next scraper box will pick it up on the next inward trip. The individual scraper boxes at Minister Achenbach are 8 ft long and 20 in. high with vertical cutting edges on each face side corner. The cutting depth is less than 3 in., but the travelling speed is comparatively high at over 3 ft per sec. The cable connections between boxes are arranged in such a way as to tend to pull the scrapers into the coal face when the cables are under tension. Guide rails are also used. Experience has shown that in the 5-ft seam at Minister Achenbach which has been worked continuously by such a scraper train for several years, the coal undercut by the scraper train has generally come down by itself. The average production has been 250 tons per shift. A 120 hp winch is required for this coal face, but the installation costs

Fig. 13—Longwall face being worked with standard coal plow.



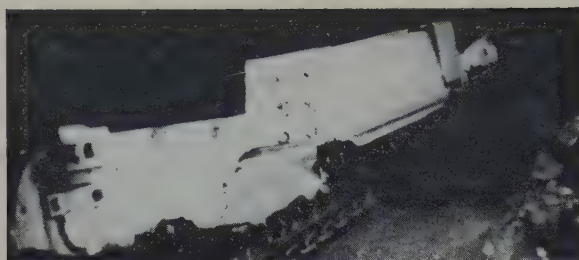


Fig. 14—View showing standard coal plow and armored conveyor.

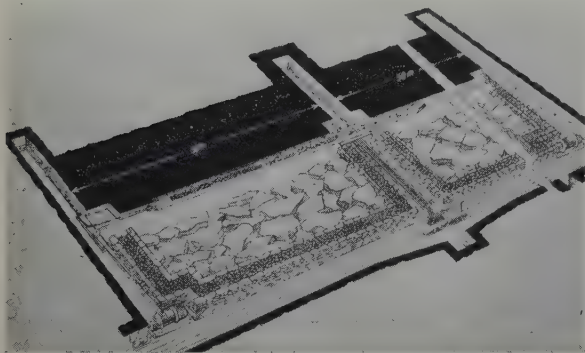


Fig. 15—Schematic view of scraper train loading coal.

are much lower than those for the Kohlenhobel with its expensive conveyor. And since this method is very adaptable to uneven seam conditions, to thin seams, etc., its use is being encouraged in other collieries such as Julia. There is no question, however, but that the standard Kohlenhobel, under the proper seam conditions, can give a much higher output.

Some efficiencies achieved in June and July 1948 by representative installations are given in table I.

Since the main subject of this paper has been German ideas and developments in coal cutters rather than machines for rock work, it will probably be sufficient to give the latter only a passing comment.

The Salzgitter loader, as many of you probably know, is copied almost directly from the Eimco shovel but is heavier. Most observations indicate that it is less efficient than the Eimco.

Another loader of a familiar type is that produced by Bergtechnik which resembles rather closely a track-mounted duck bill. The 3-m-wide duck bill is

Table I. Efficiencies of Face Stripping

(June to July 1948)

	Seam		Face Length, Ft	Metric Tons Coal Oper-ating Day	Man Shifts Per 100 Metric Tons <sup>a</sup>
	Thick-ness	Dip			
Standard Kohlenhobel					
Best	4.3 ft	25 to 30°	790	365	14.8
Typical	5.3 ft	0 to 5°	610	444	23.3
Activated Edge					
Hobel	5.0 ft	3 to 6°	660	252	29.9
Stufenhobel	2.8 ft	18°	880	228	30.0
Scraper Train	1.7-2.1 ft	10 to 15°	750	167	32.5
Steep-seam Hobel	2.5 ft	72°	310	47	16.4

<sup>a</sup> For all labor, including packing, timbering, conveying.

the head of a short shaking-conveyor which feeds an inclined belt to the wagons. No side movement is possible. A loading capacity of one ton per minute is claimed. From the front of the duck bill to the discharge point of the loading belt the length is about 50 ft.

The use of drilling frames to advance tunnels in rock has not been popular in the Ruhr district. Some of the frames run on rails bolted to the steel arches of the drift, high enough so that the frame, when in a horizontal position, will clear the loading machine. The lower middle portions of some drill frames were designed for easy disassembling in order to pass the duck bill or other mechanical loader as required.

Another development being actively investigated is high-pressure rotary rock drilling to replace hammer drilling. The pressures used are a ton for a small diameter bit and up to three for larger bits. Good results in drilling speed in sandstones of 7 to 8 hardness were reported. For example, one such sandstone percussive boring including carbide-type bits gave a drilling rate of about 3 to 4 in. per min while the high-pressure rotary drill in the same sandstone gave 24 to 40 in. per min. Another advantage claimed for rotary drilling is reduced silicosis because of less dust and more water flushing. Another result is less physical strain on the miners.

These developments in rock work, like the developments in mechanized coal cutting, are not widely in use. Under the Nazi system, where most mining was done by cheap forced labor, it was not essential to divert steel and other necessary material to the saving of labor. Since the armistice, however, the Ruhr coal miner is a free citizen with the highest possible priorities for food, consumer goods, and housing and with as much dignity as it has been possible to extend to him. However, because of the general living and industrial conditions in Germany, it has not yet been possible to find for the mines all of the steel and other materials and supplies necessary to give the miner something with which to increase his efficiency, nor has it been possible to see to it that he receives and consumes an adequate amount of food. The coal miners' ration is the highest in Germany, and is comparable to that of heavy labor elsewhere, but the miner naturally shares his ration with his family.

Another difficulty is the present lack of adequate patent protection for new developments. When this deficiency is corrected, we may see some genuinely new postwar modifications of these cutters and of the necessary hoists, conveyors, steel props, etc. to make their operation possible.

It will take time and materials to achieve anything like the potential mechanization of coal cutting and loading; but since the Ruhr coal mining industry and the United States Military Government believe that increased efficiency underground will mean increased coal output which will in turn speed the general economic recovery of Western Europe, every encouragement is being given to increase the number of mechanized longwall faces in the Ruhr as rapidly as possible under the circumstances.

#### Acknowledgment

The author wishes to acknowledge the able assistance given by R. W. Wayland, formerly Special Advisor to the Chairman (United States) of the Anglo-American Coal Control of Germany, in compiling the data for this paper.



# The Probability Theory of Wet Ball Milling and Its Application

by Elliott J. Roberts

**D**URING the course of a study of the fundamentals of classification in 1937, the need for a more basic understanding of the action of a ball mill became acute. Unless one knows how classification affects grinding, one cannot hope to effectively improve on classification.

The methods of evaluating grinding efficiency that depend on surface developed were studied but soon discarded for two reasons:

1. There was no apparent method which could be generally used to give a reliable figure for the actual new surface developed as a result of grinding. Subsequent papers have not changed this conclusion.

2. The practical evaluation of grinding in the main ore dressing applications was in terms of the percentage retained on a screen which passes 90 to 99 pct of the material and not in terms of surface area.

## The Probability Theory

With the background of our experience in the field of closed-circuit grinding, together with the papers of Lennox,<sup>1</sup> Gow,<sup>2</sup> Gaudin,<sup>3</sup> Fahrenwald,<sup>4</sup> Coghill,<sup>5</sup> and others, the approach of the theoretical physicist was then tried. The thought was somewhat as follows:

When one grinds in a ball mill, a given expenditure of power leads either to a certain number of point to point blows per hp-hr or to a certain distance of line contact per hp-hr, depending on whether the action of the balls is considered to be cascading or rolling.

It is also assumed that the balls actually come together on each blow or during the roll. Then a volume of slurry will be covered per minute which is some function of the size of the particle being considered (see fig. 1). All particles coarser than this size will be reduced through this size. This volume of slurry contains a certain weight of ore, depending on the percent solids and the density of the solids.

If we fix the percent solids and the density of the solids and let  $w$  be this certain weight of ore in the volume covered, then, in mathematical terms, what we have just postulated is,

$$\frac{w}{dt} \propto \text{hp} \quad (a)$$

If  $W$  is the total weight of ore present in the mill, then we can write,

$$\frac{W w}{W dt} \propto \text{hp} \quad (b)$$

The theory is developed that the tons ground through a given mesh per day in a wet ball mill is proportional to the percent plus that mesh in contact with the balls and the net power applied to the balls at this point. A grindability test is described.

and if  $C$  is the cumulative percent plus the size chosen at the start of the time interval  $dt$ ,

$$\frac{W w C}{dt W} \propto \text{hp} \times C \quad (c)$$

But  $\frac{wC}{100}$  is the weight plus the size chosen which at

the close of time  $dt$  is finer than that size, and  $\frac{wC}{W}$

is the decrease in the percent plus of the whole mass of ore or  $-dC$ . Then,

$$\frac{-W dC}{dt} \propto \text{hp} \times C. \quad (d)$$

In other words, the mesh tons ground through a given size per unit of time is proportional to the hp and the percent plus the mesh. A crude analogy would be to picture a 1-ft-wide steam roller going down the road at 1 ft per sec. If we place one egg on the road per square foot, one egg will be smashed per second. If we place a dozen eggs per square foot, a dozen eggs will be crushed per second.

Similarly, if all the particles in  $w$  are plus the mesh, i.e.,  $C=100$ , we should have a maximum rate of reduction. If only 10 pct of them are plus the mesh ( $C=10$ ), we would have only one tenth the maximum rate; if only 1 pct are plus the mesh, the balls have a hard time finding anything to work on. This is where the term "probability theory" comes from. The chances of the balls crushing a particle through a given mesh depends directly on the concentration of particles coarser than this mesh in the general pulp in the mill.

Giving  $W$  the units of tons and dividing equation (d) through by  $W$ , we obtain

$$\frac{-dC}{dt} = k \frac{\text{hp}}{\text{ton}} C \quad [1]$$

where  $k$  is a constant for any one size of particle, density of solid and moisture content of pulp.

Eq 1 is the rate equation for a first order reaction and says that the rate of decrease of the percent plus a given mesh with time is directly proportional to the hp per ton applied to the body of ore and to the percent plus the mesh in the ore mass as a whole. Since it is a differential equation, it only

E. J. ROBERTS, Member AIME, is Research Director, The Dorr Company, Westport, Conn.

AIME New York Meeting, February 1950.

TP 2969 B. Discussion (2 copies) may be sent to Transactions AIME before Jan. 31, 1951. Manuscript received Feb. 16, 1950.

applies to an instantaneous condition, but it can be readily integrated to give eq 2.

$$\log_{10} \frac{C_0}{C_t} = \frac{k}{2.3} \frac{\text{hp}}{\text{ton}} t \quad [2]$$

where  $C_t$  = cumulative percent plus size at time  $t$   
 $C_0$  = cumulative percent plus size at zero time.

The Lake Shore staff<sup>8</sup> noted that "almost a constant percentage of the +28 micron material remaining in the feed is ground out in each 3-min interval" and presented data to support the observation but did not pursue the subject further. Their note is another way of saying what is expressed above and eq 1 is its mathematical form.

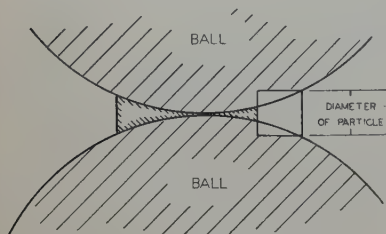


Fig. 1—Slurry volume covered by balls.

It is apparent from eq 2 that a semilogarithmic plot of  $C$  versus  $t$  should give a straight line if the equation holds. Fig. 2 shows a rather good agreement with the equation. Cumulative percent +150-mesh is plotted vertically on the log scale and time of grind is plotted horizontally. The mill was the laboratory mill described later and the charge was constant at 2 kg. Thus hp per ton is constant.

The data of fig. 2, line  $a$ , was obtained with a cement rock, ore  $B$ , by grinding charges of -20-mesh feed for 1, 2, 4, 8, and 16 min. A straight line through the first five points misses the 16-min point by 1.3 pct. A sample of the feed to  $a$  was classified at about 100-mesh removing 42.2 pct by weight of the sample. This brought the percent +150-mesh up to 86.5 pct. Grinds on this material as heads gave line  $b$ . The agreement of the individual points (circles) with the line is excellent.

If we let

$$\frac{k}{2.3} \times \frac{\text{hp}}{\text{ton}} = K \quad [3]$$

then from [2]

$$K = \frac{\log_{10} \frac{C_0}{C_t}}{t} \quad [4]$$

and it is apparent that

$$K = \frac{1}{t_n} \quad [5]$$

where  $t_n$  = time at which  $C_t = 0.1 C_0$ .

Applying this procedure, we find  $K$  for line  $a$  to be 0.055 and for line  $b$ , 0.052. This is in disagreement with eq 1, but by plotting the points of line  $a$  at  $t$  plus 3.8 min, we see that the discrepancy is not very great and the last point is practically on line  $b$ .

### Practical Implications

To show what this relationship means in practical terms, let us compare the grams ground through 150-mesh in the first 2 min in each case. With line  $a$ , there were 240 g of -150-mesh produced in 2 min, while with line  $b$  there were 370 g or a ratio of 1.54 to 1 in favor of the classified product. A more dramatic illustration is to compare the time

required to reduce 2 kg of feed to 5 pct +150-mesh. Curve  $a$  says 18.9 min. At 5 pct +150-mesh in the overflow 2 kg of feed  $a$  would give 990 g of +150-mesh to be reduced. At the 370 g per 2 min rate above determined by the upper part of line  $b$ , this would take 5.35 min or a ratio of 3.53 in power requirements for  $a$  vs  $b$ .

An actual closed-circuit test showed a falling off of  $K$  from 0.052 to 0.0472 before coming into balance, which would increase the time required to 5.9 min and cut the horsepower ratio to 3.2.

This is an unusually high ratio. It is generally more of the order of 2.0, and this lower ratio is due to a failure to obey eq 1. This is shown in fig. 3 wherein the results of two series of grinds on a gold ore (ore  $A$ ) are plotted. Only the 200, 100, 48, and 28-mesh points are shown to avoid confusion. It is seen that the 200-mesh points fall on a straight line satisfactorily, but 100-mesh points of the 16 and 17.12 min grinds drop well below the line. This drop occurs before 8 min in the case of the 48-mesh points, so that a grind to 1.5 pct +48 would take only 14.3 min instead of 24.3 min by the straight line. To grind closed circuit from 76.0 pct +48 to 1.5 pct + at 3 to 1 circulating load, we must reduce the percent +48 by  $74.5 \div 4 = 18.6$  pct in each pass through the mill. In practice this means going in at 62.6 pct + and coming out at 44.0 pct +. The corresponding times on our straight line are 1.15 and 3.35 min or a net of 2.2 min per pass. Multiplying by 4 for 4 passes we have 8.8 min to grind 2 kg of feed to 1.5 pct +48-mesh vs 14.3 min open circuits, or a power ratio of 1.63 in favor of closed circuit.

In an actual laboratory closed-circuit test,  $K$  dropped from the value 0.07 shown in fig. 2 to 0.0625, which would cut the power ratio to 1.46.

In both cases mentioned, the  $K$ 's at the mesh of separation had dropped about 10 pct when the closed circuit came into balance. This is by no means general. The  $K$  rose in another closed-circuit test on the gold ore at 1 to 1 circulating load ratio. This behavior lessened the beneficial effect of the higher circulating load. Other ores ground at 4 tons of mill discharge per ton of product ended up with a higher  $K$  than at the start and others were substantially constant. In all cases the comparisons were made at the same cumulative percent plus in the ball mill discharge, and eq 4 was used to evaluate the  $K$ 's. Fig. 4 shows the results on the secondary grind of a very hard gold ore, (ore  $C$ ), grinding -20-mesh feed with 2-in. balls. It is noted that 100 and 150-mesh results obey eq 2 very well, 65-mesh points curve downward as noted in fig. 3. The

Table I. Grinding Tests on Ore A  
Effect of Moisture

Grind Time	0	2	2	2	2
Solids, Pet Mesh	Cum. Pet +	75 Cum. Pet +	70 Cum. Pet +	60 Cum. Pet +	50 Cum. Pet +
6	8.8	0.6	1.6	1.5	1.4
8	26.4	2.8	5.4	5.9	6.2
10	39.3	6.9	10.1	11.0	11.0
14	50.1	13.6	16.8	18.3	19.5
20	59.0	23.1	25.8	28.0	30.4
28	65.8	33.0	35.3	38.2	41.1
35	71.4	43.1	45.2	47.8	51.0
48	76.0	52.2	53.4	56.5	59.4
65	79.8	59.7	61.5	63.8	66.4
100	82.8	67.2	67.8	69.9	72.0
150	85.5	71.8	73.1	75.0	76.8
200	87.4	76.3	77.4	79.0	80.7



200-mesh points lie on a straight line for the first 16 min, and then the slope of the line decreases slightly. The 325-mesh and 17.8 micron results give exaggerated forms of the 200-mesh line, the constants decreasing with time. On the same plot, denoted by circles, are the results of the fourth and fifth cycles of a closed-circuit grinding series at 4 to 1 circulating load ratio, making a 200-mesh separation. On the average, the points bracket the simple batch grind curves, showing no significant change in grinding behavior.

### Effect of Moisture

According to the original hypothesis, the moisture content of the pulp in the mill should affect the grinding rate. To be consistent with the semilogarithmic behavior just demonstrated, the rate should be proportional to the volume percent solids in the mill. Of course, above a certain figure ball action will cease and grinding will stop.

To investigate this effect, four grinds were made on ore A at 75, 70, 60, and 50 weight percent solids equalling 0.5265, 0.4635, 0.3570, and 0.270 volume fraction solids respectively. Table I presents the screen analyses of these grinds while table II presents the results in the form of ratios of equivalent times. An enlarged plot of fig. 3 was made and the times required at 70 pct solids to equal the results at the various percent solids read off. Dividing by the 70 pct solids equivalent time, table II resulted. The ratios of the various volume percent solids is noted as well as its 0.65 power which gives a reasonable agreement with the time ratios.

Thus eq 1 and 2 may be rewritten:

$$\frac{-dC}{dt} = k' \frac{hp}{ton} V^{0.65} C \quad [6]$$

$$\log_{10} \frac{C_0}{C_t} = \frac{k'}{2.3} \frac{hp}{ton} V^{0.65} t \quad [7]$$

where  $V$  = volume fraction of solids in the mill

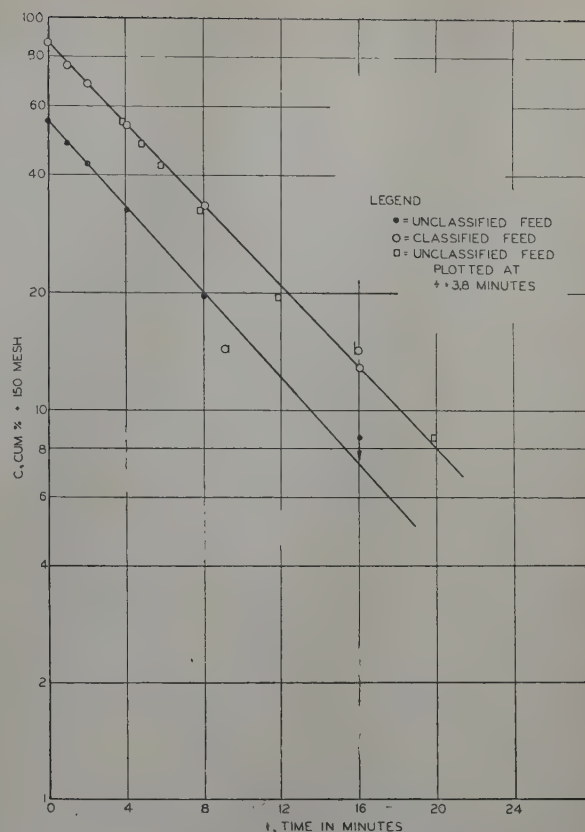
$$k' = \frac{k}{(0.468)^{0.65}} = 1.64k \quad [8]$$

A volume fraction solids of 0.468 was chosen for reference because it corresponds to 70 wt pct solids of 2.65 sp gr which was found safe for the first few grinds (1, 2, 4 min) and corresponds closely to mill practice.

The author is not inferring that the volume factor in the above equations will hold for all ores or all meshes any more than the log relationship itself holds rigorously. He merely desires to illustrate the magnitude of the effect in one case and show

**Table II. Effect of Pct Solids in Mill, Equivalent Time at 70 Pct Solids to Get Results at Other Pct Solids**

Wt. Pct Solids	75	70	60	50
Mesh				
35	1.09 min	1.00 min	0.88 min	0.76 min
48	1.11	1.00	0.86	0.70
65	1.14	1.00	0.89	0.73
100	1.14	1.00	0.84	0.70
150	1.14	1.00	0.83	0.70
200	1.12	1.00	0.83	0.65
Avg	1.12	1.00	0.85	0.71
Ratio of volume fraction solids	1.135	1.00	0.772	0.583
(Vol fraction solids ratio) <sup>0.65</sup>	1.086	1.00	0.845	0.704



**Fig. 2—Grinding tests, ore B, —20-mesh.**

that the simple picture drawn at the start is incomplete and that the situation is really much more complicated.

In fact, if our original assumptions held rigorously, not only would the exponent of  $V$  in eq 7 be unity but  $k'$  would be the same for all ores and all ores would have the same intrinsic grindability (after allowance for density).

We know that this is not so and all of our deviations are probably due to the balls, not coming together each blow or during the roll and to the fact that different materials do have different shattering characteristics.

In other words, it is perhaps fortuitous that the agreement with eq 2 is as good as it is and therefore, all the equations are, in a sense, empirical, just as with 6 and 7.

### Horsepower per Ton Factor

In view of Coghill's weight of evidence that the rate of grinding is proportional to the horsepower applied per ton, it seemed hardly necessary to go into this variable from the laboratory standpoint. However, sometimes the samples available for study were small, and it would help a great deal if the same sample could be used for the various grinds. Accordingly, a 2 kg sample was ground 2 min and 200 g removed for screen analyses; the 1800 g remaining were ground 1.8 min more to give the equivalent of a 4-min grind. This procedure was continued for 8, 16, and 32 min. Thus the last part of the 32-min grind was done on only 1200 g. To check this 32-min grind, another 2-kg sample was ground directly for 32 min. Table III shows that the results are essentially identical.

However, other tests have shown that on the coarse meshes discrepancies can occur. Table IV

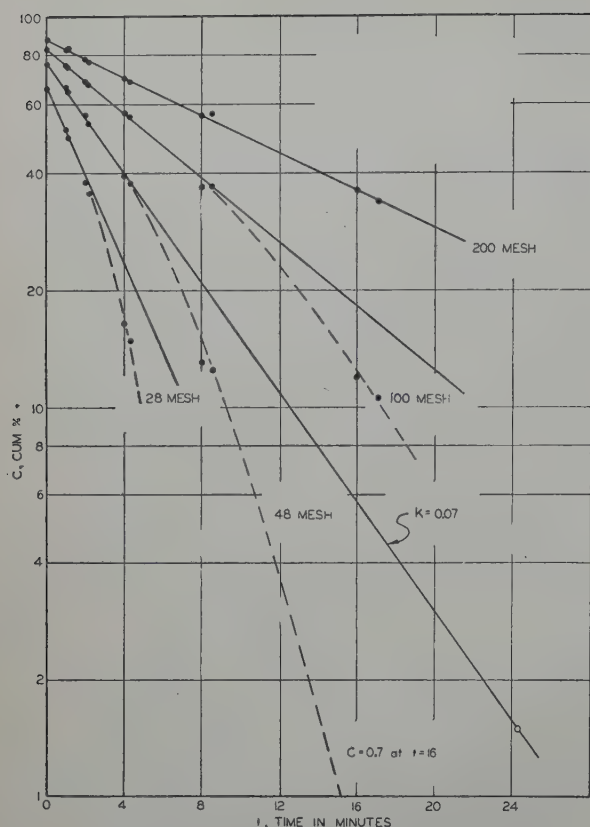


Fig. 3—Grinding tests, ore A, —4-mesh.

presents two pairs of grinds, which show that although at the fine meshes the results are equivalent whether 2 kg is ground for  $x$  min or 4 kg is ground for  $2x$  min, on the coarse meshes the larger sample ground the longer time is slightly finer.

#### The Laboratory Mill

The other aspect of the horsepower per ton factor was whether the results in a small mill could be translated directly into behavior in mills of 6, 8, and 10-ft diam. The solution of this problem was fraught with innumerable difficulties. Accurate

Table III. Screen Analyses Cumulative Pct Plus 32 min Equivalent Grinds

MESH	By Stages Ending with 1200 g	2-kg Charge Direct
65	0.4	0.5
100	3.1	3.8
150	10.5	11.6
200	20.8	22.8
325	36.7	37.3

Table IV. Screen Analyses Cumulative Pct Plus Comparing Effect of Size of Sample

MESH	—4-Mesh Ore C, 2 in. Balls		—3-Mesh Soft Copper Ore 1½-in. Balls	
	2 kg for 8 min	4 kg for 15.5 min	2 kg for 2 min	4 kg for 4 min
20	1.3	0.1	0.8	0.8
28	3.9	1.9	2.0	0.9
35	11.5	9.4	5.7	4.9
48	24.3	23.5	14.2	11.3
65	38.2	37.5	25.9	23.8
100	49.0	49.9	37.5	36.4
150	57.7	58.3	49.8	49.0
200	64.4	65.0	59.0	58.8

plant data had to be secured together with a representative sample of the feed. One of the biggest stumbling blocks encountered was the determination of the horsepower being drawn by the plant mill. Also for a direct comparison, the ball size in the two mills should be the same. This was made difficult to satisfy exactly because to keep the physical labor involved in making a test within bounds, it was decided to construct the mill from a section of 16-in. pipe measuring 15.4 in. ID by 6 in. wide.

It was felt that the ball action would be better with three rows of 2-in. balls than with larger balls. Furthermore, a charge of uniform balls would be much simpler to handle than a graded load. In addition, if the 2-in. uniform balls gave results equivalent to a 2½-in. graded load, a large number of plant feeds could be handled directly. Therefore it was decided to use a charge of 24.8 kg of 2-in. diam balls for comparative grinds. When rotated at 54.5 rpm, the loaded mill drew 0.150 hp, net, as defined by Gow.<sup>2</sup> Following Gow,<sup>2</sup> that, on the average:

Input hp to motor = gross hp =  $1.33 \times$  net hp [9]  
the above set up was rated at 0.200 hp gross.

Thus a 1-min grind on a 2-kg sample is equivalent to:

$$\frac{2000}{4.4} \times \frac{0.200}{60} = 1.515 \text{ hp-hr per ton [10]}$$

It was found that this uniform 2-in. ball would handle any —4-mesh feed put to it; i.e., the 6, 8, and 10-mesh particles were ground at such a rate that they would not build up unduly in a closed circuit operation.

In this respect the 2-in. load appears to be equivalent to at least a 2.5-in. graded load of forged balls.

On soft ores, feeds as coarse as —¾ in. appear to grind satisfactorily, but, for comparative purposes, a —4-mesh feed with 10 pct +6-mesh was accepted as standard.

As indicated earlier, the standard charge was 2 kg at 46.8 pct solids by volume for coarse grinds, the moisture being increased as the charge thickened due to grinding.

#### Closed-Circuit Tests

To isolate the effect of mill size on the horsepower per ton factor, as distinct from the other variables, closed-circuit grinds should be made exactly duplicating the grind, circulating load, etc., observed in practice.

This can be done, and, in some of the work to be reported, has been closely approximated. However, it is an exacting and time-consuming procedure to get into complete balance at the plant circulating load and unless the feed size used is exactly the same, it is impossible to duplicate all of the conditions required.

The tests reported were really designed for another purpose, which required that the tons of mill discharge per ton of new feed be in the neighborhood of 4.0. Not all of the plant tests were run at this ratio. Some of the feeds were as coarse as —1 in. while the small mill necessitated a —4-mesh feed. In the laboratory tests also, if the new feed was finer than the classifier rake product, it was put into the classifier, which was not always done in the plant. All of these variations required some adjustment, which was done by the aid of the principles previously established, but both the unadjusted



and adjusted comparisons are given so that the magnitude of the correction can be seen.

The closed-circuit tests were carried out substantially as follows: A 2-kg charge of —4-mesh ore was ground the number of minutes estimated to give a circulating load of 3 to 1. It was then classified at the proper moisture content in a 3-ft-deep tube at a draw-off rate estimated to give the proper mesh of separation. After a few trials, this could be done quite accurately. The oversize was returned to the mill with new feed equivalent to that withdrawn in the overflow and the process repeated, adjusting the grinding time to get 500-g overflow. This was continued until the grams per minute reduced through the mesh of separation was substantially constant.

It was not always possible to obtain test data which coincided with the taking of the head sample. In most cases, this was not serious because of the uniformity of the ores but could well have affected the ore *E* data.

### Comparative Results

Table V presents the comparative data. Essential data on the plant circuit is first given, ending with the hp-hr per ton actually used. Then for comparison is the laboratory tons ball mill discharge per ton of feed and the hp-hr per ton indicated by the laboratory data. Since many of the plant feeds were coarser than the laboratory feed, the laboratory horsepower figure is not the directly-observed result but was calculated as follows:

$$\text{hp-hr per ton} = \frac{(NF - O)}{100} \times \frac{2000}{R} \times 1.515 \quad [11]$$

where *NF* = Pct plus mesh of separation in plant feed

*O* = Pct plus mesh of separation in plant overflow

*R* = g ground through mesh of separation per minute of grind in the laboratory closed-circuit test

It is seen that the first part of the expression determines the number of minutes which would be required to reduce 2 kg of plant feed to the plant mesh of separation at the 0.20-hp rate, and 1.515 is the hp-hr per ton per minute of grind on 2 kg previously noted (eq 10).

Table V. Comparison of Laboratory Closed-Circuit Results with Plant Results

Plant	Plant Data				Laboratory			
	Grind Pct + Mesh	Make-up Ball Size in.	Tons BMD Ton Feed	Hp-hr Ton Feed	Tons BMD Ton Feed	Hp-hr Ton Feed	Hp-hr Ratio Plant Lab.	Hp-hr Ratio Adjusted for <i>C<sub>avg</sub></i>
A	2.6 pct + 48	2½	5.0	14.0	4.0	13.4	1.04	1.04
A	1.3 pct + 48	2½	4.5	13.8	4.0	13.9	0.99	0.99
D	6.2 pct + 65	2	1.8	10.2	4.3	9.2	1.11	0.90
E	1.4 pct + 65	2	4.8	6.8	4.0	6.4	1.06	0.97
F	1.3 pct + 35	3	4.5	4.4	5.3	3.3	1.33	0.97
F	3.7 pct + 35	3	5.0	4.0	5.3	3.4	1.18	1.06
G	3.0 pct + 48	4	3.7	12.6	3.3	11.0	1.15	1.20
C <sub>1</sub> <sup>a</sup>	20 pct + 35	3½	3.3	12.0	4.0	10.2	1.18	1.18
C <sub>2</sub> <sup>a</sup>	1.4 pct + 200	1¼	1.4	22.9	4.0	21.2 <sup>b</sup>	1.08	0.60
B <sub>1</sub>	1.6 pct + 14	4	2.9	7.2	4.4	7.0	1.03	1.11
B <sub>1</sub>	0.6 pct + 14	4	2.0	8.3	4.4	7.1	1.17	1.11
B <sub>2</sub>	13 pct + 200	2	3.5	7.6	3.8	9.7 <sup>b</sup>	0.78	0.75
B <sub>2</sub>	10 pct + 200	2	2.1	8.2	3.8	9.9 <sup>b</sup>	0.83	0.77
A	3.4 pct + 48	4	6.4	14.8	4.0	13.5	1.09	1.09
A	3.7 pct + 48	4	5.5	14.6	4.0	13.5	1.08	1.08

<sup>a</sup> Subscript 1, primary; subscript 2, secondary.  
<sup>b</sup> Grinds made on —20-mesh feed.

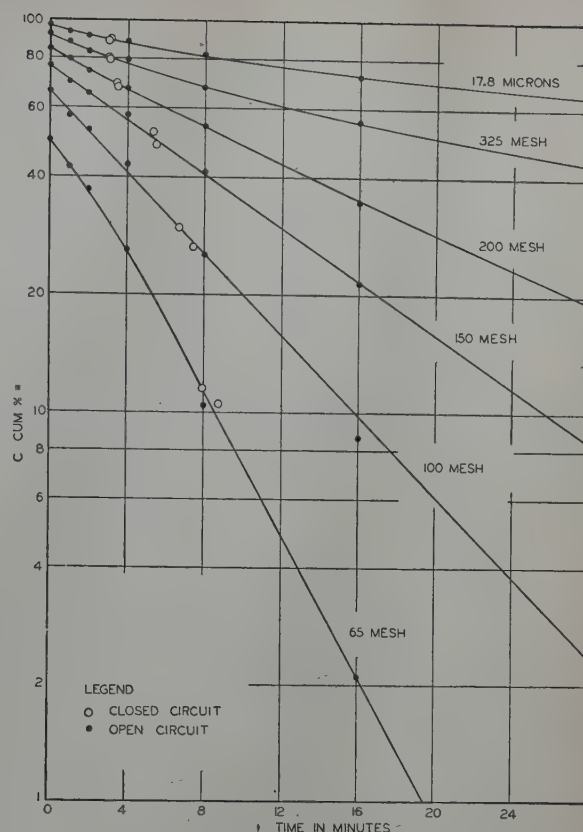


Fig. 4—Grinding tests, ore C, —20-mesh.

The hp-hr resulting from this calculation is thus the energy required in the laboratory mill to grind feed from the plant *NF* to the plant overflow fineness at the laboratory circulating load. Other qualifications were that the ball size was 2 in. uniform, that the feed was —4-mesh (or —20-mesh in some cases), and that the laboratory power figure contained a 25 pct allowance for motor, bearing, and windage losses.

Therefore, if plant feed, overflow, and circulating load conditions were matched perfectly and if the plant motor, bearing and windage losses were 25 pct; then, if the ratio between plant and laboratory hp-hr equals 1.00 for a plant make-up ball size of around 2½ in., the horsepower per ton factor in eqs 1 and 2 may be considered to hold independent of mill size.

Table VI. Summary of Adjusted Hp Ratios

Ball Size in Plant	Plant	Ad-justed Hp Ratio	Feed Size		Rel. Grindability	
			Plant	Laboratory	48-Mesh	200-Mesh
4 in.	G	1.20	— 1 in.	— 4-mesh	1.64	0.89
4 in.	B	1.11	— 1 in.	— 4-mesh	1.41	1.74
4 in.	A	1.09	— 4-mesh	— 4-mesh	1.40	1.10
3½ in.	C	1.18	— 2-mesh	— 4-mesh	1.00	1.00
3 in.	F	1.02	— 3-mesh	— 4-mesh	3.60	2.75
2½ in.	A	1.02	— 4-mesh	— 4-mesh	1.40	1.10
2 in.	E	0.97	— 8-mesh	— 4-mesh	2.38	1.86
2 in.	D	0.90	— 4-mesh	— 4-mesh	2.07	1.88
2 in.	B	0.76	— 20-mesh	— 20-mesh	1.41	1.74
1¼ in.	C	0.60	— 20-mesh	— 20-mesh	1.00	1.00

Table VII. Relative Grindability Standard K's and Percent Plus Ball Mill Discharge

Mesh	K	Percent Plus Ball Mill Discharge at Which K Was Used
14	0.0961	41.1
20	0.0818	41.7
28	0.0680	42.4
35	0.0575	43.4
48	0.0479	44.0
65	0.0398	45.7
100	0.0326	47.9
150	0.0264	51.0
200	0.0217	54.1
270	0.0174	57.5
325	0.0157	59.5
400	0.0142	61.9

Two tests on ore from plant A were the only ones where the ball size and other conditions were right, and here we find an average horsepower ratio of 1.015. The other two plant A tests with a 4-in. make-up ball gave an average ratio of 1.085, which is in line with the findings of other investigators on the effect of ball size.

In general, the other data of table V need some adjustment for circulating load. This was done by the use of the part of eq 1 which says that the rate of reduction at any moment is proportional to C. To use this relationship, we need an effective average value of C,  $C_{avg}$ , in the mill.

This  $C_{avg}$  is the logarithmic mean value of C,

$$C_{avg} = \frac{C_o - C_i}{2.3 \log_{10} (C_o/C_i)} \quad [12]$$

zero time being considered when the feed enters the mill and time  $t$  when it leaves.

This logarithmic average is lower than the arithmetic average by an amount depending on the difference between  $C_o$  and  $C_i$  and approaches the arithmetic average only at high circulating loads.

The laboratory hp-hr per ton figures are corrected to the plant  $C_{avg}$  by taking

$$\frac{\text{Lab } C_{avg}}{\text{Plant } C_{avg}} \times \text{col. 7} \quad [13]$$

Dividing this into col. 5 gives us col. 9 which is the same as:

$$\text{col. 9} = \text{col. 8} \frac{\text{Plant } C_{avg}}{\text{Lab } C_{avg}} \quad [14]$$

Actually, in some cases a further adjustment was made for the change in K with percent plus in the ball mill discharge; i.e., for the deviation from the simple semilogarithmic relationship.

These adjusted ratios or conversion factors are placed in order of descending ball size in table VI

together with data on the size of the plant and laboratory feeds.

It is obvious from the way the adjusted ratios follow the plant ball size that the hp per ton factor in eq 1 probably does hold and that these ratios are really ball-size factors. The data are neither sufficiently extensive nor accurate to permit of any generalizations on the relationship between ball size and factor at this time, however.

### Grindability

If the K's for the different meshes (as determined directly by eq 4, or by plotting and eq 5) are compared, it is apparent that a "grindability" series will result. To be strictly accurate, the K's should be those resulting from closed-circuit tests but that would require a closed-circuit test at every mesh, which would be excessively time consuming. Therefore the K's were determined from simple batch grinds made on —4-mesh (10 pct +6-mesh) ore at 46.8 pct solids by volume. The K's for the different grinds were then plotted against percent plus the mesh in the ball mill discharge and the value of K at a definite percent plus in the ball mill discharge read off from the plot. These K's were divided by the corresponding K's determined on ore C and the result called the "relative grindability." Table VII gives these basic K values together with the standard ball mill discharge percentages at which they were read off. These ball mill discharge figures correspond closely to the ball mill discharge resulting from a 3 to 1 circulating load on —4-mesh feed on an average homogeneous ore.

Table VI carries the "relative grindabilities" of the ores involved at 48 and 200-mesh. We have found only one ore which gave a value of over 4 at 48-mesh and none which gave over 4 at 200-mesh.

It has been demonstrated that in a wet ball mill-ing operation the rate of reduction through any mesh is roughly proportional to the percent plus that mesh.

It has also been demonstrated that the rate of reduction is substantially proportional to the hp per ton applied to the balls and independent of the size of the mill.

A factor has been presented for the effect of percent solids in the mill in one case and data showing the effect of ball and feed size are given.

A "relative grindability" procedure based on the above principles has been outlined.

### Acknowledgments

The writer wishes to acknowledge gratefully the help and cooperation of his professional associates both at Westport and in the other offices; and to express his thanks to the various plant superintendents and their staffs for sending samples to Westport.

### References

- W. H. Coghill, F. D. DeVaney and R. G. O'Meara: Advantages of Ball Mills of Larger Diameters. *Transactions AIME* (1934) **112**, 79.
- A. W. Fahrenwald: Some Fine Grinding Fundamentals. *Transactions AIME* (1934) **112**, 88.
- A. M. Gaudin: An Investigation of Crushing Phenomena. *Transactions AIME* (1926) **73**, 253.
- A. M. Gow, M. Guggenheim, A. B. Campbell, and W. H. Campbell: Ball Milling. *Transactions AIME* (1934) **112**, 24.
- Lake Shore Staff: Fine Grinding Investigations at Lake Shore Mines. C.I.M.M. (1940) **43**, 299.
- L. W. Lennox: Grinding Resistance of Various Ores. *Transactions AIME* (1919) **61**, 237.



## Effects of Rod Mill Feed Size Reduction

by John J. Strohl and Henry J. Schwellenbach

This paper relates the results obtained by decreasing the size of feed to a rod mill screen circuit. As would be expected, added production was gained and a finer grind with no tonnage loss was made possible.

THE MacIntyre mill of the National Lead Co. has four parallel sections in which magnetic and gravity separations are made. Ore from a 5000-ton bin is fed by Hardinge constant weight feeders to a 6x12-ft open-end Marcy rod mill in each section. Rod mill discharge is bucket elevated to two 4x10-ft Tyrock screens whose oversize is returned to the rod mill. Screen undersize is pumped to Crockett separators for magnetite removal, and the Crockett tailings are sized and tabled to make an ilmenite concentrate. Minus 200-mesh material, overflowed by the Crockett and Dorrcro sizers is thickened, deslimed, and concentrated by flotation.<sup>1</sup>

By 1948, the development of a successful slime flotation process and numerous improvements in the gravity circuit of the mill had increased the capacity of the available concentrating equipment. At the same time, improvements in the operating techniques of the dry crushing plant and improved feeding of the crusher by the mining department had appreciably increased the production rate per operating hour in that plant. Therefore, it was decided to take advantage of the added crusher capacity to produce

a finer rod mill feed and in that way increase grinding capacity. At approximately the same time, finer screens were installed in the rod mill circuits to give finer grind and thus improve liberation of the magnetite and ilmenite from the gangue and from one another.

Prior to the first part of 1948, a ¾-in. limiting screen was used for sizing rod mill feed. The first reduction in crushing plant screen size was to ½ in. With this screen, the circulating load in the tertiary crushing circuit was too high and plant capacity dropped below a practical hourly rate. A 9/16-in.-sq opening screen, installed next, proved satisfac-

---

J. J. STROHL is Mill Superintendent, National Lead Co., Tahawus, N. Y., and H. J. SCHWELLENBACH, Member AIME, formerly Assistant Mill Superintendent, National Lead Co., is now Metallurgical Engineer, Warren Foundry and Pipe Corp., Dover, N. J.

AIME Columbus Meeting, September 1949.  
TP 2963 B. Discussion (2 copies) may be sent to Transactions AIME before Jan. 31, 1951. Manuscript received Sept. 9, 1949.

---

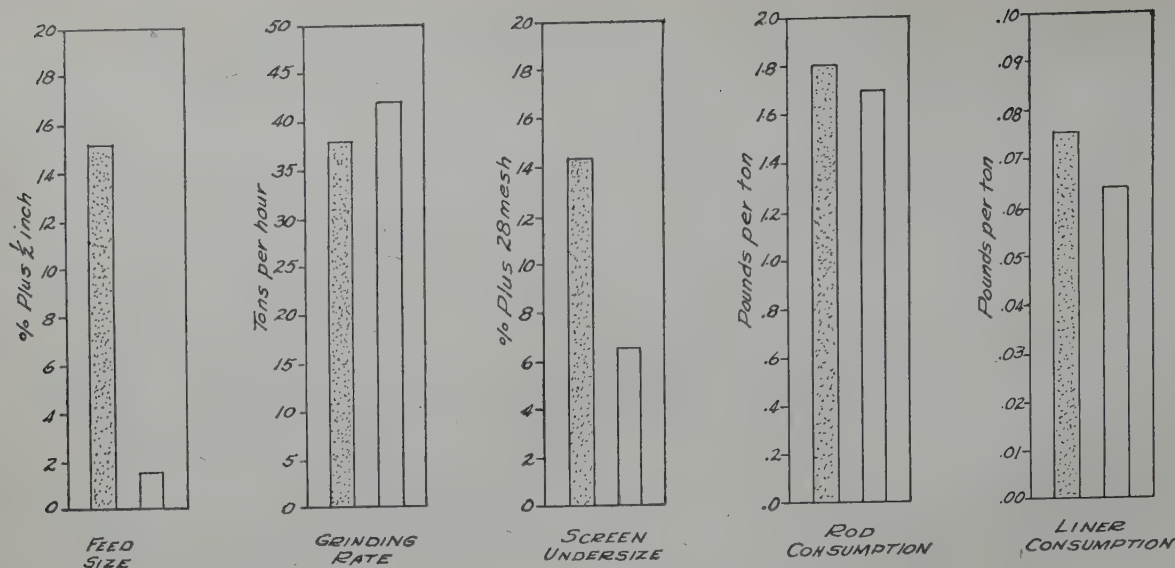


Fig. 1—Effect of rod mill size.

Dark bar is 3/4-in. limiting screen. Light bar is 9/16-in. limiting screen.

Table I. Comparative Screen Analysis of Rod Mill Feed  
Screen Analysis of Products, Weight, Pct

Limiting Screen Dry Crushing	+1/2 in.	+3	+4	+14	—14
—3/4 in.	15.2	28.8	9.9	18.9	17.2
—9/16 in.	1.7	39.5	13.0	22.3	23.5

Table II. Comparative Screen Analysis of Screen Undersize

Limiting Screen Dry Crushing	Rod Mill Screen	Screen Analysis of Products, Weight, Pct							
		+28	+35	+48	+65	+100	+150	+200	—200
3/4 in. feed	0.49 in.	14.04	13.81	15.33	13.01	11.55	8.67	5.76	17.83
9/16 in. feed	0.29 in.	6.45	12.23	15.61	14.69	12.51	11.14	7.67	19.70

tory in dry weather but blinded badly with wet ore. Production rates are maintained by using a slotted screen in the winter and spring months.

After reducing mill feed size, the hourly grinding rate per mill averaged over a number of months before and following the size reduction, increased from 38.4 tons to 42.4 tons. A 230-hp input to each rod mill remained approximately constant over these months.

With a 3/4-in. rod mill feed and 2 1/2-in. rods, rod wear was extreme at the feed end of the mills and excessive shell liner wear was evident at the feed and discharge ends of the mills. In an effort to improve feed entry into the rod mass and decrease scouring wear of the rods and liners at the feed end of the mill, a shipment of 3 1/2-in. rods was ordered. Their efficacy was not determined since the reduction of feed size was accomplished before these rods arrived. With a finer feed, the inordinate wear of the rods and shell liners at the feed ends of the mills decreased and liner life was greatly improved. Some 306,000 tons were ground by the only lining replaced since the feed change. The maximum ton-

nage ground by a set of liners prior to this time was 260,000. Rod consumption was lowered slightly.

Tables I and II give screen analyses of the two feeds. Unfortunately, more complete screen analyses of the two feeds are not available and it is impossible to evaluate the increase in surface of the finer feed as compared to 3/4-in. rod mill feed. Some 11 pct<sup>2</sup> more surface in the screened rod mill product

was produced after feed size and rod mill screen aperture were reduced.

The results of the two cases of rod mill feed size, as noted at MacIntyre, show that a finer rod mill feed will give a correspondingly greater grinding rate while grinding to a finer size, for better liberation, with a decrease in the rod and liner wear.

Fig. 1 shows graphically the variables in the two cases.

#### Acknowledgment

Appreciation is expressed to the entire staff and management of MacIntyre whose aid and participation made plant improvements possible, and to Frank R. Milliken, former Plant Manager, whose direction and advice has been invaluable to the advancement of the property.

#### References

- <sup>1</sup> F. R. Milliken: Metallurgy at National Lead Company, MacIntyre Development. *Transactions AIME* (1949) **183**, 101; *Mining Technology* (May 1948) TP 2355B.
- <sup>2</sup> The W. S. Tyler Company, Cleveland, Ohio: Catalog 53, p. 36.



# Laboratory Studies on Iron Ore Sintering and Testing

by F. M. Hamilton and H. F. Ameen

Results of a laboratory investigation of some sintering variables are presented. The effect on physical properties of mix component variation and rate of cooling of the sinter is illustrated by crush and tumbler test results and data of reducibility tests and microscopic examination of sinters are shown. The results of using solid additions in the mix to improve bed permeability are given.

SOME of the variables encountered in the sintering of iron ores and their effect on sinter properties have been investigated at the Jones and Laughlin Ore Research Laboratory in Michigan. Ores requiring agglomeration before blast furnace utilization may be classed as coarse, intermediate or fine. Coarse iron ore is considered similar to a  $-\frac{1}{4}$ -in. Mesabi wash ore containing some 95 pct of its particles between  $\frac{1}{4}$  in. and 100-mesh. Intermediate iron ore is considered similar to our New York State magnetic concentrates in which all the particles are  $-20$ -mesh and about 60 pct are  $+100$ -mesh. Fine ores are considered as being at least 100 pct  $-100$ -mesh. This paper will deal only with the two coarser classifications.

Most of the work has been done in a small experimental batch unit previously described in an AIME<sup>1</sup> paper. Results being obtained form a basis for present and future trials in production sinter plants. It is expected that new production units that may be added in the future will benefit from laboratory studies. In general, the results reported here are obtained by laboratory study, but it is

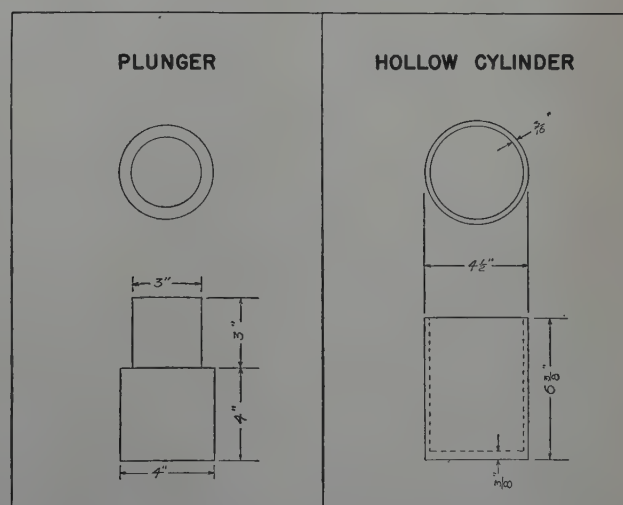


Fig. 1—Hollow cylinder and plunger, sinter crush test apparatus.

F. M. HAMILTON, Member AIME, and H. F. AMEEN, Junior Member AIME, are Research Engineers, Jones and Laughlin Steel Corp., Negaunee, Mich. AIME Salt Lake City Meeting, September 1950.

TP 2984 B. Discussion (2 copies) may be sent to Transactions AIME before Jan. 31, 1951. Manuscript received March 30, 1950.

hoped they will be helpful in future plant practices and design of new plants.

## Laboratory Equipment and Use

**Experimental Sinter Machine:** The laboratory sinter machine employed in the work consists of a chamber with 1 sq ft of grate area. It is mounted on

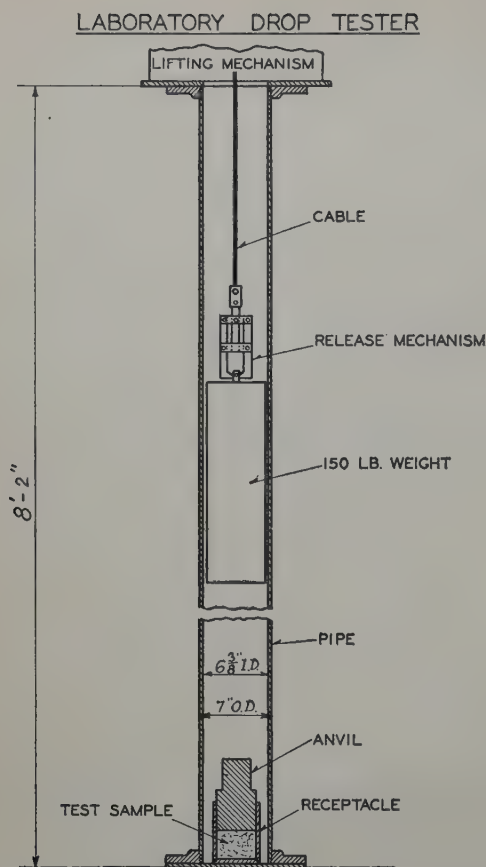


Fig. 2—Schematic diagram, sinter crush test apparatus.

trunnions and is rotatable to permit dumping. The chamber is connected through a 4-in. exhaust line to a suction fan. An orifice in the exhaust line and suitable taps to a U-tube permit gas flow measurements. A thermocouple in a well in the outlet of the sintering chamber is connected to a strip chart recorder and yields a continuous record of exhaust gas temperatures. The peak temperature is utilized as a reference point to measure the sintering time.

**Physical Test Apparatus and Typical Results:** Two methods are used for testing the physical properties of sinter. They are the result of an effort to find suitable methods of measuring these important properties. Many possible avenues were explored, including standard coke tests, impact crush tests, and pressure tests of various kinds. The methods found best for our work are crush and tumbler tests.

In the crush test, sinter to be tested is first sized by hand between 1 in. and  $\frac{1}{4}$  in. It is then placed in a hollow cylinder (fig. 1) and a plunger or anvil inserted. Next, a weight of 150 lb is dropped on the plunger from a height of 53½ in. (fig. 2). Since the results obtained are only comparative between samples, the distance of drop is not too significant. For this reason, a mechanism already on hand was used for lifting and dropping the weight and 53½ in. was one of the increments readily obtained. After crushing, the material is removed and screen analysed. Distribution of the particle size is plotted (fig. 3) and serves as a comparison between different kinds of sinter.

Fig. 3 illustrates the results of crush tests on soft and hard laboratory sinters, along with those of a highly oxidized, friable magnetite sinter obtained

from Sweden. Curve A represents the results on hard sinter, curve B the results on soft sinter, and curve C the results on the Swedish sinter. The figure shows appreciable differences between the curves of the different sinters. This test is a valuable tool in the study and examination of sinters.

The tumbler test is a modified version of the standard ASTM coke tumbler test. The apparatus used is identical to that of the coke test. However, a weight of 500 g of sinter sized between 1 in. and  $\frac{1}{4}$  in. is used. The sized material for testing is placed in the tumbler apparatus and the drum rotated 500 revolutions at 24 rpm. Upon removal, the sinter is screen analysed and plotted as shown in fig. 4. Here again, curve A represents the results on hard sinter, curve B on the soft sinter, and curve C on friable Swedish sinter.

These testing machines have also been used to test pellets of New York State magnetite concentrates. One batch was designated hard, the other soft. Crush and tumbler tests were made to determine whether appreciable differences in results could be obtained. Fig. 5 shows considerable difference in crush properties between hard and soft pellets, and fig. 6 shows the same trend in tumbler results. Both show larger amounts of -60-mesh material (after testing) than the most friable sinters we have tested.

#### Fuel Content of Mix

The amount of carbonaceous fuel required to sinter an ore to a specified physical condition (as indicated by these tests) is less for Mesabi hematites than New York magnetites. The difference for hard sinters is approximately 1 pct. A sinter such as that of curve A in figs. 3 and 4 can be made from New York magnetites with about 6 pct carbon (exclusive of returns), whereas 5 pct carbon is sufficient when using a Mesabi hematite. For any particular mix, fuel content is a very important factor in determining the physical and mineralogical characteristics of the sinter produced. These charac-

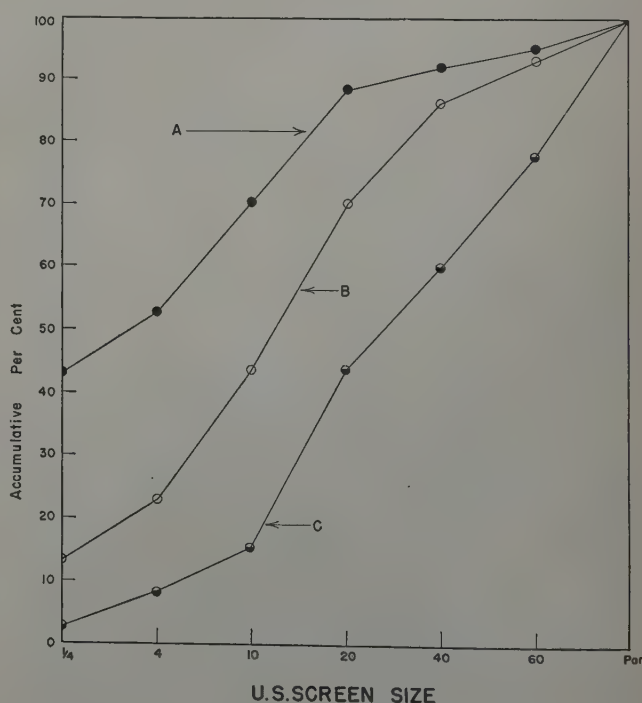


Fig. 3—Crush test results, sinter.



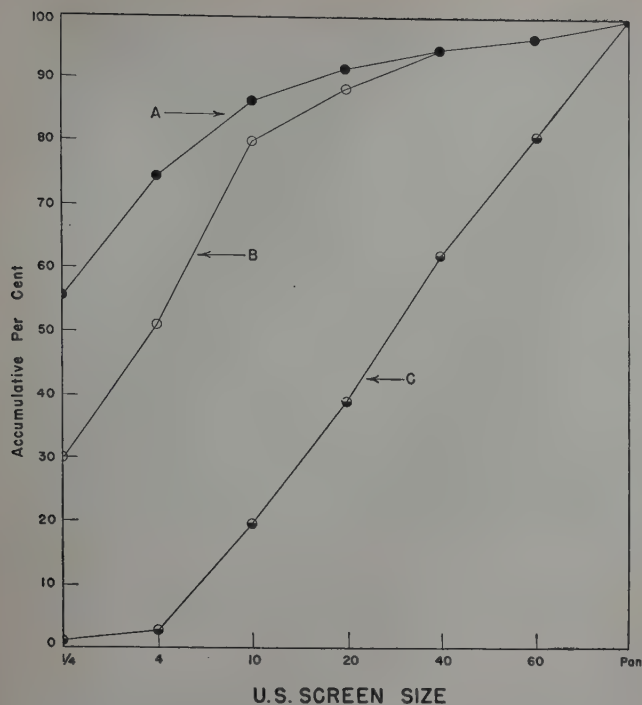


Fig. 4—Tumbler test results, sinter.

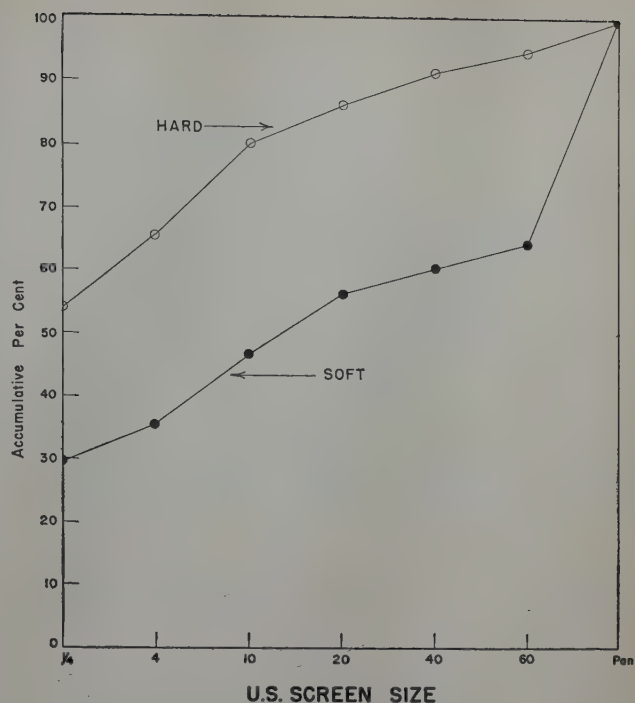


Fig. 5—Crush test results, pellets.

teristics have been and are now the subject of both physical and microscopic investigation.

**Microscopic Examinations:** In the microscopic work, the sinter samples are generally prepared as polished specimens and examined under reflected light. From this work it has been possible to determine the relationship of the different microconstituents, but as yet it has not been possible to determine the exact nature of the various silicates that are formed.

Examination of the different samples revealed that the material is composed of three major groups:

1. Unaltered fragments of the original ore minerals consisting of hematite, magnetite, and gangue.
2. Slag or binder material composed of artificial fayalite and amorphous glass.
3. Secondary iron oxides consisting of hematite and magnetite that have been dissolved and reprecipitated from the slag.

A polished section of hard magnetite sinter shows numerous medium sized equiaxed grains of magnetite surrounded by ribbons of silicate binder (fig. 7). The binder appears to be very similar to that observed in softer magnetite sinter. A few massive areas of silicate binder containing reprecipitated magnetite can also be observed.

Fig. 8 shows a soft magnetite sinter containing large grains of magnetite between which are dispersed small grains and ribbons of silicate binder. A few massive areas of silicate binders are present within which are stringers of dark-gray silicate. Numerous needles of hematite have been observed within the grains of magnetite.

A photomicrograph of a hard hematite sinter reveals small equiaxed grains of magnetite surrounded by slag (fig. 9). Note the very uniform distribution of magnetite within the slag.

A sample of soft hematite sinter shows it to be composed of fine grains of magnetite surrounded by thin ribbons of slag (fig. 10). Numerous massive areas of a eutectic-type structure may be seen.

Fig. 11 is a photomicrograph of a very soft and friable sinter made from Swedish magnetic concentrates. This material has hematite as the major microconstituent with some magnetite also present. The slag is noncrystalline and does not appear to coat the grains of iron oxide as in the other sinters. This is the sinter shown on curve C of figs. 3 and 4.

A number of facts are evident concerning the relationship of the microstructure of sinters and their physical properties. A significant factor is that con-

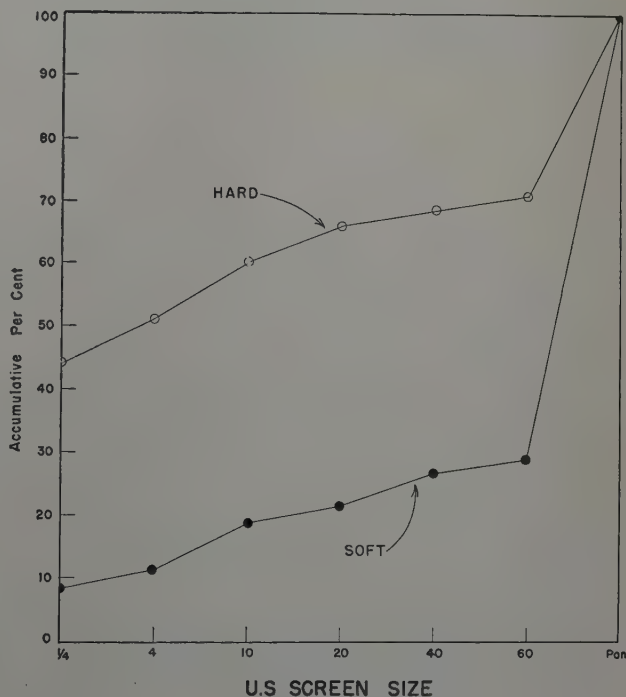
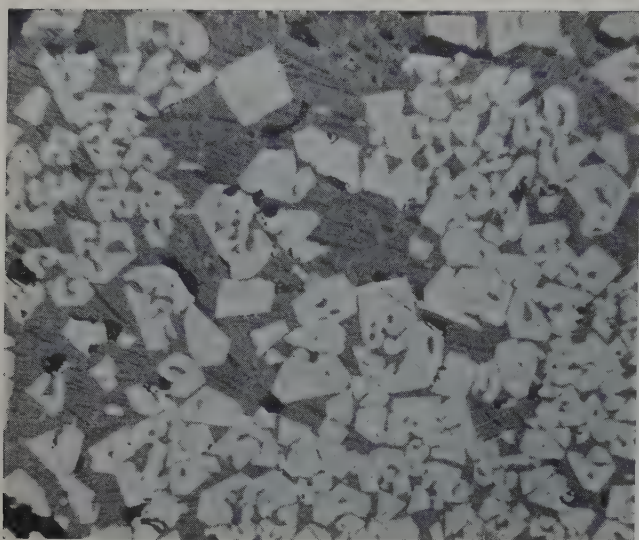
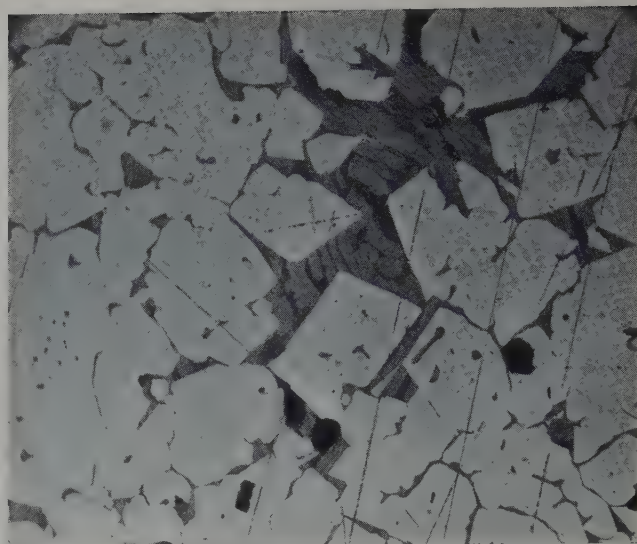


Fig. 6—Tumbler test results, pellets.



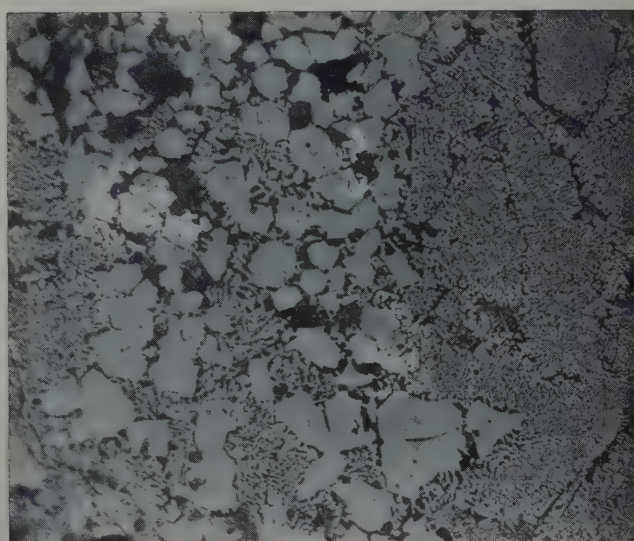
**Fig. 7—Hard magnetite sinter. Light gray grains are magnetite, medium gray areas are silicate binders. (Air cooled.) X100.**



**Fig. 8—Soft magnetite sinter. Light gray areas are magnetite, medium gray areas are silicate binder. Note dark stringer within grains of silicate binder. (Air cooled.) X100.**



**Fig. 9—Hard hematite sinter. White areas are magnetite, light gray areas are slag. (Air cooled.) X200.**



**Fig. 10—Soft hematite sinter. White areas are hematite, light gray areas are magnetite, dark gray areas are slag. (Air cooled.) X325.**

siderable solution and reprecipitation of the iron oxides takes place. It appears that upon burning the sinter mix, a considerable quantity of iron oxides are taken into solution by the molten silicates. Upon cooling to the melting point of magnetite, nuclei of magnetite begin to precipitate out as primary axes. As the process proceeds, with continued cooling, the spaces between the axes are filled, thus forming small grains of magnetite surrounded by molten slag. This process continues until the freezing point of the slag is reached whereupon the slag solidifies around the grains of magnetite. All of the original silicates do not become molten and all of the iron oxides are not taken into solution. However, the unfused particles are mechanically trapped within the sinter proper.

The predominant effect of carbon in the sinter mix (other than providing heat units) is chemical reduction. When the carbon content is lowered, reduction is lessened and/or more oxidation occurs.

It also appears that the greater the amount of carbon used and burned, the greater will be the amount of glass-like material. This glass-like material tends to enamel the surfaces of the iron minerals, acts as a binding agent, and probably renders it more difficult to reduce.

**Reducibility Tests:** Previous work by other investigators has shown that hard sinters are more difficult to reduce than soft sinters.<sup>2, 8</sup> Investigations with our hard and soft magnetite and hematite sinters show similar results (fig. 12). It was found that soft sinters show a higher reduction rate than their parent ores, whereas hard sinters show a lower rate. Also, New York magnetites exhibit slower reduction rates than Mesabi hematites.

Based on plant experience within our Corporation, we find that blast furnace operators are not too concerned with the better reducibility of softer sinters. Their main requirement is a sinter hard enough to withstand reasonably well the wear and



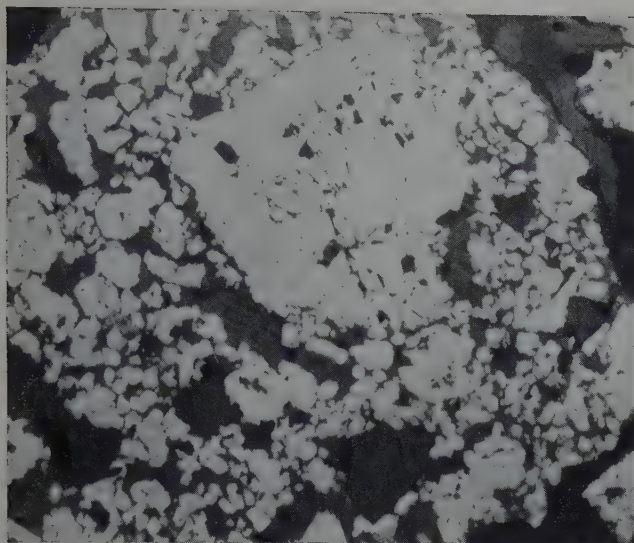


Fig. 11—Swedish sinter. White areas are hematite, light gray areas are magnetite, dark gray areas are slag. X60.

tear involved in transporting it from sinter plant to blast furnace skip.

#### Water Content

Sinter mixes have an optimum water content value for maximum flow of air through the bed. Small deviations from this optimum value result in appreciable decreases in airflow. Larger deviations result in a mix that will not sinter because it is too wet, or a mix that is partially pulled through the grate bars because it is too dry. The optimum value for any mix varies with the particle size distribution and the nature of the ore.

The effect of various water contents on airflow values in two sinter mixes, one composed of a coarse hematite, the other composed of a -20-mesh magnetic concentrate are shown in fig. 13. The data were obtained by making mixes of varying water contents, placing in the experimental sinter machine, and measuring the flow of air through the mixes at a definite vacuum. The mixes were not ignited to obtain these data. It can be seen that the hematite gives the best airflow at 10.5 pct moisture and the magnetite at 7.4 pct moisture. When the mixes were ignited and sintered, the fastest sintering rates were obtained from mixes containing amounts of moisture close to the optimum previously determined by airflow tests.

In the operation of tonnage sinter plants, close control of water in the mix is difficult. Plant application of moisture control is hindered by (1) variations in moisture content of raw materials, (2) large time lag between bins and swinging spout, (3) use of crude returns of varying size and temperature, and (4) lack of availability of a fast recording or indicating continuous moisture meter.

#### Solid Additions to the Mix

A bed of ore being sintered will have at least four general zones or layers of materials as follows: (1) Sinter layer, (2) fusion layer, (3) dried-out layer, (4) mix layer. Each layer will vary considerably in its permeability, thereby affecting the sintering rate. Coarse ores such as undersize wash ores exhibit a much higher sintering rate than intermediate size

ores. In our plants there is a 25 pct difference in sintering rate between a coarse ore with about 95 pct of its particles between  $\frac{1}{4}$ -in. and 100-mesh and an intermediate size concentrate with some 40 pct -100-mesh. Attention, therefore, has been directed towards the possibility of improving the overall permeability of the sinter bed by improving the permeability of the zones or layers of material present during the sintering operation.

The sinter layer, as would be expected, is the most permeable portion of the bed. Airflow readings on our experimental machine normally increase 300 pct to 400 pct from ignition to heat completion indicating that little could be accomplished with this layer.

Hartman<sup>4</sup> reports that basic additions to mixes of lean German ores have lowered the temperature of the fusion zone and increased the sintering rate. Work on our ores has not been successful along this line.

Examination of partially sintered heats shows that in our ores the dried narrow layer between the fusion zone and the mix appears to retain its structure well and we doubt that it is a major factor in retarding the sintering rate.

The layer of mix between the dried layer and the grate bars appears to be the main obstacle to an acceptable sintering rate in processing our intermediate size ore, and some progress has been made in alleviating this condition.

In addition to the layers of mix discussed above, there is the obstruction to airflow caused by the grate bars. Of necessity, the ratio of the solid area of grate bars to open area must be large in order to provide sufficient structural strength and resistance to excess burning and warping and to prevent the sucking through of large amounts of fines.

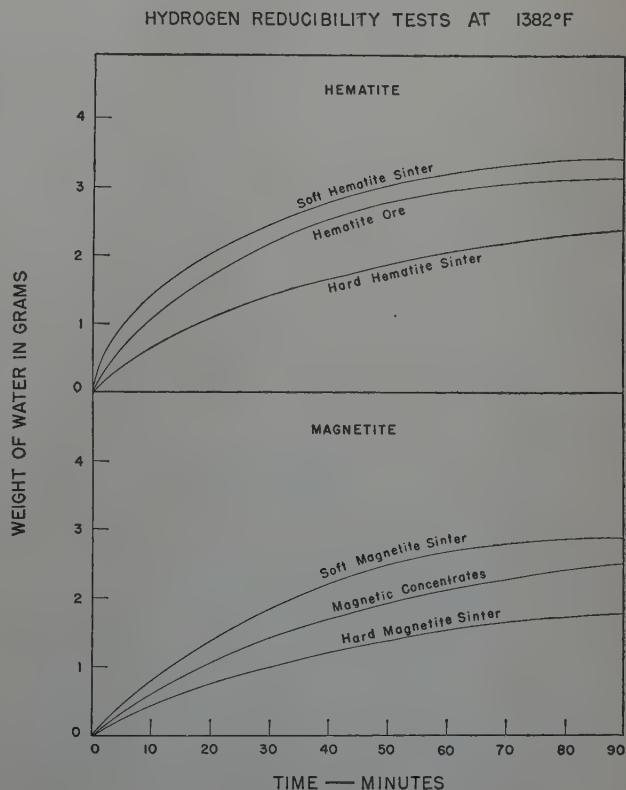


Fig. 12—Hydrogen reducibility tests, sinter.

# AIRFLOW TESTS — SINTER MIXES

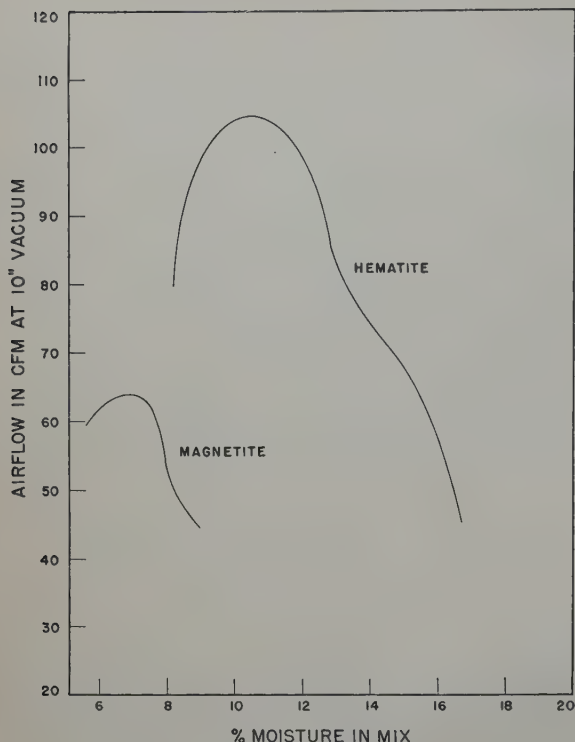


Fig. 13—Effect of water content on mix permeability.

In an effort to find out whether increased open area of the grate bars would aid materially in promoting airflow through the mix, a set of grate bars was designed and constructed for the experimental machine that increased the open area from a normal 23.5 pct to 42 pct. Subsequent tests on intermediate size ores showed no improvement in airflow during sintering.

Work on the mix layer itself has included studies on the effects of additions to the mix of (1) prepared agglomerates, (2) returns, and (3) lightweight fuel.

**Prepared Agglomerates:** In this part of the investigation, parts of normal mixes were removed and reformed as granules, briquettes or pellets. The agglomerates formed were then combined with the mix and sintered. In most cases the part removed was the -100-mesh fraction. In preparing the granules and briquettes, a number of binders were tried until one was found that enabled an agglomerate to withstand breakup until sintering could occur.

Fuel (as coke breeze) was added in various proportions to both the agglomerates and the mixes.

The results of these experiments were unfavorable as far as increasing the sintering rate was concerned. In some cases, particularly when attempting to sinter a mix with pellets, the mix showed improved permeability before ignition as indicated by airflow values higher than those usually obtained. After ignition, however, airflow values dropped markedly.

Examination of sinters made with prepared agglomerates reveals the difficulty of completely fusing briquettes or granules into sinter. Where fusion is not complete, the sulphur removal efficiency is poor. The magnetic concentrates tested contain 0.250 pct

# EFFECT OF LIGHTWEIGHT FUEL ON DENSITY OF MAGNETIC CONCENTRATE MIXES

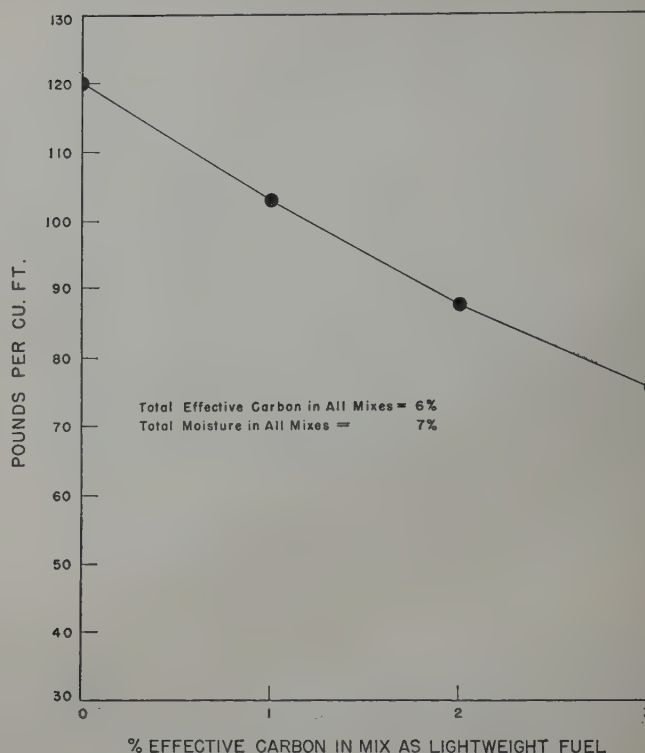


Fig. 14—Effect of sized sawdust on mix density.

to 0.300 pct sulphur and it is necessary to remove most of the sulphur to make a suitable blast furnace feed. One of the experimental heats made with prepared agglomerates showed a sulphur content of 0.043 pct in a completely fused section and a sulphur content of 0.106 pct in a partly fused section.

**Returns:** Returns larger than about 1/8 in. are of little or no value in promoting bed permeability and are a waste of good blast furnace feed. The problem seems to be determining the overall economics of operating a production sinter plant with completely processed returns.

Laboratory work shows that it is possible to prepare a mix with permeability equal to a normal mix by using less returns, if the returns are sized. Part or all of the returns larger than 1/8 in. could be reduced to a suitable size. Any remaining material could be added to the sinter production of the plant or utilized as a bedding material. Economic gains might be obtained from increased production due to improved bed permeability and/or by a reduction in the size of the circulating load.

The use of sized returns has been tried and abandoned by at least one other company<sup>5</sup> on economic grounds. It is hoped that further efforts along these lines will be fruitful.

**Lightweight Fuel:** The addition of fuels with a lower specific gravity than that of coal or coke is quite helpful in promoting the bed permeability of intermediate size ores. Materials such as sized sawdust, sized agglomerates made from fine sawdust, and buckwheat hulls have been used successfully. Shavings and peat moss have been tried without success with our magnetic concentrates.

The upper and lower size limits of the particles of the best lightweight fuel are critical and at the same time are difficult to sharply define. Above



about No. 3 mesh the particles are too large to be completely burned in the relatively short time cycle of the process. Somewhere between 28-mesh and 48-mesh these light fuel particles start to become detrimental to the permeability of the mix. Too much fine material of this kind robs the mix of its normal moisture, clogs air cavities, and causes the production of a nonuniform product. However, when the particle size is controlled, it is possible to greatly accelerate the sintering rate of our magnetite concentrates. It is possible to increase the bed permeability of the intermediate size ore so that it will sinter at a faster rate than a coarse hematite ore.

The effect of sized sawdust additions to a magnetic concentrate mix in reducing the weight per cubic foot is shown in fig. 14. The total effective carbon in all mixes was 6 pct, part of the carbon being supplied by the sized sawdust and part by coke breeze.

With 0 pct carbon as lightweight fuel, the density of the mix is 120 lb per cu ft. At 1 pct, this drops to 103 lb, at 2 pct the value is 88 lb, and at 3 pct the weight is 76 lb. For comparison, a coarse hematite ore mix will have a density of approximately 95 lb per cu ft. The sintering rate does not necessarily increase proportionally as the density of the mix decreases, but the effect is quite pronounced.

#### Effect of Cooling Rate on Sinter

Water cooling of sinter, compared to air cooling, has been the subject of some discussion in the last few years. Saussaman<sup>14</sup> in 1948 reported that air cooling of sinter had reduced the amount of  $\frac{1}{8}$ -in. sinter from 23 pct to between 3 pct and 8 pct at the Kaiser Plant in California. Tests in our plants and laboratory show that a decrease in the degree of drastic cooling results in improvements in this respect.

Figs. 15 and 16 show the effect of water and air quenching on a magnetic plant sinter, as revealed by the laboratory crush and tumbler tests. The sample was obtained by removing a large piece of sinter from one of the pellets before it was normally discharged. The sample was broken into two pieces. One piece was air cooled and the other quenched with a stream of water from a hose.

A pronounced difference in crush and tumbler results between the two differently processed sinters. These results represent extreme conditions of cooling not usually encountered in commercial sinters. However, the spread between curves is large in both cases and undoubtedly represents a major change in physical properties.

The differences between water-cooled and air-cooled sinters has been further studied with the microscope on experimental laboratory heats. Hard and soft sinters made from both coarse hematite and intermediate size magnetite were examined. An examination of the microstructures of two water-cooled magnetite sinters (figs. 17 and 18) revealed the following:

1. Watercooling alters the microstructure of magnetite sinter more drastically than it does that of hematite sinter.
2. More silicate binder is present in water-cooled sinter than in air-cooled sinter.
3. Silicate binder is more massive in water-cooled samples than in air-cooled samples.

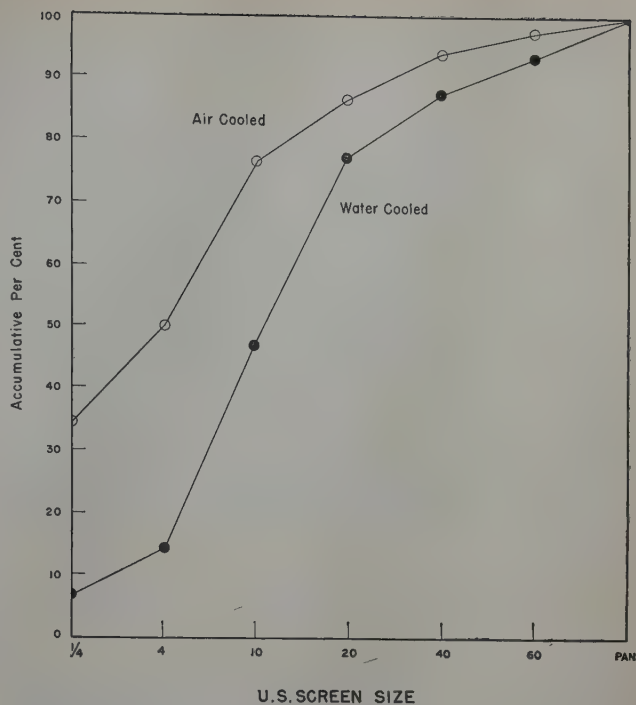


Fig. 15—Crush test results, water and air-cooled sinter.

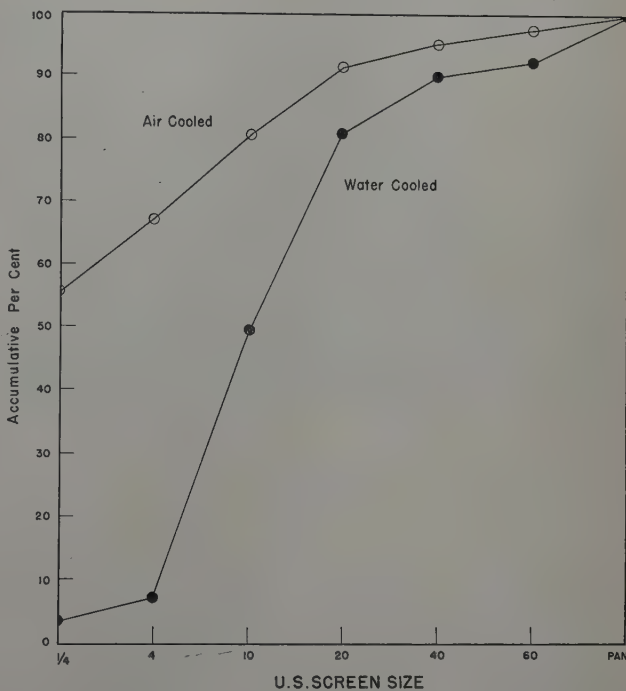


Fig. 16—Tumbler test results, water and air-cooled sinter.

4. Magnetite grains in water-cooled samples are not as coarse as those in air-cooled samples.

#### Summary

Investigational work has led to the development of two physical tests for sinter. These have been found to be valuable tools in experimental work.

Both fuel and moisture contents of sinter mixes are important variables requiring close control for

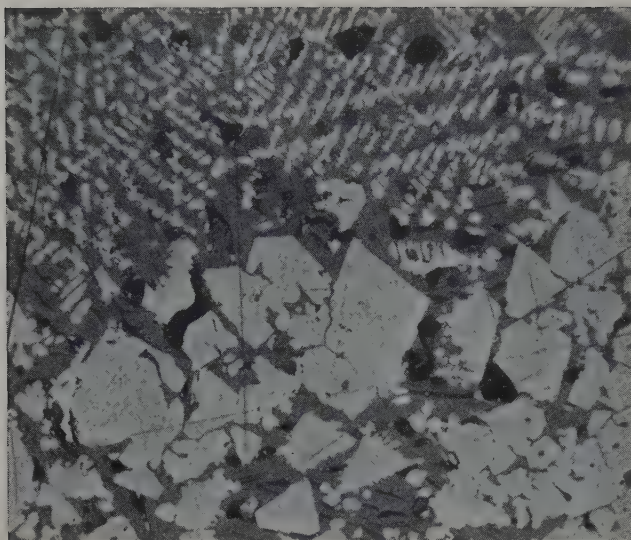


Fig. 17—Hard magnetite sinter. White areas are magnetite, light gray areas are slag. (Water cooled.) X100.

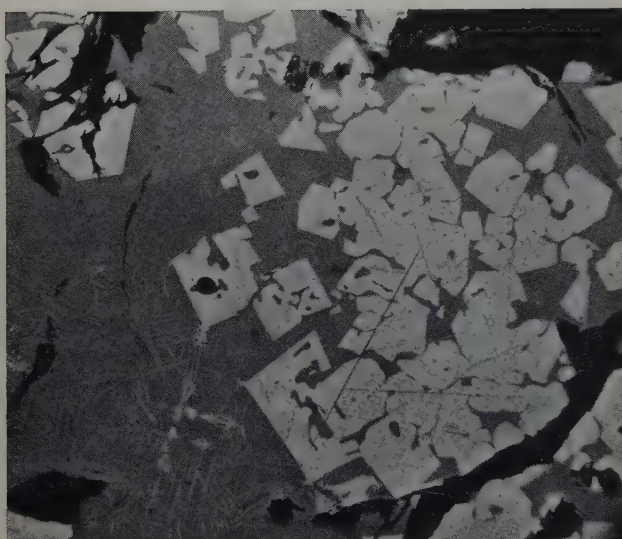


Fig. 18—Soft magnetite sinter. White areas are magnetite, light gray areas are slag. (Water cooled.) X100.

uniform operation and a uniform product. It is our observation that sufficiently accurate controls for these variables are not now available to the operators.

Reducibility tests show softer sinters to be less difficult to reduce than hard sinters, but blast furnace operators desire the hard sinters to avoid breakup in handling. Along this line, it is possible to decrease the amount of breakup if the sinter is air cooled.

Iron ores divided into three classes according to particle size distribution have been used as a basis for the work. The coarse group of ores sinter at a much faster rate than those of the intermediate group. In order to speed up the sintering rate of our intermediate group, it is necessary to improve the permeability of the mix layer, as distinguished from the three other layers present during sintering.

Additions of prepared solids to the mix can aid in this respect. When prepared agglomerates of the ore itself are added, difficulty in maintaining good permeability throughout the cycle is encountered as well as the further resistance at times of the agglomerates to fusing completely. Sized returns appear highly desirable, but the overall economics of their plant application has not been determined. The partial use of sized lightweight fuels increases the sintering rate considerably. However, because of high transportation costs of bulky material, application appears limited to plants located close to sources of supply.

As mentioned previously, we have as yet carried out little work on the fine group of ores. It seems to be the opinion in some quarters that sintering of fine material will not meet with undue difficulties, and the operation of the old Mesabi Iron Co. in Minnesota is cited as an example. We have no doubt that fine ores can be sintered, but there appears to be the important problem of sintering at an acceptable production rate. Examination of the literature<sup>6, 7, 8</sup> shows the above plant to have operated a 42-in. wide Dwight-Lloyd machine with a 6-in. bed at a speed of only 48 in. per min.

#### References

- <sup>1</sup> H. K. Work and F. M. Hamilton: Removal of Sulphur From High Sulphur Iron Ore by Sintering. *Proceedings Blast Furnace and Raw Materials Committee, AIME*, 1946, p. 85.
- <sup>2</sup> W. O. Philbrook: A Study of the Reducibility of Ores and Blast Furnace Sinter. *Blast Furnace and Steel Plant*. September 1943.
- <sup>3</sup> R. Hay and J. M. McLeod: The Principles Underlying the Sintering of Iron Ores. *Journal, West of Scotland Iron and Steel Institute*. 52, 6.
- <sup>4</sup> Fritz Hartman: Physical and Chemical Processes in the Sintering of Iron Ores. *Stahl und Eisen*, 63, 1943.
- <sup>5</sup> R. R. Burns: Operation and Practice, Producing Iron Sinter at Ducktown and Copperhill, Tennessee with Fine Flotation Sulphides. *Proceedings Blast Furnace, Coke Oven and Raw Materials Committee, AIME*, 1949, p. 146.
- <sup>6</sup> W. G. Swart: Mesabi Iron Company's Reduction Plant. *Proceedings, Lake Superior Mining Institute*, 1923, pp. 111-116.
- <sup>7</sup> A. B. Parsons: Operations of the Mesabi Iron Company. *Engineering and Mining Journal*, 117, No's. 4 and 5, Jan. 26 and Feb. 2, 1924.
- <sup>8</sup> Concentration of Eastern Mesabi Ore. *Engineering and Mining Journal*, 109, 1055, May 8, 1920.
- <sup>9</sup> A. Stanley and J. C. Mead: Sintering Characteristics of Minus Sixty-Five and Twenty Mesh Magnetite. *Transactions AIME*, 184, 181; *Mining Engineering*, June 1949, TP 2597BC.
- <sup>10</sup> A. H. Fosdick: Operating Features and Practices at the Bethlehem Sintering Plant. *Proceedings Blast Furnace, Coke Oven and Raw Materials Committee, AIME*, 1948, p. 106.
- <sup>11</sup> H. Wittenberg and K. Myer: Experiments to Increase the Capacity in the Suction Sintering of Iron Ores. *Stahl und Eisen*, 63, No. 45, 1943.
- <sup>12</sup> Karl Kintzinger: Operative Measures in Sintering Iron Ores. *Stahl und Eisen*, 63, No. 23, 1943.
- <sup>13</sup> T. W. Plante: Good Sinter and its Production. *Blast Furnace and Steel Plant*. (1946) 34, 1515-1519; (1947) 35, 100-114, 116-117.
- <sup>14</sup> J. D. Saussaman: Sintering Practice at Fontana, California. *Proceedings Blast Furnace, Coke Oven and Raw Materials Committee, AIME*, 1948, p. 95.



# MINING TRANSACTIONS

January-December 1950

## Index to Volume 187

### (Pages in Each Issue)

January .....	81- 128
February .....	201- 296
March .....	355- 402
April .....	467- 510
May .....	571- 622
June .....	679- 718
July .....	779- 810
August .....	871- 902
September .....	951- 982
October .....	1031-1078
November .....	1123-1170
December .....	1251-1282

**American Institute of Mining and Metallurgical Engineers, Inc.**

29 West 39th Street, New York 18, N. Y.





# Mining Transactions

## A

Accident prevention: see Safety	
Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions.....	952
Adsorption: dodecylamine on quartz.....	499
Aggregates: concrete.....	395
discussion.....	1168
lightweight: bloated.....	481
manufacture of.....	479
sintered.....	483
Air Pollution by Industrial Fumes, Gases, and Dusts.....	971
Alabama: fluoride in ground water.....	886
geology of.....	886
Albertson, F. E. and Knobler, R. R.: Discussion on An Improved Method of Gravity Concentration in the Fine Size Range.....	1156
Alexander, A. J.: Specific Safety Problems Applicable to West Virginia Mines.....	967
Allen-Sherman-Hoff Co.: Automatic Controls on Sand Pumps.....	801
Alizarin dyes: effect on soap flotation of cassiterite and fluorite.....	601
Allis-Chalmers Manufacturing Co.: A New Theory of Comminution.....	871
Allyn, C. L., Geer, M. R., Yancey, H. F., and Eckhouse, R. H.: Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1057
discussion.....	1160
Aluminum, activation.....	597
Ameen, H. F. and Hamilton, F. M.: Laboratory Studies on Iron Ore Sintering and Testing.....	1275
American Cyanamid Co.: Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units.....	365
Radiotracer Studies on the Interaction of Dithiophosphate with Galena.....	359
Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries.....	790
American Smelting and Refining Co.: Sinking Star Shaft at Vanadium, N. Mex.....	81
Ammonia plants.....	493
Ammonium sulphate, production.....	391
Anaconda Copper Mining Co.: Practical Dust Control in Metal Mines.....	86
Anderson, W. W.: Quantitative Efficiency of Separation of Coal Cleaning Equipment.....	256
discussion.....	1162
Anode slimes, flotation of.....	1131
Arentz, S. S.: Health and Safety Practices at Picoche.....	204
Arkansas, nonmetalliferous mineral resources.....	577
Armiger, W. H., Hill, W. L. and Gooch, S. D.: Some Properties of Pseudowavellite from Florida.....	699
Armstrong, L. C.: Diamond Drilling Quartz-Feldspar Intergrowths. discussion.....	1148
Arsenopyrite, Monserrat Mine.....	1076
Asbestos ore; caving of.....	467
Aspects of Structure and Mineralization Used as Guides in the Development of the Picher Field.....	1251
Atlas Powder Co.: Some Applications of Millisecond Delay Electric Blasting Caps in mining.....	1123
Automatic Controls on Sand Pumps.....	801
Ayrshire Collieries Corp. and Subsidiaries: Laboratory Control in Coal Washing and Drying Plants.....	609

## B

Bain, C. Kremer: Problems of a Mine Mechanization Program.....	201
Ball mill: automatic control of load.....	1040
ball consumption.....	1133
cylindrical grate, grinding tests.....	96
effect of speeds on grinding efficiency.....	1127
feeders, classifiers used as.....	951
grinding balls, relative wear rates.....	712
grinding efficiency.....	1133
grinding silica sand.....	389
slow speed, closed circuited with hydrosclillator.....	707
Ball milling: closed circuit tests.....	1270
grindability.....	1272

probability theory.....	1267
laboratory mill.....	1270
Barite, Arkansas deposits.....	582
Barium, activation of quartz.....	593
Barium chloride, effect on soap flotation of quartz.....	591
Bastas Turk Maadin Ltd.: Chromite and other Mineral Occurrences in the Tasepe District of Eskisehir, Turkey.....	108
Battelle Memorial Institute: Operating Data for a Bird Centrifuge.....	381
Baumann, H. A., Spicer, T. S. and Wright, C. C.: Some Factors Influencing the Performance of Single Retort Underfeed Stokers.....	265
Bean, James J.: Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units.....	365
discussion.....	1153
Behavior of Mineral Particles in Electrostatic Separation.....	369
Bell, Roscoe E.: Economic Factors in the Western Phosphate Industry.....	486
Bentonite, Arkansas deposits.....	584
Bernhardt, Joseph: Use of Concrete Underground.....	355
Beryl, concentration of.....	495
Bertholf, W. M.: Discussion on Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1160
Bethlehem Steel Co.: Use of Concrete Underground.....	355
Bethune, A. Y. and Wolf, W. G.: The Burt Filter.....	585
Bird centrifuge, operating data.....	381
discussion.....	1166
Bituminous Coal Research, Inc.: Organizing and Financing Cooperative Research.....	374
Blasting: electric caps.....	1123
shaft sinking.....	82
Bloecher, F. W., Jr. and Gaudin, A. M.: Concerning the Adsorption of Dodecylamine on Quartz.....	499
discussion.....	1154
Bolivia, tin deposits.....	1071
Bond, F. C. and Wang, Jen-Tung: A New Theory of Comminution.....	871
Bond, F. C.: Discussion on The Effect of Mill Speeds on Grinding Costs.....	1149
Borax, crystallization of cassiterite from Bore hole surveys.....	602
Bournonite, Monserrat Mine.....	1037
Brazil, geology of.....	1076
heavy mineral deposits.....	687
Brown, W. E.: Discussion on Laboratory Performance Tests of the Humphreys Spiral As a Cleaner of Fine Coal.....	685
Stone of the Crab Orchard District, Tennessee.....	1161
Building.....	883
Burgess, B. D.: Discussion on Ground Water in California.....	1168
Burt Filter.....	585

## C

Cady, Gilbert H.: Research in Coal Geology.....	275
Calcium, activation of quartz.....	597
Calculating machines for plane triangulation system.....	207
California Div. of Mines: California Tales.....	122
California, ground water in.....	279
California Tales.....	122
Canadian Dept. of Mines and Resources: An Improved Method of Gravity Concentration in the Fine-Size Range.....	879
Canadian Deposits of Uranium and Thorium.....	239
Canadian Johns-Manville Co.: Review of Progress in Caving of Asbestos Ore.....	467
Carbon: applied to cyanidation.....	217
reactivity.....	805
Cassiterite: effect of activators and alizarin dyes on soap flotation of Monserrat Mine, Bolivia.....	601
treatment.....	1071, 1073
Castle Dome Copper Co.: Conditioning and Treatment of Sulphide Flotation Concentrates Preparatory for the Separation of Molybdenite at the Miami Copper Company.....	879

Catavi mill, tin recovery.....	880
Caving: asbestos.....	467
Celestite, Arkansas deposits.....	583
Cement, Lepol kilns.....	289
Centrifuge, Bird.....	381
Chalcocite, flotation of.....	226
Chapman, T. G., Crabtree, E. H., Jr., and Winters, V. W.: Developments in the Application of Activated Carbon to Cyanidation Including the Desorption of Gold and Silver from Carbons.....	217
Chataignon, P., Rey, M., Formanek, V.: The Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate.....	1126
Chekang University: a New Theory of Comminution.....	871
Chert: Arkansas deposits.....	582
Tri-State district.....	1251
Chromite and Other Mineral Occurrences in the Tasepe District of Eskiseher, Turkey.....	108
correction.....	702
Chromite: Turkey.....	108
Chupak, J., Simard, G. L., and Salley D. J.: Radiotracer Studies on the Interaction of Dithiophosphate with Galena.....	359
correction.....	789
Circuitron, Massco.....	1038
discussion.....	1155
Clark, J. D.: Petalite—A New Commercial Mineral.....	1068
Classification: Northern Rhodesia Copper Mines.....	96
Classification and Application of Drill Jibs for Rock Drill Mounting.....	571
Classifier: Aikins.....	880
automatic control of feed.....	1039
closed circuited.....	708
spiral, used as ball mill feeders.....	951
Classifying: fine size.....	879
graded glass sand.....	385
kaolin.....	695
Clay, Arkansas resources.....	577
treatment.....	695
Cleaning of Fine Sizes of Bituminous Coals by Concentrating Tables.....	956
Cleveland-Cliffs Iron Co.: Development in the Use of Steel for Underground Support.....	475
Coal: bituminous, cleaning by concentrating tables.....	956
cleaning: Humphreys spiral.....	1057
Rheolaveur system.....	1137
quantitative efficiency.....	256
discussion.....	1162
continuous mining machine.....	715
cyclone separator.....	790
drying in entrained and fluidized state.....	974
face stripping.....	1262
fine coal cleaning: cyclone separator.....	790
kerosine flotation.....	1047
flotation: raw coal.....	1050
sludge.....	1049
froth flotations.....	616
geobotanical investigations.....	276
geochemistry.....	277
hydrogenation of.....	508
industry, young mining engineer in.....	718
mechanized cutting.....	1260, 1262, 1265
petrographic composition.....	889
petrography.....	277
plow.....	1260
recovery by cyclone thickener.....	106
resources study.....	276
retort stokers.....	265
safety in mines.....	867
scrapers.....	1265
washing and drying plants.....	609
Coal Preparation for Synthetic Liquid Fuels.....	507
Cobalt ore, concentration of.....	1044
Coke, low-temperature, as reactive carbon.....	805
Colls, E. A. G.: Corrosion Resistant Materials and Coatings in Trail Chemical Operations.....	491
Colmol—A Continuous Mining Machine.....	715
Colquiri mill, tin recovery.....	881
Columbia University: The Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions.....	952
Combined Metals Reduction Co.: Electronic Tramp Iron Detector for Ore Conveyor Belts.....	703



Health and Safety Practices at Ploche Commercial Testing and Engineering Co.: Quantitative Efficiency of Separation of Coal Cleaning Equipment.....	204	Dental investigations, effect of fluoride Deposits of Heavy Minerals on the Brazilian Coast.....	587	Ferguson, R. C. and Hardinge, H.: The Effect of Mill Speeds on Grinding Costs.....	1127
Comminution, new theory of.....	256	Desliming; glass sand.....	1159	discussion.....	1149
Comparative Furnace Designs for the Expansion of Perlite.....	111	Desorption: gold and silver from carbons.....	217	Ferney, F. X.: Discussion on Operating Data for a Bird Centrifuge.....	1166
Concentrating: Humphreys spiral.....	1057	Determination of the Coefficient of Linear Thermal Expansion of Rock Specimens by Means of Resistance Wire (SR-4) Strain Gauges.....	683	Fertilizer, phosphate.....	486
Concentration: aluminum phosphate.....	788	Development in the Use of Steel for Underground Support.....	475	Filters: Burt clay.....	585
cobalt ore.....	700	Development Work with Trackless Equipment.....	579	sand pumps.....	698
gravity.....	1044	Developments in the Application of Activated Carbon to Cyanidation Including the Desorption of Gold and Silver from Carbons.....	217	Fine, M. M.: Production of Graded Glass Sand by Grinding and Classification.....	803
Sullivan deck.....	881	Dewatering: coal.....	1055	Fink, C. G. and Putnam, G. L.: The Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions.....	952
tables, cleaning of bituminous coal.....	879	Disco Co.: Low-Temperature Coke as a Reactive Carbon.....	805	Finland Institute of Technology: Separation of Precious Metals from Anode Slimes by Flotation.....	1131
Concerning the Adsorption of Dodecylamine on Quartz.....	956	Distribution analyzer, use in electrostatic separation of minerals.....	369	Float-and-sink tests.....	1057
Concrete: aggregates for large structures use in underground slushing drifts.....	499	Dithiophosphate, radiotracer studies.....	359	Flocculation cell, flotation of sludge.....	1053
Conditioning, effect of flotation of chalcocite.....	395	Dodecylamine, adsorption on quartz.....	499	Florida: pseudowavellite.....	699
Conditioning and Treatment of Sulphide Flotation Concentrates Preparatory for the Separation of Molybdenite at the Miami Copper Company.....	506	discussion.....	1154	waste disposal of phosphate rock industry.....	779
Conductance Electrostatic Separation with Convective Charging.....	1041	Doherty, J. D., Crenzt, W. L., and Donath, E. E.: Coal Preparation for Synthetic Liquid Fuels.....	507	Flotation: chalcocite.....	235
Conley, John E. and Ruppert, John A.: Recent Developments in the Manufacture of Lightweight Aggregates.....	479	discussion.....	1158	froth: coarse coal.....	616
Consolidated Mining and Smelting Co.: Corrosion Resistant Materials and Coating in Trail Chemical Operations.....	491	Dolomite, Arkansas deposits.....	582	glass sands.....	1140
Continuous Countercurrent Decantation Calculation.....	223	Donath, E. E., Crenzt, W. L. and Doherty, J. D.: Coal Preparation for Synthetic Liquid Fuels.....	507	kerosine, bituminous coal fines.....	1047
Cooper, M. D.: The Young Mining Engineer in the Coal Industry.....	718	discussion.....	1158	lead carbonate.....	1159
Cordero Mining Co.: An Unusual Test of the Accuracy of Well-Surveying Methods.....	1037	Dorr Co.: Continuous Countercurrent Decantation Calculations.....	223	quartz, correlation with adsorption data.....	505
Corrosion Resistant Materials and Coating in Trail Chemical Operations.....	491	Probability Theory of Wet Ball Milling and Its Application.....	1267	separation of precious metals from anode slimes.....	1131
Costs: chromite mining in Turkey.....	110	Driessen, M. G.: Discussion on Quantitative Efficiency of Separation of Coal Cleaning Equipment.....	1164	sillimanite ore.....	788
coal washing and drying plants.....	610	Driessen, M. G. and Criner, H. E.: Cyclone Thickener Applications in the Coal Industry.....	102	soap: effects of activators and alizarin dyes on cassiterite and fluorite.....	601
drill jibs.....	574	discussion.....	1158	effect of BaCl <sub>2</sub> and other activators on quartz.....	591
flotation plants.....	1053	Drifting: comparison of methods.....	679	vacuum.....	592
freight rates, Northwest industrial minerals.....	288	Drilling: jibs.....	571	Fluoride in Ground Water of Alabama.....	886
garnet production.....	576	jumbo.....	201	Fluorite, effect of activators and alizarin dyes on soap flotation of.....	601
haulage drifts.....	358	millisecond delay blasting caps.....	1123	Foot Mineral Co.: Petalite—A New Commercial Mineral.....	1068
low ash coal in hydrogenation.....	509	Drying: coal, laboratory control.....	609	Formanek, V., Rey, M. and Chataignon, P.: The Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate.....	1126
phosphate production.....	490	Drying Fine Coal in the Entrained and Fluidized State.....	974	Fraas, F. and Ralston, O. C.: Conductance Electrostatic Separation with Convective Charging.....	1041
production, ammonium sulphate and manganese oxides.....	394	du Pont de Nemours and Co.: Deposits of Heavy Minerals on the Brazilian Coast.....	685	Discussion on Behavior of Mineral Particles in Electrostatic Separation.....	1152
shaft sinking.....	85	Dust control, metal mines.....	86	France: potash deposits.....	119
Counselman, T.B.: Continuous Countercurrent Decantation Calculations.....	223	E.....		Freyberger, W. L., Gaudin, A. M., Dasher, J., Pannell, J. H.: Use of an Induced Nuclear Reaction for the Concentration of Beryl.....	495
discussion.....	1155	Eagle-Picher Mining and Smelting Co.: Aspects of Structure and Mineralization Used as Guides in the Development of the Picher Field.....	1251	Frothmeter, visual observations of a flotation cell.....	617
Crabtree, E. H., Jr., Winters, V. W., and Chapman, T. G.: Developments in the Application of Activated Carbon to Cyanidation Including the Desorption of Gold and Silver from Carbons.....	217	Developments in the Application of Activated Carbon to Cyanidation Including the Desorption of Gold and Silver from Carbons.....	217	Fuel Economy in the Lepol Kilns.....	289
Craig, A. E., Tait, W. J. and McCurdy, E. P.: Preliminary Report of Masco Circuitron.....	1038	Use of Spiral Classifiers as Ball Mill Feeders.....	951	Fuel, synthetic liquid, from coal.....	507
discussion.....	1155	Eckhouse, R. H., Geer, M. R., Yancey, H. F., Allyn, C. L.: Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1057	Fuller's earth, Arkansas deposits.....	584
Crandall, J. S.: Production and Marketing of Garnet Abrasive Sands from Emerald, Benewah County, Idaho.....	575	discussion.....	1160	Furnaces: expansion of perlite.....	111
Crenzt, W. L., Doherty, J. D., Donath, E. E.: Coal Preparation for Synthetic Liquid Fuels.....	507	Economic Factors in the Western Phosphate Industry.....	486	horizontal stationary.....	113
discussion.....	1158	Education: mining engineer.....	718	indirectly-fired rotary.....	116
Criner, H. E. and Driessen, M. G.: Cyclone Thickener Applications in the Coal Industry.....	102	Effect of BaCl <sub>2</sub> and Other Activators on Soap Flotation of Quartz.....	591	long concurrent rotary.....	118
discussion.....	1158	Effect of Conditioning on Flotation of Chalcocite.....	226	multiple-hearth.....	115
Cross, A. T.: Discussion on Coal Preparation for Synthetic Liquid Fuels.....	1158	Effect of Mill Speeds on Grinding Costs.....	1127	short concurrent rotary.....	115
Crushing: rock.....	871	Effect of Waste Disposal of the Pebble Phosphate Rock Industry in Florida on Condition of Receiving Streams.....	779	vertical stationary.....	114
Curtis, C. H.: Conditioning and Treatment of Sulphide Flotation Concentrates Preparatory for the Separation of Molybdenite at the Miami Copper Co.....	506	Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite.....	601		
Cutter loaders, coal.....	1260	Effects of Rod Mill Feed Size Reduction.....	1273	G.....	
Cyanidation: application of activated carbon.....	217	Eldorado Mine, pitchblende.....	244	Galena: interaction with dithiophosphate.....	359
decantation calculations.....	223	Eldorado Mining and Refining (1944) Ltd.: Canadian Deposits of Uranium and Thorium.....	239	Monserat mine.....	1075
dissolution of gold.....	952	Electrodes, use in electrostatic separation.....	1041	Gangue, Monserat Mine.....	1076
Cyclone separator, Dutch State Mines.....	790	Electronic Tramp Iron Detector for Ore Conveyor Belts.....	703	Garms, W. I.: Discussion on Progress Report on Grinding at Tennessee Copper Company.....	1151
discussion.....	1153	Electrostatic separation: behavior of mineral particles.....	369	Garnet, abrasive sands, production and marketing.....	575
Cyclone Thickener Applications in the Coal Industry.....	102	discussion.....	1152	Gas recovery tests, various reducing agents.....	808
D.....		conductance.....	1041	Gaudin, A. M. and Bloecher, F. W., Jr.: Concerning the Adsorption of Dodecylamine on Quartz.....	499
Dahlstrom, D. A.: Discussion on Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Shurries.....	1153	Estill, R. R.: Mechanized Cutting and Face Stripping in Ruhr.....	1260	discussion.....	1154
Dasher, J., Gaudin, A. M., Pannell, H., Freyberger, W. L.: Use of an Induced Nuclear Reaction for the Concentration of Beryl.....	495	Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units.....	365	Gaudin, A. M., Dasher, J., Pannell, J. H. and Freyberger, W. L.: Use of an Induced Nuclear Reaction for the Concentration of Beryl.....	495
Davis, E. W.: Discussion on Relative Wear Rates of Various Diameter Grinding Balls in Production Mills.....	714	Extensometer, sag measurements with.....	1033	Geer, M. R. and Yancey, H. F.: Discussion on Quantitative Efficiency of Separation of Coal Cleaning Equipment.....	1163
Diamonds, Arkansas deposits.....	584	F.....		Geer, M. R., Yancey, H. F., Allyn, C. L. and Eckhouse, R. H.: Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1057
Diamond Drilling Quartz-Feldspar Intergrowths, discussion.....	1148	Falconer, S. A.: Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Shurries.....	790	discussion.....	1160
Diamond drilling, radioactive minerals.....	252	discussion.....	1153	Geological Survey of Canada: Canadian Deposits of Uranium and Thorium.....	239
Deister diagonal-deck table.....	959	Farbenindustrie: Coal Preparation for Synthetic Liquid Fuels.....	507	Geology: aggregate sources.....	395
Density recording instrument, for heavy-media separation units.....	365	Faster Calculation of Plane Triangulation Systems by Calculating Machine and Semigraphical Methods.....	207	Brazil.....	685
				building stone, Tennessee.....	883
				chromite and other Turkish minerals.....	108
				Eldorado Mine, Canada.....	245
				garnet sands.....	575
				ground water in California.....	279
				magnetic fields with igneous pipes.....	1143
				main geological divisions of Canada.....	241
				nonmetalliferous minerals, Arkansas.....	577
				Picher, zinc-lead mining field.....	1251
				potash, Germany, France and Spain.....	1175
				research in coal.....	276
				rock, Alabama.....	886
				sillimanite of the southeast.....	786
				tin deposits, Bolivia.....	1071



Germany, potash deposits.....	117
Gibson, R. and Turneure, F. S.: Tin Deposit of the Monserrat Mine, Bolivia.....	1071
Gildersleeve, B.: Building Stone of the Crab Orchard District, Tennessee.....	883
Gillson, J. L.: Deposits of Heavy Minerals on the Brazilian Coast.....	685
Glass sand: Arkansas resources.....	580
beneficiation of.....	1139
production by grinding and classification.....	385
Godfrey, L. Cabot, Inc.: Industrial Mineral Economics and the Raw Materials Survey.....	292
Gold: desorption from carbon.....	217
dissolution by cyanide solutions.....	952
separation from anode slimes.....	1131
Gold leaf test, dissolution of gold by cyanide solutions.....	952
Goldfields Area, Canada: geology.....	243, 250
radioactive minerals.....	251
Gooch, S. D., Hill, W. L., and Armiger, W. H.: Some Properties of Pseudowavellite from Florida.....	699
Granite, Canadian deposits.....	242
Greensand, Arkansas deposits.....	583
Griffen, J.: Rheolaveur System of Fine Coal Cleaning.....	1137
Discussion on Quantitative Efficiency of Separation of Coal Cleaning Equipment.....	1162
Grinding: automatic control by Circuitron.....	1038
discussion.....	1155
ball mill tests: Mufulira Copper Mines Ltd.....	96
probability theory.....	1267
effect of mill speeds on.....	1127
discussion.....	1149
graded glass sand.....	385
relative wear rates of balls.....	712
rock.....	871
Tennessee Copper Co., progress reports.....	707, 1133
discussion.....	711, 1151
Ground water, fluoride in.....	886
Ground Water in California.....	279
discussion.....	1169
Guillotine machine, stone cutting.....	885
Gypsum, Arkansas deposits.....	583

## H

Haller, F. J.: Development in the Use of Steel for Underground Support.....	475
Hamburger, Richard: Faster Calculation of Plane Triangulation Systems by Calculating Machine and Semigraphical Methods.....	207
Hamilton, F. M. and Ameen, H. F.: Laboratory Studies on Iron Ore Sintering and Testing.....	1275
Hardinge Co.: The Effect of Mill Speeds on Grinding Costs.....	1127
Hardinge, H. and Ferguson, R. C.: The Effect of Mill Speeds on Grinding Costs.....	1127
discussion.....	1149
Hardinge, H.: Discussion on An Electronic Tramp Iron Detector for Ore Conveyor Belts.....	706
Discussion on Progress Report on Grinding at Tennessee Copper Company.....	711
Harvard University: Tin Deposits of Monserrat Mine, Bolivia.....	1071
Haulage: asbestos ore.....	473
Northwest industrial minerals.....	285
Haulages, concreting of.....	358
Health and Safety Factors at Piche.....	204
Health program, Piche silver mine.....	205
Heavy-media separation units.....	365
discussion.....	1153
Heavy minerals, recovery of.....	879
Hematite ore, beneficiation of.....	1042
Heyl and Patterson: Cyclone Thickener Applications in the Coal Industry.....	102
Operating Data for a Bird Centrifuge.....	381
Hill, W. L., Arminger, W. H., and Gooch, S. D.: Some Properties of Pseudowavellite from Florida.....	699
Hoisting: asbestos ore.....	474
Holmes, C. R.: Magnetic Fields Associated with Igneous Pipes in the Central Ozarks.....	1143
Hudson, S. B.: Discussion on Behavior of Mineral Particles in Electrostatic Separation.....	1152
Hukki, R. T. and Runicinna, U.: Separation of Precious Metals from Anode Slimes by Flotation.....	1131
Hydrogen plants.....	493
Hydrogenation, coal.....	508
Hydrooscillator: grinding efficiency.....	710
operation of.....	1133

## I

Idaho Garnet Abrasive Co.: Production and Marketing of Garnet Abrasive Sands from Emerald Creek, Benewah County, Idaho.....	575
Improved Method of Gravity Concentration in the Fine-size Range.....	879
Industrial Mineral Economics and the Raw Materials Survey.....	292
Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate.....	1126
Inspiration Consolidated Copper Co.: Faster Calculation of Plane Triangulation System by Calculating Machine and Semigraphical Methods.....	207

International Uranium Ltd., Silver-pitchblende Deposit.....	248
Iron: activation.....	598
electronic detector for conveyor belts.....	703
Iron ore: sintering and testing.....	1275
Iron oxide, reduction of.....	807
Isotherms, from galena preparations.....	361

## J

James, W. F., Lang, A. H., Murphy, R., and Kesten, S. N.: Canadian Deposits of Uranium and Thorium.....	239
Jenkins, R. W. and Neslage, O. J.: A Classification and Application of Drill Jibs for Rock Drill Mounting.....	571
Jibs, drill, classification and application of.....	571
Johnson, O.: Discussions: On Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills.....	1151
On the Effect of Mill Speeds on Grinding Costs.....	1149
Johnson's Co.: Review of Progress in Caving of Asbestos Ore.....	467
Jones and Laughlin Steel Corp.: Laboratory Studies on Iron Ore Sintering and Testing.....	1275
Jones, E. A.: Development of Work with Trackless Equipment.....	679
Joy Manufacturing Co.: A Classification and Application of Drill Jibs for Rock Drill Mounting.....	571
Jumbos: boom-type.....	571
jib-type.....	572
self-propelled.....	573

## K

Kaiser, E. R.: Organizing and Financing Cooperative Research.....	374
Kaolin Production and Treatment in the South.....	694
Kellogg, H. H.: Discussion on Measurement of Equilibrium Forces Between and Air Bubble and an Attached Solid in Water.....	1154
Kerosine Flotation of Bituminous Coal Fines.....	1047
Kesten, S. N., James, W. F., Lang, A. H., Murphy, R.: Canadian Deposits of Uranium and Thorium.....	239
Ketzlach, Norman: Production of Ammonium Sulphate and Manganese Oxides.....	391
Kick theory.....	873
Kiln: Lepol cement.....	289
rotary.....	479
King, T. C.: The Use of Spiral Classifiers as Ball Mill Feeders.....	951
Kinzie, R. A., Jr.: Fuel Economy in the Lepol Kilns.....	289
Knobler, R. R. and Albertson, F. E.: Discussion on An Improved Method of Gravity Concentration in the Fine Size Range.....	1156
Korman, S. and Tuwiner, S. B.: Effect of Conditioning of Flotation of Chalcocite.....	226
Kromer, F.: Chromite and Other Mineral Occurrences in the Tasepe District of Eskisehir, Turkey.....	108
correction.....	702

## L

Laboratory Control in Coal Washing and Drying Plants.....	609
Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1057
discussion.....	1160
Laboratory Studies on Iron Ore Sintering and Testing.....	1275
Ladoo, Raymond E. and Stokes, C. A.: Industrial Mineral Economics and the Raw Materials Survey.....	202
discussion.....	1169
LaMoreaux, P. E.: Fluoride in Ground Water of Alabama.....	886
Lang, A. H., James, W. F., Murphy, R., and Kesten, S. N.: Canadian Deposits of Uranium and Thorium.....	239
Launders, fine coal.....	1137
Leaching: acid, glass sands.....	1141
Burt filter.....	585
manganese ore.....	391
Lead carbonate, flotation of.....	1126
Leith, C.: Some Factors in Selection and Testing of Concrete Aggregates for Large Structures, Discussion, reply for author.....	1168
Leshner, C. E.: Low-Temperature Coke as a Reactive Carbon.....	805
Lewis, F. M.: Discussion on Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units.....	1153
Lewis, F. M. and Myers, J. F.: Progress Report on Grinding at Tennessee Copper Company.....	707, 1133
discussion.....	711, 1151
Lightweight Aggregate Industry in Oregon, discussion.....	1170
Limestone, Arkansas deposits.....	552
Loaders: cutter.....	1261
Joy.....	680
Loading: continuous mining machine.....	715
trackless equipment.....	679
Low-Temperature Coke as a Reactive Carbon.....	805

Lyden, J. P.: Aspects of Structure and Mineralization Used as Guides in the Development of the Picher Field.....	1251
Lyons, Orville R. and Richardson, A. C.: Operating Data for Bird Centrifuge.....	381
discussion.....	1166

## M

Magnesite: Turkey.....	109
Magnetic Fields Associated with Igneous Pipes in the Central Ozarks.....	1143
Manganese oxides, production.....	391
Manganese Products, Inc.: Production of Ammonium Sulphate and Manganese Oxides.....	391
Marcasite, Monserrat Mine.....	1075
Marquardt, C. M.: An Electronic Tramp Iron Detector for Ore Conveyor Belts.....	703
discussion.....	706
discussion on Preliminary Report of Massco Circuitron.....	1155
Massachusetts Institute of Technology: Concerning the Adsorption of Dodecylamine on Quartz.....	499
Effect of BaCl <sub>2</sub> and other Activators on Soap Flotation of Quartz.....	591
Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite.....	601
Improved Method of Gravity Concentration in the Fine-size Range.....	879
Use of an Induced Nuclear Reaction for the Concentration of Beryll.....	495
Mason, R. S. and Wagner, N. S.: Lightweight Aggregate Industry in Oregon, discussion.....	1170
Mather, W. B.: Nonmetalliferous Mineral Resources in Arkansas.....	577
Discussion on Lightweight Aggregate Industry in Oregon.....	1170
Discussion on Recent Developments in the Manufacture of Lightweight Aggregates.....	1170
May, A. J.: Sinking Star Shaft at Vanadium, New Mexico.....	81
McCabe, L. C.: Air Pollution by Industrial Fumes, Gases and Dusts.....	971
McCurdy, E. P., Craig, A. E. and Tait, W. J.: Preliminary Report of Massco Circuitron.....	1038
discussion.....	1155
McFarland, D. M.: Some Applications of Millisecond Delay Electric Blasting Caps in Mining.....	1123
McKinnon, W. D.: Discussion on Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills.....	1151
McLachlan, C. G.: Discussion on Continuous Countercurrent Decantation Calculations.....	1155
McLaughlin, J. F. and Otto, E. H.: Discussion on Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1161
McNally Pittsburgh Manufacturing Corp.: Behavior of Mineral Particles in Electrostatic Separation.....	369
Rheolaveur System of Fine Coal Cleaning.....	1137
Measurement of Equilibrium Forces between an Air Bubble and an Attached Solid in Water.....	91
Mechanism of Coarse Coal and Mineral Froth Flotations.....	616
Mechanization, mine.....	201
Mechanized Cutting and Face Stripping in the Ruhr.....	1260
Meerschbaum: Turkey.....	110
Merle, J. J. and Mullins, R. A.: Laboratory Control in Coal Washing and Drying Plants.....	609
Miami Copper Co.: Conditioning and Treatment of Sulphide Flotation Concentrates Preparatory for the Separation of Molybdenite at the Miami Copper Company.....	506
Milling: ball, see ball mill	
pebble, see pebble mill	
rod, see rod mill	
sillimanite ore.....	788
Mills: wear rates of grinding balls.....	712
Mine and Smelter Co.: Preliminary Report of Massco Circuitron.....	1038
Mine mechanization.....	201
Mine structure, roof studies.....	1031
Minerals et Metaux: The Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate.....	1126
Mineral economics, raw materials survey.....	292
discussion.....	1169
Minerals, heavy, Brazil.....	685
Mines: metal, dust control.....	86
steel supports underground.....	475
use of concrete underground.....	355
Mining methods: caving of asbestos ore face stripping.....	1262
mechanical shaft sinking.....	355
mechanization.....	201
mechanized cutting.....	1260, 1262
sillimanite ore.....	788
Missouri School of Mines and Metallurgy: Measurement of Equilibrium Forces between an Attached Solid in Water.....	91
Moeller, J. E. and Norquist, D. E.: Relative Wear Rates of Various Diameter Grinding Balls in Production Mills.....	712
discussion.....	714



Molybdenite, treatment of sulphide flotation concentrates prior to separation from.....	506	Pebble mill, grinding with.....	387	ture of Lightweight Aggregates.....	479
Monseratt mine, tin deposit.....	1071	Pegmatite, Canadian deposits.....	242	Relative Wear Rates of Various Diameter Grinding Balls in Production Mills.....	712
Morgan, J. D., Jr., Sun, Shiou-Chuan, and Wesner, R. F.: Behavior of Mineral Particles in Electrostatic Separation.....	369 1152	Pembroke Chemical Corp.: Some Properties of Pseudowavellite from Florida.....	699	Research, cooperative, organizing and financing.....	374 275
Morris, T. M.: Measurement of Equilibrium Forces between an Air Bubble and an Attached Solid in Water.....	91 1154	Pennsylvania State College: Behavior of Mineral Particles in Electrostatic Separation.....	369	Resources: nonmetalliferous minerals of Arkansas.....	577
Moyd, Louis: Determination of the Coefficient of Linear Thermal Expansion of Rock Specimens by Means of Resistance Wire (SR-4) Strain Gauges.....	683 81 82	Cleaning of Fine Sizes of Bituminous Coals by Concentrating Tables.....	956	Review of Progress in the Caving of Asbestos Ore.....	467
Mucker, Riddell.....	81	Mechanism of Coals and Mineral Froth Flotations.....	616	Rexford, Elliot P.: Some Factors in the Selection and Testing of Concrete Aggregates for Large Structures.....	395 1168
Muckling, for shaft sinking.....	82	Some Factors Influencing the Performance of Single Retort Underfired Stokers.....	265	discussion.....	1168
Mufulira Copper Mines: Northern Rhodesia Mufulira Copper Mines, Ltd., Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Balls Mills.....	96	Perlite: furnace designs for expansion of.....	111 111 480	Rey, M.: Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate.....	1126
Mullins, R. A. and Merle, J. J.: Laboratory Control in Coal Washing and Drying Plants.....	609	types of rock.....	111	Discussion on Cyclone Thickener Applications in the Coal Industry.....	1158
Murdock, J. B. and Stein, H. A.: Comparative Furnace Designs for the Expansion of Perlite.....	111	Perlite Corp.: Comparative Furnace Designs for the Expansion of Perlite.....	111 1068 1070 1069	Discussion on Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite.....	1155
Murphy, R. James, W. F., Lang, A. H. and Kesten, S. N.: Canadian Deposits of Uranium and Thorium.....	239	Petalite—A New Commercial Mineral.....	1068	Rheolaveur System of Fine Coal Cleaning.....	1137
Meyers, J. F. and Lewis, F. M.: Progress Report on Grinding at Tennessee Copper Company.....	707, 1133 711, 1151	commercial applications.....	1070	Richards, Leslie C.: Studies of the Effect of Freight Rates on Marketing Northwest Industrial Minerals.....	285
<b>N</b>		thermal expansion of.....	1069	Richardson, A. C. and Lyons, Orville R.: Operating Data for a Bird Centrifuge.....	381 1166
National Coal Association: The Young Mining Engineer in the Coal Industry.....	718	Phelps Dodge Corp.: Effect of Conditioning on Flotation of Chalcocite.....	226	Ridland, G. Carman: Radioactivity at the Caribou Silver Mine, Boulder County, Colorado.....	98 872
National Lead Co.: Effects of Rod Mill Feed Size Reduction.....	1273	Phosphate: aluminum, description of.....	699	Rittinger theory.....	872
National Securities Resources Board: Behavior of Mineral Particles in Electrostatic Separation.....	369	Arkansas, deposits.....	583	Roberts, E. J.: The Probability Theory of Wet Ball Milling and Its Application.....	1267 871
Nepheline syenite, Arkansas deposits.....	583	chemical behavior under thermal treatment.....	700	Rock: crushing of.....	683
Neslage, O. J. and Jenkins, R. W.: A Classification and Application of Drill Jibs for Rock Drill Mounting.....	571	fertilizer value of.....	702	linear thermal expansion of.....	683
Netschert, B. C.: Discussion on Industrial Mineral Economics and the Raw Materials Survey.....	1169	plant.....	493	Rod mill: effects of feed size reduction grinding with.....	1273 388
New Mexico School of Mines: Magnetic Fields Associated with Igneous Pipes in the Central Ozarks.....	1143	rock industry, waste disposal.....	779	Roof Studies and Mine Structure Stress Analysis, Bureau of Mines Oil Shale Mine, Rifle, Colo.....	1031
New Theory of Communion.....	871	western industry.....	486	Rope, used for pillar wrapping.....	898
Newmont Mining Corp. of Canada: Canadian Deposits of Uranium and Thorium.....	239 494	Picher field, structure and mineralization used as guide to development.....	1251	Ruhr, mechanized cutting and face stripping.....	1260
Nitrate plants.....	494	Pillars, wrapping with old hoist rope.....	898	Runolinn, U. and Hukki, R. T.: Separation of Precious Metals from Anode Slimes by Flotation.....	1131
Nonmetalliferous Mineral Resources in Arkansas.....	577	Pilot plant: coal drying.....	976	Ruppert, John A. and Conley, John E.: Recent Developments in the Manufacture of Lightweight Aggregates.....	479
Norman, T. E.: Discussion on Relative Wear Rates of Various Diameter Grinding Balls in Production Mills.....	714	recovery of cobalt ore.....	1044	<b>S</b>	
Norquist, D. E. and Moeller, J. E.: Relative Wear Rates of Various Diameter Grinding Balls in Production Mills.....	712 714	tests.....	879	Safety: accident prevention, shaft sinking.....	85
Northern Rhodesia Mufulira Copper Mines Limited Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills.....	96	Pilot-Plant Investigation of Concentration of Blackbird Cobalt Ore by Roast-Flotation Process.....	1044	dust control.....	86
Notes on the Geology of the Potash Deposits of Germany, France, and Spain.....	117	Pitchblende: Canadian deposits.....	239	Pioche, Nev., silver mine.....	204
Novaculite, Arkansas deposits.....	580	discovery in silver veins.....	98	use of steel in underground support.....	475
<b>O</b>		Poland, J. F.: Ground Water in California.....	279	West Virginia mines.....	967
Office of Economic Warfare: An Improved Method of Gravity Concentration in the Fine-size Range.....	879	Pollution, air.....	971	Safety Problems Applicable in West Virginia Mines.....	967
Operating Data for a Bird Centrifuge.....	381	Potash: deposits, France.....	117	St. Joseph Lead Co. Development Work with Trackless Equipment.....	679
Ore: asbestos, caving of.....	467	Germany.....	117	Problems of a Mine Mechanization Program.....	201
drawing.....	470	Spain.....	117	Wrapping Pillars with Old Hoist Rope.....	898
recovery.....	471	Practical Dust Control in Metal Mines.....	86	St. Louis University: Magnetic Fields Associated with Igneous Pipes in the Central Ozarks.....	1143
Ore conveyor belts, electronic tramp iron detector.....	703	Prakash, B. and Schuhmann, R., Jr.: Effect of BaCl <sub>2</sub> and Other Activators on Soap Flotation of Quartz.....	591	Salley, D. J. and Simard, G. L.: Discussion on Concerning the Adsorption of Dodecylamine on Quartz.....	1154
Organizing and Financing Cooperative Research.....	374	Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite.....	601 1155	Salley, D. J., Simard, G. L. and Chupak, J.: Radiotracer Studies on the Interaction of Diethiophosphate with Galena.....	359 789 117
Otto, H. H. and McLaughlin, J. F.: Discussion on Laboratory Performance Tests of the Humphreys Spiral As a Cleaner of Fine Coal.....	1161	discussion.....	1155	Sandstone, Arkansas deposits.....	581
Oxidation Method for Investigating the Petrographic Composition of Some Coals.....	889	Preliminary Report of Massco Circutiron.....	1038	Santa Cruz Portland Cement Co.: Fuel Economy in the Lepol Kilns.....	289
<b>P</b>		Price, J. D. and Bertholf, W. M.: Discussion on Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1160	Schiffman, L. E.: Kerosine Flotation of Bituminous Coal Fines.....	1047 1159
Pannell, J. H., Gaudin, A. M., Dasher, J., Freyberger, W. L.: Use of an Induced Nuclear Reaction for the Concentration of Beryl.....	495	Probability Theory of Wet Ball Milling and Its Application.....	1267	School of Mines (Paris, France): The Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate.....	1126
Parry, V. F. and Wagner, E. O.: Drying Fine Coal in the Entrained and Fluidized State.....	974	Problems of a Mine Mechanization Program.....	201	Schuhmann, R., Jr. and Prakash, B.: Effect of BaCl <sub>2</sub> and Other Activators on Soap Flotation of Quartz.....	591
Parton, W. J.: Discussion on Kerosine Flotation of Bituminous Coal.....	1159	Production and Marketing of Garnet Abrasive Sands from Emerald Creek, Benewah County, Idaho.....	575	Effects of Activators and Alizarin Dyes on Soap Flotation of Cassiterite and Fluorite.....	601 1155
Patino Mines and Resources: An Improved Method of Gravity Concentration in the Fine-size range.....	879	Production of Ammonium Sulphate and Manganese Oxides.....	391	Schwellenbach, H. F. and Strohl, J. J.: Effects of Rod Mill Feed Size Reduction.....	23 790
<b>Q</b>		Production of Graded Glass Sand by Grinding and Classification.....	385	Screening: cyclone separator.....	1273
Quantitative Efficiency of Separation of Coal Cleaning Equipment.....	256	Progress Report on Grinding at Tennessee Copper Company.....	707, 1133	Scrubber, attrition.....	1141
Quarrying: building stone.....	884	Pseudowavellite, Florida, properties of.....	699	Seismic investigations.....	1032
Quartz: adsorption of dodecylamine on.....	499	Pumps, sand, automatic controls.....	801	Separation: Bird centrifuge.....	381
discussion.....	1154	Putnam, G. L. and Flink, C. G.: The Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions.....	952 1075	concentrating tables.....	957
calcium activation of.....	597	<b>R</b>		electrostatic.....	369, 1041
crystals, Arkansas deposits.....	582	Radioactivity at the Caribou Silver Mine, Colorado.....	98	heavy-med units.....	1140
effect of alizarin dyes on flotation of "giant quartz veins," Canada.....	606	Radioactive minerals, Canadian deposits of.....	239	quantitative efficiency of coal cleaning equipment.....	365
soap flotation of.....	591	Caribou silver mine.....	98	rod mill.....	256 1273
<b>R</b>		dithiophosphate.....	359	Separation of Precious Metals from Anode Slimes by Flotation.....	1131
Radiotracer Studies on the Interaction of Dithiophosphate with Galena.....	359	Radioactivity: Geiger-Muller Survey, Caribou Silver Mine.....	98	Separator, cyclone, Dutch State Mines.....	790
correction.....	789	Radiotracer Studies on the Interaction of Dithiophosphate with Galena.....	359	Shafts: sinking Star shaft.....	81
Ralston, O. C. and Raas, F.: Conductance Separation with Convective Charging.....	1041	Ravitz, S. F. and Zimmerley, S. R.: Pilot-Plant Investigation of Concentration of Blackbird Cobalt Ore by Roast-Flotation Process.....	1044	Shale, Arkansas resources.....	580
Discussion on Behavior of Mineral Particles in Electrostatic Separation.....	1152	Raw materials survey.....	292	<b>S</b>	
Recent Developments in the Manufacture of Lightweight Aggregates.....	479	Recent Developments in the Manufacture of Lightweight Aggregates.....	479	<b>S</b>	



Sheffield Steel Corp.: Relative Wear Rates of Various Diameter Grinding Balls in Production Mills.....	712	Sun, Shiou-Chuan and Zimmerman, R. E.: The Mechanism of Coarse Coal and Mineral Froth Flotations.....	616	University of Florida: Effect of Waste Disposal of the Pebble Phosphate Rock Industry in Florida on Condition of Receiving Streams.....	779
Sherman, Gerald: Review of Progress in the Caving of Asbestos Ore.....	467	Sunnyhill Coal Co.: The Colmar-A Continuous Mining Machine.....	715	University of Michigan: Tin Deposit of Monserrat Mine, Bolivia.....	1071
Shotts, R. Q.: An Oxidation Method for Investigating the Petrographic Composition of Some Coals.....	889	<b>T</b>		University of Washington: Action of Sulphide Ion and of Metal Salts on the Dissolution of Gold in Cyanide Solutions.....	952
Shugert, J. L. and Williamson, W. C.: Practical Dust Control in Metal Mines.....	86	Tabling, theory of.....	957	Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1057
Shuttle car, Joy.....	680	Tait, W. J., Craig, A. E. and McCurdy, E. P.: Preliminary Report of Masco Circuitron.....	1038	Unusual Test of the Accuracy of Well-surveying Methods.....	1037
Silica, Arkansas resources.....	580	discussion.....	1155	Uranium, Canadian deposits.....	239
Sillimanite in the Southeast.....	785	Talcs, California, descriptions of mines; Silver Lake mine.....	123	Use of an Induced Nuclear Reaction for the Concentration of Beryl.....	495
Silver, deposits at Caribou Hill, Colo.....	98	Western mine.....	125	Use of Concrete Underground.....	355
desorption from carbon.....	217	Talc City mine.....	127	Use of Spiral Classifiers as Ball Mill Feeders.....	951
separation from anode slimes.....	1131	Talc City Mine; talc.....	127	<b>V</b>	
Silver Lake Mine; talc.....	123	Tartarou, F. X.: Discussion on Measurement of Equilibrium Forces Between an Air Bubble and an Attached Solid in Water.....	1154	Ventilation, at Thetford asbestos ore mine.....	472
Simard, G. L., Chupak, J. and Salley, D. J.: Radiotracer Studies on the Interaction of Dithiophosphate with Galena.....	359	Teague, K. H.: Sillimanite in the Southeast.....	785	Vermiculite, use as aggregate.....	480
correction.....	789	Teallite, Monserrat Mine, Bolivia.....	1071	<b>W</b>	
Simard, G. L. and Salley, D. J.: Discussion on Concerning the Adsorption of Dodecylamine on Quartz.....	1154	Teichman, H. L. and Sippelle, E. M.: Roof Studies and Mine Structure Stress Analysis, Bureau of Mines Oil-Shale Mine, Rifle, Colo.....	1031	Wagner, N. S. and Mason, R. S.: Lightweight Aggregate Industry in Oregon, discussion.....	1170
Sinking Star Shaft at Vanadium, New Mexico.....	81	Tennessee, building stone.....	883	Wagner, E. O. and Parry, V. F.: Drying Fine Coal in the Entrained and Fluidized State.....	974
Sintering; aggregates.....	483	Tennessee Copper Co.: Progress Report on Grinding at Tennessee Copper Co.....	707	Wang, Jen-Tung and Bond, F. C.: A New Theory of Commminution.....	871
effect of cooling rate.....	1281	Tennessee Valley Authority, Building Stone of the Crab Orchard District, Tenn.....	883	Walker, G. B.: Discussion on Some Factors in Selection and Testing of Concrete Aggregate for Large Structures.....	1168
laboratory equipment.....	1275	Sillimanite in the Southeast.....	785	Washing; coal: cyclone thickeners.....	102
prepared agglomerates.....	1280	Thetford Mines, asbestos ore.....	468	washing tables.....	609
Sippelle, E. M. and Teichman, H. L.: Roof Studies and Mine Structure Stress Analysis, Bureau of Mines Oil-Shale Mine, Rifle, Colo.....	1031	Thickening; cyclone.....	102	preparing coal for synthetic fuel industry.....	507
Slags, expanded.....	481	Thorium, Canadian deposits.....	254	Waste disposal, phosphate rock industry.....	779
Slate, Arkansas resources.....	580	Thunae, A. and Spedden, H. R.: An Improved Method of Gravity Concentration in the Fine-size Range.....	879	Weiss, Norman: Discussion on An Electronic Trap Iron Detector for Ore Conveyor Belts.....	706
Sloss-Sheffield Steel and Iron Co.: Kerosine Flotation of Bituminous Coal Fines.....	1047	discussion.....	1156	Well-surveying methods.....	1037
Smith, J. P.: Notes on the Geology of the Potash Deposits of Germany, France, and Spain.....	117	Timbering; for shaft sinking.....	82	Wesner, R. F., Sun, Shiou-Chuan, and Morgan, J. D., Jr.: Behavior of Mineral Particles in Electrostatic Separation.....	369
Snyder, C. H.: The Colmar-A Continuous Mining Machine.....	715	Tin: activation.....	599	discussion.....	1152
Some Applications of Millisecond Delay Electric Blasting Caps in Mining.....	1123	recovery of.....	879	West Virginia Department of Mines: Specific Safety Problems Applicable in West Virginia Mines.....	967
Some Factors in the Selection and Testing of Concrete Aggregates for Large Structures.....	395	Tin Deposit of the Monserrat Mine, Bolivia.....	1071	West Virginia, mine safety problems.....	967
Some Factors Influencing the Performance of Single Retort Underfeed Stokers.....	265	Tricone, mill.....	709	Western Mine, talc.....	125
Some Properties of Pseudowavellite from Florida.....	699	operation of.....	1133	Westman, B. J.: Discussion on Diamond Drilling Quartz-Feldspar Inter-growths.....	1148
Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries.....	790	Tripoli, Arkansas deposits.....	581	White, Jack: Northern Rhodesia Mufuira Copper Mines Limited Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills.....	96
Southwest Research Institute: Non-metaliferous Mineral Resources in Arkansas.....	577	Tri-State District, development.....	1251	discussion.....	1151
Spain: potash deposits.....	120	Turkey, mineral occurrences.....	108	Williamson, W. C. and Shugert, J. L.: Practical Dust Control in Metal Mines.....	86
Specht, R. C.: Effect of Waste Disposal of the Pebble Phosphate Rock Industry in Florida on Condition of Receiving Streams.....	779	Turmeaure, F. S. and Gibson, R.: Tin Deposit of the Monserrat Mine, Bolivia.....	1071	Williston, S. H.: An Unusual Test of the Accuracy of Well-surveying Methods.....	1037
Special Methods for the Beneficiation of Glass Sand.....	1139	Tuwiner, S. B. and Korman, S.: Effect of Conditioning on Flotation of Chalcocite.....	226	Winters, V. W., Crabtree, E. H., Jr., Chapman, T. G.: Developments in the Application of Activated Carbon to Cyanidation Including the Desorption of Gold and Silver from Carbons.....	217
Specific Safety Problems Applicable in West Virginia Mines.....	967	Tyler, P.: Kaolin Production and Treatment in the South.....	694	Wire, resistance, strain gauges.....	683
Spedden, H. R. and Thunae, A.: An Improved Method of Gravity Concentration in the Fine-size Range.....	879	Special Methods of the Beneficiation of Glass Sand.....	1139	Woolf, W. G. and Bethune, A. Y.: The Burt Filter.....	585
discussion.....	1156	<b>U</b>		Wrapping Pillars with Old Hoist Rope.....	898
Sphalerite, Monserrat Mine.....	1074	U. S. Bureau of Mines: Air Pollution by Industrial Fumes, Gases, and Dusts.....	971	Wright, C. C., Baumann, H. S., and Spicer, T. S.: Some Factors Influencing the Performance of Single Retort Underfeed Stokers.....	265
Spicer, T. S., Baumann, H. S. and Wright, C. C.: Some Factors Influencing the Performance of Single Retort Underfeed Stokers.....	265	Coal Preparation for Synthetic Liquid Fuels.....	507	Wright, Lauren A.: California Talcs.....	122
Spiral, Humphreys, fine coal cleaning.....	1057	Conductance Electrostatic Separation with Convective Charging.....	1041	Wurtzite, Monserrat Mine.....	1074
discussion.....	1160	Drying Fine Coal in the Entrained and Fluidized state.....	974	Wykoff, B. T.: Wrapping Pillars with Old Hoist Rope.....	898
Stannite, Monserrat Mine.....	1076	Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1057	<b>Y</b>	
State Geological Survey Division: Research in Coal Geology.....	275	Pilot-plant Investigation of Concentration of Blackbird Cobalt Ore by Roast-Flotation Process.....	1044	Yancey, H. F. and Geer, M. R.: Discussion on Quantitative Efficiency of Separation of Coal Cleaning Equipment.....	1163
State Research Institute (Helsinki, Finland): Separation of Precious Metals from Anode Slimes by Flotation.....	1131	Production of Graded Glass Sand by Grinding and Classification.....	385	Yancey, H. F., Geer, M. R., Allyn, C. L. and Eckhouse, R. H.: Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal.....	1057
Steel: for underground support.....	475	Recent Developments in the Manufacture of Lightweight Aggregates.....	479	discussion.....	1160
Stein, H. A. and Murdock, J. B.: Comparative Furnace Designs for the Expansion of Perlite.....	111	Roof Studies and Mine Structure Stress Analysis, Bureau of Mines Oil-Shale Mine, Rifle, Colo.....	1031	Young Mining Engineer in the Coal Industry.....	718
Stephenson, W. B.: Automatic Controls on Sand Pumps.....	801	U. S. Coal and Coke Co.: Mechanized Cutting and Face Stripping in the Ruhr.....	1260	<b>Z</b>	
Stoker, single retort.....	265	U. S. Corps of Engineers: Determination of the Coefficient of Linear Thermal Expansion of Rock Specimens by Means of Resistance Wire (SR-4) Strain Gauges.....	683	Zimmerley, S. R. and Ravitz, S. F.: Pilot-Plant Investigation of Concentration of Blackbird Cobalt Ore by Roast-Flotation Process.....	1044
Stokes, C. A. and Ladoo, Raymond B.: Industrial Mineral Economics and the Raw Materials Survey.....	292	Some Factors in the Selection and Testing of Concrete Aggregates for Large Structures.....	395	Zimmerman, R. E.: The Cleaning of Fine Sizes of Bituminous Coals by Concentrating Tables.....	956
discussion.....	1169	U. S. Department of Agriculture: Some Properties of Pseudowavellite from Florida.....	699	Zimmerman, R. E. and Sun, Shiou-Chuan: The Mechanism of Coarse Coal and Mineral Froth Flotations.....	618
Stone, Tennessee building.....	883	U. S. Department of the Interior: Economic Factors in the Western Phosphate Industry.....	486	Zinc sulphate, filtration of.....	585
Strohl, J. J. and Schwellenbach, H. J.: Effects of Rod Mill Feed Size Reduction.....	1273	Ground Water in California.....	279		
Studies of the Effect of Freight Rates on Marketing Northwest Industrial Minerals.....	285	U. S. Military Government: Mechanized Cutting and Face Stripping in the Ruhr.....	1260		
Sullivan Mining Co.: The Burt Filter.....	585	U. S. Potash Co.: Notes on the Geology of Potash Deposits of Germany, France, and Spain.....	117		
Sulphate plant.....	494	U. S. War Department: Coal Preparation for Synthetic Liquid Fuels.....	507		
Sulphide: effect on conditioning of chalcocite.....	232	University of Alabama: An Oxidation Method for Investigating the Petrographic Composition of Some Coals.....	889		
electrochemistry of.....	234	University of Arizona: Developments in the Application of Activated Carbon to Cyanidation Including the Desorption of Gold and Silver from Carbons.....	217		
oxidation of.....	237				
flotation concentrates, treatment prior to separation of molybdenite.....	506				
Sulphur dioxide, recovery.....	492				
Sun, Shiou-Chuan, Morgan, J. D., Jr., Wesner, R. F.: Behavior of Mineral Particles in Electrostatic Separation.....	369				
discussion.....	1152				



